LORNEX MINING CORPORATION LTD.

VANCOUVER, BRITISH COLUMBIA

GEOLOGY AND EXPLORATION

LORNEX PROJECT REPORT NUMBER ONE

Prepared by

RIO ALGOM MINES LIMITED

Toronto, Ontario

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THIS REPORT IS ONE OF TEN DOCUMENTS PREPARED TO RECORD, SUMMARIZE, AND EVALUATE ALL OF THE TECHNICAL, COMMERCIAL AND ECONOMIC DATA DEVELOPED IN THE PHASE 11 PROGRAMME OF THE LORNEX PROJECT. A FULL LIST OF THESE REPORTS IS GIVEN BELOW:

1 GEOLOGY AND EXPLORATION
2 METALLURGY
3 PLANT AND FACILITIES DESIGN
4 MINING AND PRODUCTION PLANS
5 TOWNSITE AND CAMP ACCOMMODATION
6 OPERATING, ORGANIZATION AND COSTS
7 TAXATION
8 TRANSPORTATION AND MARKETING
9 WORKING CAPITAL, FINANCING AND SUNDRY EXPENSES
10 ECONOMIC EVALUATION

TABLE OF CONTENTS

REPORT NUMBER ONE
GEOLOGY AND EXPLORATION

By

J. A. Sadler, President
E. L. Evans, Chief Geologist
H. W. Marsh, Geologist
C. D. Spence, Geologist
C. A. Devine, Geologist

LIST OF TABLES AND FIGURES

LIST OF DRAWINGS

LORNEX MINING CORPORATION LTD.

REPORT NUMBER ONE

GEOLOGY AND EXPLORATION

SECTION 1 - INTRODUCTION

The Lornex copper and molybdenum deposit lies in Southern British Columbia within the Guichon Batholith on the south side of the Highland Valley and almost directly across the valley from the open pit mine of Bethlehem Copper Corporation Limited (Dwg. No. 101). Copper mineralization was discovered on the Lornex property in 1964 by Mr. Egil Lorntzsen who investigated it by bulldozing long trenches which demonstrated that the mineralization had an appreciable lateral extent. In the Spring of 1965, management of the property was acquired by Rio Tinto Canadian Exploration Limited as a result of an agreement between Lornex Mining Corporation Ltd. and Rio Algom Mines Limited. Since then the property has been under continuous investigation by percussion drilling, rotary drilling, diamond core drilling, geological, geophysical and geochemical surveys and underground development and drilling. As a result of this work, a large low-grade copper-molybdenum deposit has been delimited a short distance north of the original discovery. Tonnage and grade have been estimated from the results of surface diamond drilling.

An underground programme was started in January 1967 for the purpose of:

(a) providing further geological information as most of the deposit is covered by overburden;

(b) confirming grade indicated by surface drilling through bulk sampling;

(c) supplying material for pilot plant testing.

With the completion of this underground work, evaluation of the mineral deposit is now possible. This report, therefore, is designed to summarize the work done, to record the data resulting from the work, to give a geological appraisal of the results, and an evaluation of the mineral deposits.

1.1 HISTORY

Management of the property was taken over by Rio Tinto Canadian Exploration Limited in the Spring of 1965 by which time bulldozer trenching and percussion drilling were already in progress and this programme was continued by them throughout 1965 and 1966. A truck-mounted Copco overburden drill was used to drill 511 vertical holes for a total footage of approximately 91,000 feet. Of this percussion programme, 192 holes totalling 45,845 feet were drilled on the main mineralized zone and the remainder to the north and south in other areas of interest.

A programme of bulldozer trenching was carried out concurrently with the percussion drilling stripping overburden to bedrock where possible. In excess of 18,000 linear feet of trenching was accomplished over parts of the property in 1965 and 1966 and about 5,000 feet of this work was done on the main mineralized zone.

Geological mapping, magnetometer, induced polarization and reconnaissance geochemical surveys were completed during 1965 together with the first diamond drilling consisting of seven holes totalling 3,123 feet. This drilling was done to the southeast of the main zone of mineralization.

Diamond drilling was resumed to the north of the initial drilling in December 1965 within an area where an induced polarization survey had outlined a larger area of mineralization known as the North Zone in which a higher grade was indicated by percussion drilling and trenching. The planned programme consisted of 21 holes on four lines 800 feet apart with holes spaced at 300 to 400-foot intervals on the lines. Sufficient encouragement was obtained from this programme to continue drilling on intermediate and additional lines to January 1968 by which time a further 47 holes totalling 49,824 feet had been completed. An additional 11 holes totalling 7,778 feet were drilled on adjoining areas to the north and south for various reasons. Total diamond drilling amounted to 86,017 feet in 87 holes, of which 66 holes totalling 74,044 feet were drilled in the main or North Zone.

Collaring of a shaft prior to sinking and underground development was begun in January 1967 and sinking of the shaft started in March. A sampling plant, pilot mill and complete assay laboratory were constructed and operating by April 1967. By September 22nd, the shaft was completed to a depth of 550 feet and crosscutting started. In all, approximately 2,644 feet of lateral development was completed including 1,674 feet of crosscutting and

970 feet of drifting north and south to provide stations for diamond drilling. Underground diamond drilling consisted of 10 holes totalling 5,300 feet.

On surface, a Becker rotary drill programme consisting of 19 vertical holes totalling 4,211 feet of drilling was carried out in January and February 1968 to test the oxide layer at the top of the mineralized zone. The results, compiled with the results from previous percussion drilling and a seismic survey completed in 1967, helped in the definition of the oxide zone in greater detail. The investigation of the deposit, which has taken about three and a half years, was completed in July 1968 to a point where an evaluation could be made.

1.2 PROPERTY AND AGREEMENTS

The Lornex Mining Corporation Ltd. has acquired or has under option 204 mineral claims in the Highland Valley area of the Kamloops Mining Division in the Province of British Columbia (Dwg. No. 102). Mining properties have been acquired for the issue of 875,000 shares of capital stock and $53,500 cash.

Prior to May 1st, 1965, the Company had acquired 94 mineral claims for the issue of 750,000 shares and 50,000 shares had been issued in exchange for a royalty under an option agreement with Mr. Jack Crighton. The original purchase option with Crighton was dated July 22nd, 1964 and was exercised by the Company to acquire mineral claims.

By an agreement dated May 1, 1965 and amended July 20, 1965, between Lornex Mining Corporation Ltd., Rio Algom Mines

Limited, Mr. Egil H. Lorntzsen and Mr. A. David Ross (the latter two being shareholders of Lornex holding in excess of 50% of the issued stock at that date) Rio was granted an option to purchase up to 2,400,000 free Treasury Shares at prices ranging from $0.75 to $2.50 per share over a period from the date of the agreement to September 1, 1968. The options have all been exercised and the money paid for these shares has been used for development of the mineral claims held by Lornex.

Under a separate exploration agreement, Yukon Consolidated Gold Corporation Limited was allowed to subscribe for 40% of the Lornex shares issued to Rio Algom pursuant to the above-mentioned options. The 2,725,948 shares held by Rio Algom and Yukon at the present time represent the optioned shares together with shares acquired under a rights issue and represent 60.3% of the issued capital stock. Rio Algom and Yukon by a voting agreement maintain voting control of Lornex and provide continuity of management by Rio Algom until at least December 31, 1972.

If production is undertaken on certain claims purchased by Lornex from Skeena Silver Mines Limited (Dwg. No. 103) under an agreement dated May 17, 1965, Lornex is required to pay royalties of 5% of net smelter returns up to a maximum of $500,000 with a minimum annual payment of $15,000 to commence on May 31, 1971.

Lornex also has the right to purchase certain mineral claims under an agreement with Skeena Silver Mines Limited, dated May 31, 1965 (amended April 30, 1968), such right being obtained by an assignment dated August 12, 1965 from Rio Algom. The purchase option is in good standing and Lornex is required to fulfill work commitments of $300,000 by May 31, 1970. The terms of this agreement further provide for payment of 5% net smelter returns up to $3,000,000. In the event that Lornex does not bring these claims into production prior to May 31, 1972 it shall pay the sum of $2,000 per month until the claims are brought into production or until May 31, 1974 whichever shall be the earlier.

Under an agreement dated July 1, 1967 with Noranda Exploration Company Limited (N.P.L.) Lornex has the right to use the area of ten mineral claims for dumping of waste materials. Certain work commitments are also encompassed in the agreement. Lornex now has a 15% interest in the mineral rights of these claims.

A list of mineral claims owned or held under option by Lornex is shown in the Appendix 'A' and on an accompanying map (Dwg. No. 102).

SECTION II - SUMMARY

The Lornex porphyry copper deposit, located in the Highland Valley of British Columbia, has been investigated over a period of 3 1/2 years by percussion drilling, diamond drilling and underground exploration. The accumulated data have been compiled and assessed to give a drill-indicated in situ reserve within the limits of a designated open pit of 274,000,000 tons of ore averaging 0.447% Cu. and 0.0147% Mo., with a cut-off grade of 0.26% Cu. and a mill feed grade reserve of 292,800,000 tons averaging 0.427% Cu. and 0.0140% Mo.

Only the surface diamond drill core results were used in this instance to obtain the above figures. The drilling in the pit area amounted to 66 holes totalling 74,044 feet and the holes were spaced at about 350-foot intervals on lines 400 feet apart. With a few exceptions all the holes were drilled to the west at -60° to cross-cut the structure.

Copper trends were plotted from core assays to conform with known structure and these trends were used as the basis for calculation. The trends were plotted first on sections then projected to plans at 80-foot intervals within the confines of the pit. Each block used for the calculation of an in situ reserve was assigned a grade from the nearest of intersecting drill holes and was bounded by trend line margins half the distance to adjacent sections and 40 feet up and down in a vertical direction. A factor of 12.4 cubic feet per ton was used to convert volume to tons.

Two independent reserve calculations were made on the material within the confines of the pit to determine the mill feed reserve. The first was set up as a computer programme in which the deposit was divided into 80-foot cubes with each cube being assigned a grade from the trend lines which contain it. The second was similar to the first except that it was a manual calculation from the cubes. The cube method introduces a smoothing effect and hence a dilution and is believed to represent possible mill head grade as contrasted to the in situ grade for the initial calculation. The three calculations, i.e. in situ calculation, dealt with in this report and the two independent calculations, agree within acceptable limits.

The underground programme was designed to:

(1) Provide material for the pilot mill.

(2) Check the reliability of the copper and molybdenum distribution as interpreted in the grade trend plans.

(3) Examine the nature of the mineralization.

(4) Give a measure of the accuracy of the assays in the surface diamond drill holes.

The programme consisted of a centrally located cross-cut driven from east to west, drifts to the north and south from the cross-cut and underground diamond drilling. It enabled a comparison of grades to be made between those produced by;

(a) underground diamond drilling and bulk sampling in the cross-cut; (b) bulk sampling and surface diamond drilling, and (c) underground diamond drilling and surface diamond drilling.

Using this information it was possible to establish that the distribution of the copper and molybdenum as interpreted on the grade trend plans from the surface diamond drilling is essentially correct although allowance must be made for wandering of surface drill holes and the fact that the level is at a slightly different elevation from that used in plotting. Further, it established that the grade interpreted from the surface diamond drill holes possibly errs on the conservative side and, therefore, can be used with confidence for a feasibility study. In this regard the underground data when compared to the surface data in the same area gives a copper grade about 9% higher than that from the surface drilling.

Inasmuch as the accuracy of the assaying methods is vital to the calculation of grade of the deposit a system of check assaying between Coast Eldridge (an independent laboratory) the Lornex laboratory and the Elliot Lake laboratory, was established for all samples from underground. The system established the reliability of the Lornex and Elliot Lake laboratories whose assays were used in all calculations.

Thorough checks were made of sampling methods for diamond drill core and bulk sampling.

Studies made on surface drill hole assays by computer have established the following confidence for the in situ reserve: ± .009% Cu. for a 97.5% confidence limit.

SUMMARY OF RESERVE ESTIMATE

Two methods of calculation give the following estimates of the reserves within Pit 15:

1. Calculated from grade trend blocks.

IN SITU RESERVE		TONS	% Cu.	% Mo.
A.	Ore 0.30% Cu and greater	251,100,000	.462	.0152
B.	Ore 0.29% - 0.26% Cu	22,900,000	.271	.0091
TOTAL A & B				
	0.26% Cu and greater	274,000,000	.447	.0147
C.	Sub-ore (Stockpile) 0.25% - 0.20% Cu	38,700,000	.226	.0066
D.	Internal Waste 0.19% Cu and less	46,200,000	.135	.0040

2. Calculated by 80-foot cubes.

MILL FEED RESERVE				
A.	Ore 0.26% Cu and greater	292,800,000	.427	.0140
B.	Waste (including oxide)	252,500,000	No Grades	

Waste To Ore Ratio - 0.862:1

SECTION III - GEOLOGY

3.1 GENERAL

The Guichon Batholith is a complex intrusive, of Lower Jurrassic age, having a length of about 40 miles in a north-south direction and a width of about 16 miles. It forms part of the interior plateau of British Columbia (Dwg. No. 101) which, in the vicinity of the Lornex deposit, is gently rolling with peaks reaching about 6,000 feet elevation and the valley floors about 4,000 feet. The area is generally heavily wooded with second growth lodge pole pine, some spruce and poplar. The upland sections contain many small lakes, ponds and creeks.

The northern part of the batholith is covered by Tertiary basalt lava and outliers of this basalt show that it was once more extensive. Although the batholith has been mapped for many years (Cockfield 1948; Duffell and McTaggart 1942)* detailed studies have been made only on the north side of the Highland Valley in the vicinity of the Bethlehem Mine (White, Thompson and McTaggart 1956; Carr 1960). These reveal a complex of intrusives not recognized elsewhere in the batholith. At Bethlehem the oldest rock is the Guichon quartz diorite which is intruded by a younger quartz diorite known as the Bethlehem quartz diorite which in turn is intruded by granite and porphyry dykes. Pipe and dyke-like masses of breccia are associated with copper mineralization on the Bethlehem property and on the Trojan property which adjoins to the north (White, Thompson and McTaggart, 1956)

* References appear at the end of this report

On the south side of the Highland Valley the dyke rocks though present, are less abundant and the two main rock types distinguished are Skeena quartz diorite, (which may be equivalent to the Bethlehem quartz diorite) and the Bethsaida granodiorite. No breccia has been observed.

3.2 GEOLOGY OF THE DEPOSIT

The Lornex mineral deposit is contained in a roughly elliptical area about 4,000 feet long and 1,600 feet wide at its widest point and its long axis trends north northwest. (Dwg. No. 103). It is a porphyry copper type of deposit in that the host rock for the mineralization is an acid intrusive in which the sulphide minerals are more or less uniformly distributed through the rock in fractures and faults and in isolated grains disseminated among the rock minerals.

The maximum height above sea level on the Lornex property is about 5,500 feet. This is attained in the southern part of the claim group and from there the ground slopes gently northward to the Highland Valley which has an elevation of about 4,000 feet. The mineral deposit lying beneath this gently northward slope has an elevation at its south end of 5,150 feet dropping to 4,700 feet at the north end.

Award Creek which flows northward through the property has carved a broad valley in the overburden to a maximum depth of about 90 feet over the north-western part of the mineral deposit. The overburden consists of rudely sorted glacial till which has a maximum thickness of 250 feet beneath the Award Creek valley and a minimum thickness of about 5 feet over the south and southeast parts of the mineral deposit. Total volume of the till over the proposed open pit area is 50,978,000 cubic yards.

The glacial till has been partly water sorted for within it there are horizons of fine sand and clay and, near surface, an horizon containing numerous boulders up to 2 feet in diameter. Boulders of unaltered quartz diorite and granodiorite up to 6 feet in diameter occur in places just above bedrock and may form part of a continuous horizon. Over the northern end of the mineral deposit the till, near bedrock, contains appreciable water.

The Skeena quartz diorite and the Bethsaida granodiorite feature largely in the Lornex mineral deposit as it lies mainly in the Skeena at its contact with Bethsaida (Dwg. No. 103). The contact trends north-northwest across the property with an essentially vertical dip. Within the mineralized area it swings sharply west to a point where it intersects a major north striking fault called the Award Creek fault. This fault which dips steeply west forms the contact between the two rocks for an undetermined distance to the north.

Sulphide mineralization occurs in both Skeena and Bethsaida although it dies out in the Bethsaida a short distance west of the contact. Access for the mineralization appears to have been gained through the contact fault and through faults adjacent to it in the Skeena quartz diorite. Eight major faults have been noted in the mineralized zones and the rock between them is highly fractured and contains many minor faults.

3.2.1 Rock Description

The Skeena quartz diorite is a medium to coarse grained (1-10 mm) equigranular rock with visible quartz, interstitial to the plagioclase feldspar. Hornblende is the chief ferromagnesian mineral but in places biotite occurs with or without hornblende. The distinguishing features of this rock are the interstitial quartz and the presence of a moderate amount of ferromagnesian minerals.

The Bethsaida granodiorite is typically light coloured, medium to coarse grained (1-10 mm) and equigranular with large quartz aggregates scattered among the feldspar. Ferromagnesian minerals are a minor constituent and biotite is the main one so that the rock is distinguished by its light colour and abundance of large quartz grains. It appears to be younger than the Skeena quartz diorite.

Light coloured, fine-grained, quartz porphyry with small quartz "eyes", intrudes the Skeena near the southern end of the main deposit (Dwg. No. 103). It strikes roughly northwest and its western contact appears to dip west about 75° whereas its eastern contact dips east at 60°. It is well mineralized with sulphides where it extends into the mineral zone.

Dyke rocks, although present, are not nearly so common as at Bethlehem and do not form an important feature of the deposit. A fine grained granite dyke striking northwest and dipping from 15° to 30° northeast occurs in the northeastern part of the deposit. Small, intermittent, basic to intermediate, fine grained dykes occur in the southwestern part of the deposit. Other acid dykes are present but are somewhat obscured by the intense alteration of the rock.

3.2.2 Structure and Alteration

Structure and alteration in the mineralized zone are intimately related as the latter occurs in and adjacent to zones of extensive faulting. In part, faulting may have been controlled by the contact between the Skeena quartz diorite and the Bethsaida granodiorite as the contact along the western part of the mineral deposit is a faulted contact dipping steeply west. To the south the contact, not faulted, strikes approximately N 10° W and dips steeply west.

Eight major fault zones have been observed in the underground development and many others may well exist. These fault zones, estimated to contain about 20% fault gouge, are from 50 to 100 feet wide and consist of fault breccia and interweaving gouge-filled faults. The most easterly of these fault zones has a central zone of about fifteen feet of gouge. Six of the fault zones are believed to strike almost due north-south and dip from 55 to 65 degrees to the east. The most westerly fault has a similar strike but apparently dips steeply to the west. One fault zone with a strike of N 65° W and a dip to the south has been observed. Due to the abundance and extent of faulting, a clear understanding of the pattern will probably not be obtained until the zone has been stripped and more completely exposed.

More than 180 faults ranging in width from one to six inches have been observed in 1400 feet of underground lateral workings. The distribution of the faults shown on a polar net (Fig. 1) suggests a wide variation in strikes and dips but four general sets of faults can be recognized. The first set which occurs mainly in the eastern half of the deposit strikes from N 20° E to N 15° W and dips from 45° to 85° east. A second set strikes east-west and dips from 45° to 80° south. A third set strikes from N 35° W to N 80° W and dips from 45° to 85° southwest and a fourth set strikes north-south and dips from 50° to 85° west. These latter two sets of faults occur mainly in the western half of the deposit. All the faults show post-sulphide mineralization movement.

Alteration of the rocks in the deposit is related to the faulting (Dwg. No. 123). All of the rocks have been altered to some extent - beginning with the conversion of hornblende to biotite and later the breakdown of the biotite to sericite and other minerals. Because the predominant mineral is feldspar, its breakdown has resulted in the development of abundant sericite. In the fresh samples, incipient development of sericite is quite evident but the feldspar is readily identifiable as oligoclase by its twinning. With more advanced alteration sericite is abundant and the twinning though somewhat obscured is still visible. In the most advanced alteration the outline of the feldspar crystals can be detected but the twinning is completely obscured. The breakdown of the ferromagnesian minerals results in the development of quartz and chlorite and in some cases siderite and sericite.

The most intense alteration appears to be related to sulphide mineralization as samples containing sulphides are altered to a degree related to the sulphide content.

Faulting has resulted in the formation of gouge zones containing sericite, chlorite and calcite in zones up to 30 feet wide. Other minerals observed include siderite, secondary biotite, gypsum, pyrite, epidote, hematite and the ore sulphides. Siderite and hematite are probably the result of alteration of magnetite which occurs as a minor constituent in fresh quartz diorite.

Degrees of alteration can, in most cases, be correlated directly with the copper grade. The weakly altered rock about the perimeter of the deposit and, in some cases, between the major fault zones, is of lower copper grade whereas high grade zones or bands are intensely altered.

Even weakly altered rock, however, is well fractured and sulphide mineralization occurs along these fractures. The fractures appear to be shear fractures in some cases although some may be tension cracks and these too occur in fairly well defined sets. In the eastern half of the deposit two sets are apparent. One strikes north-south and the other N 20° E and dips steeply east. In the western half of the deposit at least three sets are apparent. One strikes about N 20° E and dips steeply east; a second strikes about N 45° E and dips southeast and a third strikes east-west and dips to the south.

Mineralization appears to be related primarily to the north-south striking major fault zones. However, one of the better defined and widest zones of above average copper grade lies along the east side of a quartz porphyry which intrudes the Skeena quartz diorite near the south end of the deposit.

A similar and more strongly developed high grade zone lies in a parallel position to the north and east of the block of Bethsaida granodiorite which occurs in the southwest corner of the deposit on Lines 5S, 1S, 3N and 7N on the plan of the 4552 level. (Dwg. No. 124).

A probable relationship of the two higher grade zones to these intrusions into the Skeena quartz diorite is apparent and is interpreted as reflecting zones of dilatancy developed by their buttress effect during the period of subsequent faulting and shearing to which the sulphide mineralization is related. In this context it is noted also that at these points the prominent trend of the structural features deviates from a north northwest strike to one more northerly.

Some of the faults and fractures are occupied by quartz along with sulphides. Post-ore movement along the strike faults has crushed the mineralized quartz veins and movement along the cross faults has produced minor displacement of the veins. The Award Creek fault along the present Bethsaida-Skeena contact has apparently displaced a portion of the deposit lying to the west of the fault. The faulted segment has not been located but it is believed that the displacement of the west portion of the mineralized zone was to the north.

3.2.3 Mineralization

The principal ore minerals in the unweathered zone of the deposit are: chalcopyrite, bornite, chalcocite, covellite and molybdenite. The gangue minerals include: quartz, feldspar hornblende, biotite, sericite, calcite, siderite, chlorite, pyrite hematite and magnetite.

It is estimated that 35% of the contained copper occurs in the form of bornite and the balance as chalcopyrite with only minor amounts in chalcocite and covellite. Pyrite, magnetite and hematite are sparsely distributed, making up less than one per cent of the rock.

In general, bornite is the principal sulphide mineral in the higher grade zones and chalcopyrite in the lower ones. A rough zoning of the sulphides has been detected by estimating percentages of bornite, chalcopyrite and pyrite at a particular horizon in the surface diamond drill core. (Dwg. No. 104). Sparsely disseminated pyrite forms an envelope on the margins of the mineralization enclosing an interior envelope in which chalcopyrite predominates over bornite. Bornite predominates over chalcopyrite in the central section.

The ore minerals occur mainly as fracture fillings either in quartz carbonate veins up to a foot in width or along joints, slips and minute fractures. They also occur sparsely disseminated through the host rock in most places as minute grains replacing hornblende or biotite. Coarse aggregates of copper and molybdenum sulphides occur in many of the quartz veins and in the eastern portion of the deposit a set of widely spaced quartz veins cutting across the main mineralized fracture system carry more than usual amounts of molybdenite. On fault planes along which there has been post-mineral movement, the sulphides have been pulverized and impart a dark grey to black colour to the gouge.

Molybdenite appears to be more closely associated with chalcopyrite than with bornite. It occurs in veins and veinlets of quartz, in fault gouge and in fractures associated with hairline quartz stringers. It has not been observed occurring as isolated flakes disseminated among the rock minerals.

3.2.4 The Oxide Zone

The oxide or weathered zone has been determined by percussion drilling, diamond drilling and rotary drilling. Some difficulty has been encountered in accurately defining the limits of the oxide zone because of inherent limitations in the equipment used.

The percussion drill is capable of drilling only to a maximum of 300 feet so its use was confined to areas of shallow overburden. Bedrock surface was detected by examining drill cuttings under a binocular microscope and confirmed from assays by a sharp increase in the copper content at bedrock. The cuttings from this machine are believed to be subject to down-the-hole salting, therefore, the lower contact as indicated by the percussion drill is suspect.

The diamond drill gave better information on the lower contact of the oxide zone due to the fact that it produced core. Its information on the upper contact is poor, however, because no core is produced from the hole in the cased sections and the diamond drillers almost invariably drove the casing from surface well into bedrock.

The best information was obtained from the Becker rotary drill which is capable of drilling to a depth of about 500 feet. Cuttings from this drill are not subject to salting because of a unique method of recovering them. The Becker drill was used near the end of this programme, to test areas of sparse or unreliable information.

It is believed that in spite of these difficulties the oxide zone as delimited is sufficiently accurate and will not introduce any great error into the overall reserve estimates.

The base of the oxide zone is where the soluble copper content drops below 20% of the total copper content. The top is, of course, the bedrock surface. The zone is illustrated on all the sections accompanying this report (Dwg. Nos. 105 - 118) and on section 11N (Dwg. No. 119) showing percussion holes with total and soluble copper assays.

The oxide zone has a maximum thickness of about 200 feet excluding certain narrow sections where oxidation has penetrated to greater depths along faults. Its greatest thickness is confined to the eastern side of the deposit where bedrock is near surface and it thins progressively towards areas of deep overburden. Much of the oxide zone is low grade but better grade oxide material will be stockpiled for possible future treatment to recover the copper.

The principal secondary copper minerals in the oxide zone are malachite, azurite, cuprite and native copper. Tennorite and secondary chalcocite have been tentatively identified. Secondary molybdenum minerals have not been positively identified but the predominant one usually associated with limonite is probably ferrimolybdite ($Fe_2(MoO_4)_3$ $8H_2O$ which has been observed in the shaft and test pit within a few feet of bedrock surface.

Malachite with associated small amounts of azurite is by far the most abundant secondary copper mineral and occurs throughout the oxide zone whereas cuprite and native copper are very minor constituents confined chiefly to the base of the zone. Primary copper sulphides are present throughout the zone but increase in proportion to the oxide minerals towards the base. The degree of oxidation, therefore, is progressively weaker towards the basê of the oxide zone.

SECTION 1V - EXPLORATION METHODS

Three zones or areas of mineralization are distinguished (Dwg. No. 103). The first is the Discovery Zone where bulldozing of long sinuous trenches along the banks of small gulleys exposed appreciable copper mineralization. This zone connects northward with the North Zone which was discovered by extending the induced polarization survey north from the Discovery Zone. The third zone, the Camp Zone, lies a short distance southwest of the Discovery Zone and was discovered also by an induced polarization survey. Limited work on this zone suggests, as with the Discovery Zone, that it is too small and too low grade to be of interest at present.

The following sections describe the methods used to explore the mineralized areas in general and the North Zone in particular.

4.1 TRENCHING

Extensive bulldozer trenching (Dwg. No. 103) was done on the southern area in the vicinity of the Discovery Zone and on the east side of the North Zone where overburden was not thick. On the North Zone the trenches were excavated to bedrock where possible in an east-west direction for lengths of 200 to 700 feet on north-south intervals of 200 to 400 feet. Because the overburden is moderately consolidated glacial material it was possible to excavate to a depth of about 30 feet.

Bedrock was exposed throughout most of the trenches but provided limited geological information because of the badly broken character of the bedrock surface. It did provide, however, some information on faults and fractures and on the eastern limit of the mineralization.

4.2 GEOCHEMISTRY

A reconnaissance geochemical sampling programme was undertaken over most of the Lornex property during the summer of 1965 when a field laboratory was set up on the property to test for trace amounts of copper and molybdenum (Newell 1966). Sampling was confined to the main stream courses and numerous discontinuous drainage gulleys, samples being taken with a hand auger at depths varying from three inches to three feet. Samples taken in the gulley beds in most places contained organic material, giving high results. Assuming that one organic sample can be compared with another, anomalous conditions can be detected. In rocky stream beds, where auger samples were unobtainable, fine silt was obtained by sieving a larger coarse sample. No attempt was made to achieve saturation geochemical coverage on a grid pattern because rapid changes of overburden depth and soil type, visible in the trenches, indicated that grid sampling might prove misleading.

Histograms showing frequency of occurrences of different value range were plotted. The patterns for organic 'bed' samples are broadly similar to those for the inorganic 'bank' samples, although the peak sample density moves into a higher range of values. Threshold values were established for both bed and bank samples at: 800 ppm Cu, 60 ppm Mo, and 700 ppm Cu, 60 ppm Mo respectively. Anomaly classification is essentially the same for both sample types and values in dispersions from the main mineralized zone are consistently in excess of 1000 ppm Cu.

The anomalies terminated sharply near the western limit of the mineralization, due to the deepening of overburden and to the direction of glaciation (northwest to southeast). The eastward dispersion by ice movement is reinforced by generally eastward drainage. Strong anomalies persist some 3000 feet to 4000 feet east of the known mineralization but anomalies are absent in Award Creek due to the excessive depth of overburden (greater than 200 feet).

Molybdenum values show a marked increase relative to copper on moving to the southeast, a trend which corresponds with the mineralization observed in rock exposures.

The vast majority of anomalous samples can be directly related to known mineralization, following its distribution in the rocks. The entire area west of Award Creek and southwest of Skuhun Creek shows no geochemical results of interest and no significance is attached to occasional isolated anomalously high samples here.

The geochemical approach on the Lornex property was not sufficiently definitive to provide targets for trenching or drilling, but was indicative of areas where more detailed work using more precise methods was warranted.

4.3 GEOPHYSICS

Induced polarization surveys were carried out over the Lornex property between June and October 1965, concurrently with geochemical sampling and after some trenching and percussion drilling of mineralized zones.

These surveys covered the whole property from north to south on east-west lines spaced at 800-foot intervals on the northern three-quarters and at 1600-foot intervals on the southern quarter of the property. Induced polarization and resistivity readings were taken with electrode spacing of 200, 300 or 400 and 800 feet over the areas of known mineralization and where anomalous readings were obtained. On the remainder of the property the readings were taken with electrode spacings of 400 feet.

The 800-foot electrode spacing was particularly significant over the North Zone, part of which lies under overburden 100 to 300 feet deep. This survey outlined two north-south zones of high chargeability with up to twelve times background readings. The main chargeability contours of these combined zones are depicted on the diamond drill hole plan (Dwg. No. 120) and shown in relation to the geology (Dwg. No. 103).

Subsequent core drilling of the anomalies was based on these induced polarization results.

A magnetometer survey was run concurrently with that of induced polarization, over the same grid, to assist in the interpretation of geological trends. No direct correlation with the induced polarization results was evident as the area of high polarization lies near the eastern edge of a broad magnetic depression.

Trends in the magnetic pattern show a general north, northwest strike over the southern part of the grid but change to a more northerly strike below the induced polarization anomaly that reflects the main mineralized area in the North Zone.

4.4 PERCUSSION DRILLING

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. Drilling was limited to a depth of about 300 feet so little information was obtained in deep overburden sections. Bedrock cuttings were collected for every 10 feet of drilling and the samples assayed for soluble and total copper and MoS_2. On comparison of percussion assays against later assays from diamond drill cores, it was found the percussion drilling tended to produce down-the-hole salting so that the assays could not be used in evaluating the deposit. Percussion drilling did, however, define more clearly the extent of the mineralization and gave information on the location of bedrock surface and the thickness of the oxide zone. Further, it acted as a guide for the diamond drill programme which followed.

4.5 ROTARY DRILLING

Rotary drilling was tried twice as a means of testing the deposit in the hope that it might reduce the amount of the more expensive diamond core drilling. The first attempt was with a large rotary drill imported from the United States which had been successfully used on the Carlin gold deposit in Nevada. An attempt was made to drill dry with this machine but difficulties developed once drilling penetrated the water table. Introduction of additional water to aid the drill did not improve the performance so the drilling was discontinued.

The second attempt was with a Becker rotary drill with the intention of testing the deposit to a depth of 1000 feet.

This drill uses double annular rods with air or air and water passing down between the rods and the cuttings returning through the centre of the inner rod. This method has the advantage that the outer rod acts essentially as a casing and prevents caving and contamination of the samples. The cuttings, raised by the compressed air and water mixture were blown into a cyclone and passed from there into a mechanical splitter. This machine was successfully used down to a depth of about 500 feet where mechanical difficulties forced abandonment of the idea of drilling deeper. The machine worked well in overburden and rock to shallow depths so a smaller machine of the same type was employed near the end of the 1968 diamond drilling programme to drill some 19 shallow holes for information on bedrock surface and thickness of the oxide zone in heavily overburdened areas.

4.6 DIAMOND CORE DRILLING

Diamond drilling commenced on the property in July 1965 and seven BX holes were drilled in the South Zone. In December 1965 two holes were drilled in the North Zone with encouraging results. Then a full scale diamond drill programme was initiated for this zone.

Canadian Longyear, Limited contracted to carry out the drilling using their No. 44 machines, NQ wireline equipment and mud as a lubricant. This equipment produces core 1-7/8 inches in diameter.

The drill holes were laid out on a regular grid along east-west oriented lines 800 feet apart but later fill-in drilling reduced the interval between lines to 400 feet. The spacing of holes along these lines was 300 to 350 feet, depending on topography.

Surface exposures indicated that the mineralized fractures dipped to the east so a pattern of holes inclined to the west at 60° from the horizontal was established. Holes inclined to the east or vertical were drilled in a few instances where special information was required, especially about the periphery of the deposit. As a general rule holes were drilled for a length of 1200 to 1400 feet giving a vertical depth of about 1000 feet. These holes are shown on the vertical cross-sections in this report (Dwg. Nos. 105 - 118).

Total footage drilled from surface on the property was 86,022 feet in 87 holes. Of this, 10,102 feet were drilled in overburden and 75,919 feet were drilled in bedrock. Due to the poorly consolidated nature of the oxidized surface of the deposit, much of this upper horizon could not be cored and is included in the overburden footage.

The 86,022 feet were drilled in the following areas:

(1)	Discovery Zone	7 holes	3,123
(2)	North Zone	66 holes	74,044
(3)	Camp Zone	8 holes	5,377
(4)	To fulfil option commitments	6 holes	3,478
		TOTAL -	86,022

All collar locations were surveyed for control in plotting. Acid dip tests were taken at 250-foot intervals down each hole and showed that in all cases the inclined holes steepened.

Attempts were made to Tropari survey the bearing and dip of the holes and although results were inaccurate, it was found that the holes tended to deviate to the south.

The length of all core was measured for recovery, logged and split. One half of the split core was sent for assay in 10-foot sections and the second half was stored in core trays at the site. Geologists logged the core noting rock type, alteration, structures and an estimate of the mineral content. The holes were plotted on vertical cross sections, scale 1" = 100', showing copper and MoS_2 assays and geology.

SECTION V - SURFACE AND UNDERGROUND DEVELOPMENT

5.1 THE SHAFT

Once surface drilling had indicated that the deposit contained several hundred million tons of possible ore grade material, a decision was made to investigate it underground. The reasons for this were:

(1) The grade as indicated by surface drilling was suspect because core recovery though good, was not 100%. It was reasoned that the very friable sulphide minerals were disproportionately lost to the sludge and could not be recovered due to the use of mud as a drilling lubricant.

(2) To provide muck for the pilot plant designed to determine metallurgical factors and recovery rates of the copper and molybdenum content.

A shaft location was chosen on line 11N (Dwg. No. 120) at a point roughly 300 feet west of the east margin of the mineralized zone. This site was chosen as it lies near the centre of the zone in a north-south direction and at a place where reasonably competent rock for the shaft could be expected. It was located within the zone to provide mineralized muck for the sampling plant and pilot mill. Further, from this shaft location the crosscuts would be driven along one of the most completely diamond drilled sections existing at that time - Section 11N.

The 8-foot by 18-foot three compartment shaft was collared to a depth of 32 feet in January 1967 and sinking began by the end of March. Construction of the sampling plant was underway in April and the pilot plant in June. A new assay laboratory and warehouse, together with new and larger camp facilities, were also constructed in the same period.

Sampling of the shaft collar muck was accomplished by picking two samples, one of fine material and the other coarse, from each loader scoop and averaging the results.

All four walls of the shaft were mapped from the top of the collar to its final depth at 550 feet by increments representing each bench after blasting and mucking out. All mappable features on the scale used (1" = 10') including attitude of faults, major fracturing and veining and frequency of fracturing too small to be mapped, were reported. Amount and distribution of mineralization was also reported.

All the muck from the shaft, calculated at 8,092.3 tons, was put through the sampling plant to obtain samples of suitable size for the assay laboratory. The muck was divided according to benches and that from each bench run through the sampling plant individually. Prior to sinking, vertical diamond drill hole No. 36 was drilled on the shaft location. The core from this hole was assayed in portions corresponding to each bench of the shaft, permitting a comparison between grade of the diamond drill hole and that of the material from the shaft.

5.2 LATERAL WORK

The shaft was completed to a depth of 550 feet by the end of September 1967 and crosscutting across the mineralized zone commenced shortly thereafter. The crosscut level was established at the 4,552 foot elevation or 497 feet below the shaft collar. Crosscuts 7' x 7' were driven 248 feet east and 1426 feet west from the shaft station (Dwg. No. 111) on a section through line 11N.

The underground programme as originally planned consisted of a crosscut across the mineralized zone and raising upon surface diamond drill holes to check their grade against bulk samples. It became apparent from the crosscut that raising was not practical in such heavy ground and so north-south drifts with horizontal drill holes east and west were substituted (Dwg. No. 125). Drill stations were established at roughly 200 and 400 feet north and south of the crosscut and a total of seven horizontal holes were drilled east and west, four from the north drift and three from the south. A hole planned to be drilled west from the south drift at 200 feet from the crosscut was not drilled due to lack of time.

Prior to driving the crosscut west, diamond drill holes were drilled ahead to permit a comparison between drill core and bulk sample. The first such hole was started at 96 feet west of the shaft at which point the crosscut was on line with Section 11N. Three such holes were drilled, UG-1 to 603 feet, UG-2 to 1032 feet and UG-6 to 1488 feet west of the shaft. The drilling was done with a Longyear underground drill with wireline equipment and mud as a lubricant. The whole core was assayed in lengths corresponding to the crosscut rounds. No drilling was done on the east crosscut.

Mapping procedures developed in the shaft were followed in the crosscut and drifts where the back and both walls were mapped.

All the muck from the crosscut and the muck from 40 rounds from the drifts was used as a bulk sample. The muck from each of these rounds was put through the sampling plant separately. For the remainder of the drift rounds, car sampling was used. In both the shaft and the crosscut, channel sampling was attempted but was abandoned first, because it was extremely dangerous in the bad ground and second, because reasonable channels could not be cut due to the variable friability of the rock.

5.3 GEOLOGY OF THE CROSSCUT

As the deposit is almost entirely covered by overburden, the crosscut provided the most complete exposure of the rock in the mineralized zone. Even where it was uncovered in trenches by bulldozing in the area of shallow overburden on the east side of the deposit, the rock nature and structure is obscured by weathering. The small open pit near the shaft reveals both rock and structure but is confined also to a limited area on the east side of the deposit.

The rock exposed in the crosscut is Skeena quartz diorite varying from fresh though fractured, to rock in which alteration has almost entirely destroyed the original minerals. In general, the least altered rock occurs in the eastern 300 feet of the crosscut and the more intensely fractured, faulted and altered rock in the remainder.

The crosscut was driven 248 feet east and 1426 feet west measured from the shaft. East of the shaft it intersected weak to moderately altered, relatively hard quartz diorite which is intensely fractured causing blocky ground. The majority of fractures are mineralized with chalcopyrite and bornite and strike from north-south to N 20° E and dip to the east. Their frequency in most places is about two to the foot but over short distances may be as high as five to the foot. These fractures are cemented by quartz and/or sulphides. Post mineralization movement along some of the fractures has resulted in the development of a film of gouge.

A set of post mineralization faults, in most places less than six inches wide, strike east-west parallel to the crosscut and are filled with gouge. The crosscut terminates to the east in a strong north-south fault dipping 55° east and filled with an estimated 15 feet of gouge.

A series of quartz veins from one to five inches thick striking north-south to N 45° E and dipping east are well mineralized with coarse molybdenite as well as chalcopyrite and bornite. These produce a higher grade molybdenite zone in the eastern part of the crosscut. A section of detailed mapping of the crosscut from 195 to 345 feet west of the shaft is shown (Dwg. No. 122).

The rock conditions to the east of the shaft persist to about 50 feet west of the shaft. Beyond this point the density of fracturing is greater and the alteration more intense. The fracture pattern is similar to that east of the shaft to a point about 550 feet west of the shaft but post mineralization faulting is more evident with the development of up to six inches of gouge in the faults. The faults are in three sets striking in the general directions - (1) north-south, (2) east-west, and (3) N 35° W.

Beyond 550 feet west of the shaft the mineralized fractures have a frequency of about three to the foot and in places are up to ten to the foot in any one set. They also occur in three sets - (1) east-west, (2) N 20° E, and (3) N 45° E. Gouge-filled faults are also more numerous and most of them dip to the south or west. Rock alteration is much greater and this, coupled with a higher percentage of gouge, causes the rock to crumble and slough when exposed to air. Even a small flow of water causes almost continuous sloughing into the crosscut on a minor scale.

The last 300 feet of the crosscut contains areas where potassium feldspar and/or quartz porphyry dykes have been introduced into the quartz diorite. The quartz porphyry dyke contacts are indistinct due to alteration so their attitudes could not be discerned.

The intense fracturing and faulting as revealed in the crosscut could materially affect the slope stability in the open pit. The fact that no underground opening will stand without timber support indicates that rock stability in the mineral deposit is low, particularly in areas of more intense faulting with abundant wet gouge. The waste wall rock, however, should be considerably more competent than the ground observed in the crosscut.

The pit slopes used in Report No. 4, Mining and Production Plan, are those recommended by Golder, Brawner and Associates. Comments on these pit slopes are contained in a letter from A. Soderburg which appears in Report No. 4.

5.4 TEST PIT

A small test pit was developed in July of 1967 to provide coarse material for autogenous mill testing. This pit also provided oxide material for metallurgical test work (Dwg. No. 120).

The pit was located northwest of the shaft between lines 11N and 15N where overburden was shallow and the rock provided muck reasonably representative of the mineral deposit. It was excavated to a depth of about 60 feet to obtain material below the oxide zone and a total of 77,965 cubic yards of material consisting of 25,318 cubic yards of glacial till and 52,647 yards of rock was removed.

Some 6000 tons of oxide material, 6800 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 at the pit site for delivery to the mill if and when required.

It was found that, with the exception of a few small areas of hard rock, all material could be ripped and scraped with a bulldozer without resorting to drilling and blasting. Drilling and blasting was done in the sulphide zone but only because it was believed that this would provide coarser and more suitable muck for the autogenous mill tests.

The overall pit wall slope achieved was about 45°. In the nine months that have elapsed since the pit was excavated, no major slides have occurred but sloughing from the walls has taken place particularly at the east end of the pit where fracturing dips into the wall and on the north and south walls in areas of strong gouge-filled faults. At these places the overall slope is approaching the angle of repose of the rubble (35°).

The walls of the pit were mapped but no sampling was done as the grade of the ore in the pit has little bearing on the overall grade calculations.

SECTION V1 - SAMPLING AND ASSAYING

6.1 SAMPLING METHODS

6.1.1 Percussion Drilling

Preliminary sampling of the upper horizon of the deposit was done by percussion drilling. Because of suspected down-the-hole salting, the results of this drilling were not used in calculating grades of the deposit. These results were used, however, to delimit the extent of the oxidized capping (Dwg.No. 119).

Using an Atlas Copco truck-mounted percussion drill, holes of two inch diameter were drilled to a maximum depth of 300 feet. In deep overburden, casing was set to a maximum depth of 70 feet and beyond this an open hole was drilled. In all, 511 vertical holes were drilled for a total footage of 91,000 feet. Of these 192 holes, for 45,845 feet, were drilled in the main mineralized zone.

Water circulation was used for most of the drilling and the cuttings brought to surface as a slurry. At the machine the slurry was passed through a motor driven rotary splitter where a 1:8 split was made. Samples were collected from consecutive 10-foot lengths and the resulting 4 to 5 pound product was sent to the assay laboratory for sample preparation and assaying.

6.1.2 Surface Diamond Core Drilling

Surface diamond drilling was considered to be the most accurate method of sampling the sulphide zone, though it was found to be of little value for sampling the oxidized zone.

Total surface diamond drill footage on the property was 86,022 feet in 87 holes. Of this footage, 74,044 feet in 66 holes were drilled in the main mineralized zone.

Average measured core recovery by length for this drilling was 94.3%. As the core was split before sampling, it was not possible to obtain an accurate weight recovery, but experience has indicated that this would have been 5% to 10% lower than the recovery calculated from core length measurements.

All core was split and sampled in 10-foot sections using Heath and Sherwood core splitters. One-half was bagged for delivery to sample preparation and the other half placed in core trays for storage. The fines from each 10-foot section were tabled and halved. One-half of the fines west to the sample and the second half to the storage tray.

As a check on the accuracy of the core splitting the second half of the core from some of the holes was taken from storage and assayed. The following results were obtained:

HOLE NO.	FOOTAGE REPRESENTED	FIRST HALF CORE		SECOND HALF CORE	
		% Cu	% Mo	% Cu	% Mo
9	440 - 630	0.32	-	0.33	-
24	75 - 1394	0.37	.016	0.40	.014
36	36 - 613	0.44	.021	0.43	.021
TOTAL	2,086				
AVERAGE		0.37	.018	0.39	.017

These checks indicate that the split core makes a satisfactory sample.

6.1.3 Underground Bulk Sampling

All the muck from the shaft and crosscut, as well as some of the muck from the north and south drives, was put through the sampling plant.

The muck from each bench from the shaft and each round from the lateral work was hoisted separately to surface, trucked to the sample plant area and placed in one of the six concrete storage bays where it was assigned an identifying number.

A 1-1/2 yard front end loader delivered feed to a receiving hopper at the crushing and sampling plant. It was withdrawn by a 24" reciprocating feeder onto a 24" belt which carried the muck to the 'Universal Pacemaker' crushing plant. This unit was equipped with an 18" x 24" jaw crusher, 30" crushing rolls and double deck screens dressed with 1" and 3/8" screen cloth.

Screen undersize was fed to the sampling plant which consisted of two Snyder and one Denver sampler and two sets of roll crushers, 12" x 8" and 10" x 6".

Individual rounds were crushed and sampled separately and the whole plant cleaned out after each batch to prevent contamination. A 10% cut was taken from the minus 3/8" screen undersize and reduced in the 12" x 8" crushing rolls to minus 1/4". A second 10% cut was taken from this product and reduced to minus 10 mesh before the final 10% sample was cut. Original design stipulated that the final sample was to be 0.10% of the total; actually it amounted to about 0.13%.

Three typical tests taken in different months are shown in the following table:

SUMMARY OF % WEIGHTS PRODUCED BY SAMPLE CUTTERS FROM THE FEED

TEST NUMBER	1	2	3
1st Cutter	14.10%	14.5%	14.0%
2nd Cutter	10.4%	11.5%	10.4%
Final Cutter	9.1%	9.3%	9.1%
Final Sample	0.13%	0.15%	0.13%

Particle size on the final sample was set at 5% plus 10 mesh and this was checked regularly by the Geological Department. Any increase in the plus 10 mesh fraction was immediately corrected by adjustment of the sampling plant crushing rolls.

The final sample, whose weight varied from 75 to 300 pounds in proportion to the size of the round or bench, was placed in heavy-duty plastic bags and delivered to the sample preparation room.

Attention is directed to the Flowsheet and Legend for the crushing and sampling plant (Fig. No. 2).

To check that the sample plant product was truly representative of the bulk sample, the sample plant rejects from five separate rounds were passed again through the sample plant. The assay results obtained from this new sample checked against the original as follows:

ROUND NUMBER	ORIGINAL PRODUCT ASSAY		RE-RUN PRODUCT ASSAY	
	% Cu	% Mo	% Cu	% Mo
Bench 14	0.89	0.104	0.84	0.117
Bench 49	0.11	0.004	0.12	0.005
XCW 202	0.54	0.012	0.52	0.012
XCW 203	0.50	0.007	0.51	0.007
XCW 287	0.31	0.003	0.32	0.003
AVERAGE OF 5 CHECKS	0.47	0.026	0.46	0.029

These checks demonstrated that the sampling plant produced a sample representative of the whole bench or round being tested.

6.1.4 Underground Diamond Core Drilling

The proposed underground sampling programme included horizontal drill holes to lengths of 600 to 650 feet to be drilled east and west from centrally located drill stations in the north and south drifts on Sections 7N, 9N, 13N and 15N to give coverage of a total of 1200 to 1500 feet east and west parallel to the cross-cut. It was found, however, that the Longyear underground machine using NQ wireline equipment and mud on horizontal holes was not as efficient as the No. 44 machine on angle holes from surface. Not only was the core recovery less but also it was not practical in the bad ground to drill to the depths proposed. Maximum lengths attained by the holes varied from 500 to 600 feet.

As these holes were to be used to extend the sampling area north and south from the crosscut, similar holes were drilled ahead of the crosscut so that results from these holes could be compared with those from the bulk sampling of the crosscut and so assist in evaluation of the holes from the drifts.

The core from the underground holes was logged and measured for recovery. An accurate weight recovery was obtained as the whole core was sent for assay. The core from the holes ahead of the west crosscut was sampled in lengths equal to the equivalent crosscut rounds but core from all other holes was sampled in 10-foot lengths. A comparison of the measured core recovery by weight and by length is as follows:

HOLE NO.	RECOVERY BY WEIGHT	RECOVERY BY LENGTH
UG-1	81% (0-368')	91%
UG-2	71%	88%
UG-3	86%	90%
UG-4	74%	87%
UG-5	87%	91%
UG-6	88%	94%
UG-7	77%	83%
UG-8	84%	87%
UG-9	97%	94%
UG-10	86%	92%
AVERAGE	83.1%	89.7%

The samples were prepared for assay using the same procedure as used for surface diamond drill core.

6.1.5 Car and Channel Sampling

Inasmuch as the north and south drifts were driven parallel to the general trend of the mineralization, it was decided not to bulk sample all the rock from these headings. Some rounds were bulk sampled but most were car sampled. A car sample consisting of one handful of fines and one of muck measuring approximately two inches square was taken from each one ton car of muck. The samples from each of these cars from a single round were combined to make up a sample of from 20 to 30 pounds. The sample was then prepared for assay in the same manner as the diamond drill core.

To compare the results of car sampling with those of the bulk sample method the muck from 33 rounds was both car sampled and bulk sampled. The average grade for the 33 bulk samples was 0.426% copper and 0.011% Mo. while the car samples averaged 0.417% copper and 0.010% Mo. From these results it was concluded that car sample results would be satisfactory for the north and south drifts.

As a part of the underground sampling programme, channel sampling of the shaft and crosscut walls was planned. Using an air driven chipper an attempt was made to cut channels six inches wide and one inch deep, catching the cuttings on a canvas sample sheet. In the shaft, vertical channels were cut on the north and south walls, whereas in the crosscut, horizontal channels were cut along the north and south walls.

Because of the extremely fractured and friable nature of the rock, it was impossible to obtain a reasonably accurate sample. For the same reason, it was found that the practice was dangerous and in most cases impractical due to tight lagging of the walls so all attempts to channel sample were eventually discontinued.

6.2 SAMPLE PREPARATION

6.2.1 Diamond Drill Samples

The diamond drill core was prepared for assaying in the following manner:

(a) Dried overnight

(b) Reduced to minus 1/2 inch in a jaw crusher

(c) Quartered in a 3/4 inch Jones riffle

(d) Reduced to minus 10 mesh in a gyro-crusher

(e) Cut to 1 to 2 lbs. in a 1/2 inch Jones riffle

(f) Pulverized to minus 80 mesh

(g) Rolled on paper

(h) Two 100 gram samples dipped out with a spatula

As a check on the accuracy of the sample preparation method, the 1/2 inch rejects from six samples were re-prepared and assayed. The following results were obtained:

SAMPLE NO.	ORIGINAL ASSAY % Cu	% Mo	SECOND ASSAY % Cu	% Mo
20805	0.25	.002	0.26	.003
20806	0.18	.001	0.18	.001
20816	0.58	.015	0.56	.014
20832	0.44	.013	0.43	.016
20833	0.34	.003	0.30	.002
20889	0.30	.085	0.31	.105
AVERAGE	0.35	.020	0.34	.024

These checks show that the sample preparation was satisfactory.

6.2.2 Underground Bulk Samples

The underground bulk samples from the sampling plant were prepared for assay in the following manner:

(a) The sample was reduced to about 10 lbs. in weight by means of a Jones riffle with 3/4 inch openings.

(b) This 10 lb. sample was dried overnight in an open gas fired dryer at around 130° F.

(c) The dried sample was crushed by small gyra-disk crusher to pass a 20 mesh screen.

(d) The sample was cut to about 2 lbs. with a Jones riffle with 1/2 inch openings.

(e) The 2 lb. sample was pulverized and screened through a 100 mesh screen with oversize being reduced in hand mortar and pestle.

(f) The sample was then placed in a bottle mixer and mixed for at least 5 minutes, then placed on a clean piece of paper, flattened out and the required samples taken out by dipping with a small spatula. The remainder was stored in plastic bags with fold-over seals for future use.

As a check on the sample preparation method the sample preparation rejects from the following samples were re-prepared and assayed:

ROUND NUMBER	ORIGINAL ASSAY % Cu	% Mo	RE-RUN ASSAY % Cu	% Mo
Bench No. 8	0.80	.031	0.81	.031
XCW No. 8	0.46	.018	0.47	.024
XCW No. 10	0.27	.024	0.26	.019
XCW No. 12	0.35	.022	0.36	.022
XCW No. 14	0.43	.027	0.51	.030

From these results it is concluded that the sample preparation method is satisfactory.

6.3 ASSAY METHODS

Early in the exploration programme a simple laboratory was set up at Lornex to run percussion chip samples for total and soluble copper employing the quick iodide titration method. These samples were also sent to Coast Eldridge laboratory, an independent assay company in Vancouver, for total molybdenum and copper assay.

During this early work, diamond drill core was split on the property and sent to the Company's Elliot Lake laboratory where total copper, soluble copper and sulphide molybdenum assays were run. Samples were pulverized to minus 100 mesh and assayed for total copper by colorimetric method (cuproine) using a Beckmann B spectrophotometer for reading. Soluble copper assays were run essentially the same way, but using a different dissolution stage at the beginning.

Assays for sulphide molybdenum were run by a colorimetric technique and read on a Beckmann B Spectrophotometer following preparation by acid digestion to remove soluble molybdate. Every fifth sample was sent to Coast Eldridge laboratory in Vancouver where checks on total copper and soluble copper were run using a similar technique. The Coast Eldridge technique for molybdenum differed in that it did not include pre-digestion to remove the soluble molybdate.

From August 1966 on an atomic absorption unit (Techtron A.A-2 model) was used at the Elliot Lake laboratory for copper assaying of Lornex samples and from September on, constant checks were run colorimetrically using the spectrophotometer. The atomic absorption unit was used for molybdenum determinations and constant checks were run on the spectrophotometer with every fifth sample being sent to Coast Eldridge for check.

The Lornex laboratory went into operation in November 1966. Prepared samples were delivered to the assay office as they were completed. The bag was opened and the required amount was dipped out by means of a small spatula. Copper analyses were made by the Atomic Absorption Spectrophotometric Method (Perkins-Elmer 290 Model) and molybdenum assays were done using the Thiocyanite-Stannous Chloride Photometric Procedure.

6.3.1 Soluble Molybdenum

Assays for soluble molybdenum were run on Lornex surface diamond drill core samples in August 1966, but the values reported were so low that it was decided to discontinue this assaying. It was resumed, however, in December 1967 and January 1968 when percussion drill samples, primarily from the oxide zone, were run for soluble molybdenum. The results showed a soluble molybdenum content in the oxide zone ranging from 0.001% to 0.012%. They also showed, when compared to the total molybdenum content of the same samples, that an appreciable percentage of the molybdenum in parts of the oxide zone is in the form of sulphide.

Soluble molybdenum assays were run on muck samples from the shaft and the content was in excess of 10% of total molybdenum in the upper 62 feet in what has been delimited as the oxide zone. Below, in the sulphide zone, a marked decrease in the soluble molybdenum content was indicated.

The method used to determine the soluble molybdenum and sulphide molybdenum was as follows:

(a) Weighed samples were subjected to a leach using hot concentrated hydrochloric acid, then filtered. This filtrate was then tested for soluble molybdenum.

(b) The filtrate was digested in a hot concentrated mixture of acids including perchloric acid and when it had dissolved the solution was tested for molybdenum representing that originally in the sulphide form.

6.3.2 Check Assays

A system of regular checks on assaying of drill core samples was set up from the very beginning of the diamond drill programme in 1965. As no laboratory was in operation on site, all split core was sent to Elliot Lake for copper and molybdenum assay and, in turn, pulp samples were sent to Coast Eldridge laboratory for check copper and molybdenum. Every fifth core sample (one sample representing 10 feet of core) was checked. After the Lornex laboratory went into operation in November 1966, the original copper analyses were made there, every fifth pulp sample being sent to Elliot Lake for check assaying and approximately every tenth to Coast Eldridge. In June 1967, a system of checks was initiated whereby the Lornex laboratory results for drill core samples were checked exclusively at Coast Eldridge, the latter being an independent and impartial assay house. Every tenth sample was sent to Coast Eldridge for checks on copper and molybdenum and all soluble copper assays were run at Coast Eldridge and checked at Elliot Lake.

Prior to November 1966, all drill core assays used for calculations were from the Elliot Lake laboratory but after November all core assays used were from the Lornex laboratory. At no time were check assays from another laboratory inserted when discrepancies existed and a variance simply called for re-assaying at the laboratory of origin.

When the underground exploration programme was underway <u>every</u> bench sample from the shaft and <u>every</u> round sample from the lateral development was sent for check assaying to Coast Eldridge. If the copper analyses did not agree within 5% for an assay of greater than 0.20% copper or within 10% for an assay of less than 0.20% copper, the sample was re-assayed at the Lornex laboratory. In the case of molybdenum, re-assaying was called for when the analyses did not agree within 10% for assays above 0.005% total molybdenum or within 100% for assays below 0.005% total molybdenum. The result used was the Lornex assay that checked the best with that of Coast Eldridge. The Elliot Lake laboratory was used to re-check only on those occasions when a wide discrepancy persisted after the sample had been assayed twice at the Lornex laboratory.

To summarize; in order to ensure the accuracy of results reported by the Lornex laboratory,the following system of check assays was instituted in June 1967:

(a) The Lornex laboratory sent out the following samples to Coast Eldridge, Professional Service Division, for analysis for total copper and total molybdenum:

  (i) every tenth core and percussion sample from surface drilling,

  (ii) all shaft bench samples,

  (iii) all round samples from lateral development work,

  (iv) all underground diamond drill core samples.

(b) Coast Eldridge assay values were recorded and compared.

(c) Samples showing discrepancies were re-assayed in the Lornex laboratory.

(d) If a wide discrepancy still persisted, a fourth analysis was made by the laboratory at Elliot Lake.

(e) Every tenth sample of mill products assayed by Lornex and also sent to the Elliot Lake laboratory for checking. Where agreement could not be reached, the sample was then sent to Coast Eldridge.

(f) Periodically, samples were taken at random or for special checks and submitted to other analysts.

In this way, it was possible to develop confidence in the accuracy of assays reported by the Lornex laboratory as trends were promptly spotted and investigated.

The following are averages of the results of all checked assays from the surface drill core as received from the Lornex, Elliot Lake and Coast Eldridge laboratories:

MoS_2

HOLE NO.	NUMBER OF SAMPLES	LORNEX %	ELLIOT LAKE %	COAST ELDRIDGE %
8 to 72	861	-	0.019*	0.023+

COPPER

HOLE NO.	NUMBER OF SAMPLES	LORNEX %	ELLIOT LAKE %	COAST ELDRIDGE %
8 to 40	566	-	0.379	0.390
41 to 72	174	0.295	0.284	0.306
41 to 87#	216	0.314	-	0.323

* Represents sulphide molybdenum only converted to MoS_2
+ Represents total molybdenum converted to MoS_2
# Holes 70 and 73-77 omitted

Attached are four graphs (Figs. 3-6) illustrating the degree of comparison for copper and MoS_2 assays between Elliot Lake and Coast Eldridge and Lornex and Coast Eldridge. In each case the Coast Eldridge assay has been used as a base.

Table 1 shows the assays and their checks for the first 45 rounds from the west crosscut.

When considering these comparisons it should be noted that in the case of molybdenum assaying there were the following differences in assay technique:

(1) Prior to June 1, 1967, the Elliot Lake laboratory assayed for molybdenum after preparation by acid digestion to remove the soluble molybdate. This was not done at the Coast Eldridge laboratory. The Elliot Lake results are thus based on the sulphide molybdenum content converted to MoS_2 while the Coast Eldridge results are based on the total molybdenum content converted to MoS_2.

This difference applies to the results of diamond drill holes 8 to 72 which are compared above. It also applies to the results illustrated in Figure 6.

(2) Following June 1, 1967, all laboratories concerned assayed for total molybdenum and reported it as such. The assays for the crosscut rounds shown in Table 1 were run after June 1, 1967 when no differences in assay techniques existed.

SECTION VII - ASSESSMENT OF MINERAL DEPOSIT

7.1 GRADE TRENDS

7.1.1 General

An interpretation of the results of surface diamond drilling to produce trends in the distribution of the copper and molybdenum was attempted following the completion in 1967 of the drilling up to hole number 82 and the start of the underground programme. Results of holes 83 - 87 drilled in late 1967 and 1968 were later incorporated into the trend pattern which is shown on the drill hole sections of lines 11S - 39N (Dwg. No. 105 - 118) that span the length of the main mineralized zone and the proposed open pit.

For the purposes required, it was found to be practical to attempt a zoning of the copper distribution only and molybdenum distribution was interpreted then only insofar as it is governed by zones based on copper content.

Grade trends were established on the belief that the distribution of copper and molybdenum that produces low and higher grade zones are governed by a pattern of structural features such as fracture, fault or shear zones of a more or less continuous nature. Trend lines shown, therefore, do not represent contours between assay value intervals but mark only the attitude in strike and dip of variations in copper content of zones of respectively higher and lower magnitude.

The results are considered to present a pattern of sulphide occurrence consistent with geological understanding of the zone, that is, useable in planning of the pit and in the estimation of reserves.

7.1.2 Establishing Trends

Differences in copper grades on which trend lines were established are based on grouping of contiguous assays selected as surface drilling progressed. This selection was predominantly a visual judgement. As trends developed, re-examination of the grouping was attempted in some cases but only minor changes were introduced. In only rare cases were averages used in the trends derived from a group of assays representing less than 50 feet of core length.

The pattern of the copper trends was first established on section and only after completion of the sections was the distribution in horizontal plan attempted.

The attitudes of the grade trends in dip were based on primary structural features such as faults, shears and fracture angles as recorded in logs of surface drill holes and portrayed on geological cross sections. An example of this information, as interpreted and used, is shown on the geological cross section on Line 11N (Dwg. No. 111).

A general guide to the steep easterly dip of the mineralization is also provided by the eastern or hanging wall where there is a marked change from a low copper content (.006 - .015) to a relatively much higher content. This change is shown on sections 1S and 11N (Dwg. Nos. 107 & 110) and illustrates the easterly dip in this part of the deposit. This is confirmed by a well marked dip of a subsidiary higher grade zone on the eastern side of section 3N. (Dwg. No. 108).

Similar dips to the east, though apparently steepening westwards, are substantiated by the distribution of low and higher grade zones throughout the deposit. Most zones are continuous but others are pinched out.

The trend pattern in plan was derived by the transference of the points of intersection of trend lines on section to plans spaced at 80-foot vertical intervals over the vertical range 3752-5112 feet.

Assays attributed to the zones developed in plan were those of either one of the adjacent holes - dependent on whether the plan cut the zone above or below a horizontal line drawn half way along the median line between holes in the particular zone.

Little surface geological information was available due to overburden. Geophysical work suggested a NW strike over the southern end of the mineralization, swinging to a more northerly direction over the rest. Regional geological mapping and that in the test pit and underground workings also indicates a generally N-S trend. This information and the general grade distribution pattern was used to define the grade trends in plan, producing a pattern consistent with that seen in section. As in the case on section a clear hint as to the general strike trend is also provided by the reasonable continuity of the change from low grade material on the eastern side to higher grade material in the main zones. Similarly, two fairly consistent subsidiary higher grade zones in the lower grade part indicate a strike north or west of north.

A plan at the 4552 elevation (Dwg. No. 124) shows the grade trends at the approximate level of the underground development. On this plan, trend lines are shown projected into the oxide zone and overburden in order to show continuity. This projection indicates the probable pattern of grade immediately below the oxide or overburden zones should they not continue to the plan below.

Grades deduced from these grade trends were designated "assigned" grades and provided a figure against which underground drill hole and the crosscut bulk sampling results could be assessed. A detailed plan (Dwg. No. 125) at the 4552 foot elevation shows the grade trends within the area tested by the underground diamond drilling, the east and west crosscuts and the north and south drives.

7.2 DISTRIBUTION OF COPPER

Examination of the grade trend plans and sections clearly show that the copper mineralization is largely confined to a major zone extending from Section 5S to Section 31N. This zone has an overall strike slightly west of north.

The eastern margin is generally continuous though subsidiary parallel zones occur to the east of the general main mineralized zone. The western margin, where seen, is less clear but the Award Creek fault forms a sharp cut-off along the greater part of the western edge. Mineralization decreases to the north and south as, in general, grades decrease and higher grade zones feather or pinch out.

Table II indicates the weighted assigned copper grades on sections 5S to 31N between continuous trend lines delimiting the major part of the main zone at the 4552 elevation and at 4852 and 4232 on section 11N. These results indicate that the assigned grade is fairly constant over the central part of the deposit, i.e. from section 3N to 23N, but a clear decrease in width and grade occurs towards the north and south ends.

In greater detail, the plans show a concentration of copper in two main higher grade zones, one in the southern part along the eastern edge and the other to the north along the west side in a vague en echelon arrangement. Both zones display a change in strike from NNW to one more northerly. The eastern zone is of generally smaller size and is less uniform as it contains minor weakly mineralized sections. Both zones show a spatial and apparently structural relationship to pre-ore intrusions, the quartz porphyry and Bethsaida granodiorite. In general, an increase in grade towards the centre of these two zones is apparent. This increase being often recognizable along individual zones defined by copper grade trend lines.

Some difference in the nature of occurrence of the copper oontent between the eastern high grade zone and that to the west is apparent in the results of underground work. Much of the eastern zone shows erratic high and low grade copper assays as shown in the bar graph (Dwg. No. 126 - 129) while cross-cut rounds of the western high grade zone exhibit a more uniform grade distribution. A section of detailed mapping of the cross-cut (Dwg. No. 132) portrays part of the erratic mineralization illustrated by rounds WXC 28 - 47 and part of the more uniform type by rounds WXC 48 - 70. It is clear that the former is related to an area of erratic but numerous quartz veins carrying concentrations of copper and molybdenum sulphides and the latter to rock more uniformly fractured. Underground drill holes to the north and south of the erratic zone do not show clearly that this type of mineralization is continuous.

7.3 DISTRIBUTION OF MOLYBDENUM

No separate attempt has been made to define a pattern of molybdenum distribution. Individual assays in surface and underground drilling show that the molybdenum content is more erratic than that of copper due to the greater occurrence of molybdenum in high grade veinlets and fracture coatings.

A relationship of molybdenum to copper mineralization based on trends established for copper is discernible. Greater concentrations of molybdenum occur in the higher grade copper zones though no factor governing the assay ratio is determinable except over large sections. In each of the four categories in the reserve estimate, the molybdenum to copper ratio is about 1:30. This relationship and visual checks indicate that no significant zones of higher grade molybdenum will be omitted by the presently determined classification of ore in or out of the pit.

Examination of the bar graph of the bulk sampling of the east and west crosscuts show some common features and differences in the molybdenum and copper distribution such as:

(1) Peaks in the copper distribution are usually accompanied by high molybdenum values.

(2) The erratic high grade copper assays in the zone of veining are also reflected in the molybdenum assays.

(3) The concentration of molybdenum in the eastern high grade copper zone is not repeated in the western high, but more uniform grade, copper zone.

The assay results in the west crosscut show the following grades of molybdenum for the respective zones. Distances are measured from the shaft.

ZONE	FROM	TO	DISTANCE	% MO
East	55	300	245	0.0430
"	300	436	136	0.0117
Centre	436	771	335	0.0096
West	771	1426	655	0.0126

7.4 DYKES

The affect of various intrusive rocks into the Skeena quartz diorite on grade distribution has been examined. Most dykes are considered to be of pre-sulphide age and thus do not introduce zones of relatively low grade or waste.

The Bethsaida granodiorite in the SW corner of the zone is mineralized. Assays are in general not different from those elsewhere except for one section of .95% Cu. This area is, however, generally outside the limits of interest and anomalous values will not greatly affect reserve estimates.

No variation in copper or molybdenum content is discernible as being related to the intrusions of quartz porphyry in the south end of the deposit though, as noted elsewhere, a structural relationship may exist. Similarly, the flatter granite dyke in the northern part of the zone has no affect on copper or molybdenum distribution.

Basic to intermediate dykes such as intersected in surface drill holes 13 and 18, shown on the geology vertical cross section (Dwg. No. 111), commonly contain an unusually high copper content.

Two dykes of this type were intersected in underground drill hole 9 and showed a markedly anomalous high copper grade. Adjustment for this has been made in the case of these dykes where the affect on grade trends and hence reserve estimate is significant. However, the overall effect is considered as negligible as the dykes make up a very small proportion of the ore, and the affect on grade is not readily calculable as their attitude is uncertain.

7.5 UNDERGROUND TEST PROGRAMME

The underground programme of crosscutting, drifting and flat hole drilling was carried out to:

(1) provide material for the pilot mill,

(2) to check the reliability of the copper and molybdenum distribution as interpreted in the grade trend plans,

(3) to examine the nature of the mineralization and

(4) to give measure of the accuracy of the assays surface drill holes.

The following are the average grades for the underground development and drill holes.

		Ft.	% Cu	% Mo
Shaft		33 - 550	.500	.0312
East crosscut		248	.214	.0074
West crosscut		1426	.508	.0170
North drive		484	.411	.0155
South drive		389	.395	.0122
DDH's UG	1	506	.514	.0215
	2	400	.505	.0084
	6	457	.597	.0109
	3	523	.363	.0247
	4	580	.403	.0178
	5	565	.482	.0259
	7	566	.512	.0150
	8	550	.338	.0054
	9	600	.520	.0076
	10	552	.466	.0256

The layout of the development and the location of the drill holes are shown in relationship to the grade trends established for the area on the plan of part of the 4552 elevation plan (Dwg. No. 125) and the crosscuts and shaft on section 11N (Dwg. No. 111).

Four stages of interpretation and assessment of the results are attempted in the following discussion. These are:

(1) That the crosscut is representative of a significant portion of the ore body.

(2) That the flat drill holes give results comparable to the bulk sample as obtained in the crosscut.

(3) Whether there is any significant variation in copper or molybdenum grade between surface drill hole and underground results.

(4) That the crosscut and flat drill hole results are compatible with the grade trend pattern and thus the latter is a valid basis for an assessment of the deposit.

7.5.1 Location of Crosscut

The crosscut is located to cut approximately across the centre of the main mineralized zone and to sample both the eastern and western high grade zones on a section of typical surface holes drilled at several stages in the exploration programme. Analysis of the average copper grades in Table II indicates that those assigned for the line of the crosscut are typical of the main part of the deposit, i.e. from section 3N to section 23N. The average grades along these lines were calculated between reasonably continuous grade trend lines marking the eastern and western limits of the main part of the deposit which represent approximately the zone tested by the crosscut. The eastern limit passes through the east crosscut 140 feet from the shaft and the western limit at 1337 feet along the west crosscut. On more northerly sections the Award Fault is the western limit.

7.5.2 Comparison of Crosscut to Underground Drill Holes

Concurrently with the advance of the west crosscut, three drill holes, UG-1, UG-2 and UG-6 were drilled ahead as pilot holes.

Bar graphs (Dwg. Nos. 126 - 129) show copper and molybdenum grades for portions of the drill holes and corresponding crosscut rounds.

Table III presents a comparison of the averages of the holes and the corresponding lengths of the crosscut. A gap of 30 feet between holes UG-1 and UG-2 has not been used in calculations for the crosscut.

Logging of UG-2 prior to assaying disclosed that from 277.5 - 295 feet the drill hole ran along a high grade copper bearing vein. Due to this and the lack of any comparative increase in the grade of the corresponding crosscut rounds (Nos. 175 - 179) this section was omitted from the calculations and comparisons. With these adjustments the cumulative weighted averages compare as follows:

	Feet	% Cu.	% Mo.
West Crosscut	1282	.524	.0174
DDH's UG-1, 2 and 6	1282	.516	.0174

These values are sufficiently close, in the case of copper, to assume the equivalence of underground drill holes and crosscut bulk sampling. Due to this and the assumption that the crosscut is a reliable sample no cutting of copper assays has been done for any drill holes.

The cumulative weighted averages for molybdenum show that the crosscut gives a value about 18% higher than the drill holes.

To allow a comparison of drill hole results to assigned grades and to facilitate the development of a zoning of grade trends based on underground results the following adjustments have been made to the results of UG-1, 2 and 6.

(a) Crosscut assays for the gap between UG-1 and 2 have been added to UG-2.

(b) Corresponding crosscut assays have been substituted for those omitted in UG-2 because of the high grade vein.

With these adjustments the cumulative averages for the drilling compare to those for the crosscut as follows and as shown in Table IV.

	Feet	% Cu.	% Mo.
West Crosscut	1330	.521	.0171
UG-1, 2 and 6	1330	.514	.0146

These results are essentially identical with those in Table III.

In general, the copper averages for zones in Table IV are comparable though differences in individual sections are evident. These differences may in part be accounted for by deviation of the drill holes from the crosscut line causing a displacement of zones. The main discrepancies however are attributed to the drill hole sampling method. It is inherent in such sampling that variations between individual drill hole samples will be greater compared to those of a bulk sample which has a smoothing effect. Drill holes will be more affected by small, low grade zones or high grade veins, etc.

Molybdenum results, also shown on the bar graphs (Dwg. No. 126 - 9), show more clearly the erratic and wider variation in drill hole samples compared to bulk samples. The role of veins is probably of greater influence in the distribution of molybdenum values than copper values.

7.5.3 Comparison of U/G DDH's and Development to Surface Drill Holes

Underground Drill Holes

Ten flat holes were drilled across the main mineralized zone from the crosscut and drill hole stations on sections to the north and south. Total footage drilled was 5300 feet in the holes as follows.

Section 15 N	East	UG-8	551 feet
	West	UG-7	566 feet
13 N		UG-3	523 feet
		UG-4	580 feet
11 N	X Cut	UG-1	506 feet
		UG-2	400 feet
		UG-6	457 feet
9 N	East	UG-5	565 feet
7 N	East	UG-10	552 feet
	West	UG-9	600 feet
			5300 feet

The location of these holes is shown on a plan of the 4552 foot level (Dwg. No. 125) in relation to the grade trends from which copper and molybdenum assigned grades were forecast for each hole and intervening drill hole stations.

Drill holes on sections 15 N, 11 N and 7 N were assigned grades directly from grade trend plans. Those on sections 13 N and 9 N, i.e. between lines of surface drill holes were assigned grades interpolated from sections to either side.

Results of the drilling are tabulated in Table V.

Round assays have been inserted for drill hole stations and gaps in the drilling along the crosscut to provide continuous lengths for comparison with assigned grades. Round assays have also been substituted for a vein section in UG-2.

All the underground holes along lines of surface holes show increases in average copper content above that assigned from surface holes. Underground holes UG-3 and UG-4, however, produced average copper values lower than the interpolated assigned figure. UG-5, similarly interpolated, shows an increase above the assigned.

Over the total footage drilled the cumulative averages compare as follows:

D.D.H.	5440 Feet	.459% Cu.	.0163% Mo.
Assigned	5440 Feet	.422% Cu.	.0165% Mo.

These indicate that flat drill holes give results essentially higher in the case of copper and no significant variation in the case of molybdenum. The copper average is 8.7% above that assigned.

Crosscut

Crosscutting to the east and west of the shaft was continuously bulk sampled. These assays are shown on the bar graphs (Dwg. Nos. 126 - 129) against assigned grades. Correlation between assigned grades is locally close, as in the east crosscut. Elsewhere, though generally apparent, the correlation is more vague, - most zones being displaced.

Cumulative averages of assigned grades compare to those of bulk sampling as follows:

	FEET	CROSSCUT		ASSIGNED	
		% Cu.	% Mo.	% Cu.	% Mo.
East	248	.214	.0074	.239	.0097
West	1426	.508	.0170	.454	.0184
AVERAGE	1674	.465	.0156	.423	.0168

The crosscut bulk samples are higher in copper than the assigned by 9.9%, an amount slightly greater than, but comparable to, that of the flat drilling above the assigned, i.e. 8.7%.

Molybdenum in the crosscut is about 7% less than that expected. This result is contrary to that of the flat drill holes which indicated a grade equal to that assigned. All or part of this discrepancy may be due to the inclusion in the cumulative assigned average grade of a section of 73 feet of 0.184% molybdenum. Three ten-foot samples in this section average 0.307% molybdenum. As noted earlier the crosscut showed an increase of about 18% in molybdenum over that in the flat holes.

North and South Drives

Results of the sampling of these two headings are not compared to assigned grades as both were advanced nearly parallel to or along the grade trends. It should be noted, however, that the south drive, due south and parallel to trends, showed a marked uniformity of grade over its full length.

Shaft

No comparison of grades in the shaft with those assigned to zones is attempted but comparison of the overall averages to those of the shaft hole No. 36 are as follows:

	Feet	% Cu.	% Mo.
Shaft	33 - 550	.500	.0312
Surface Hole	39 - 550	.455	.0228

The shaft copper grade is 9.9% higher than that in Hole 36. The molybdenum is 37% higher.

7.5.4 Grade Trends

The assay results of the underground drill holes and crosscuts have been grouped according to pattern and magnitude and the average assays of these sections are shown against the hole or crosscut on the plan of the test area (Dwg. No. 121) and in Table IV. These results have been reinterpreted to produce grade trends on the same basis as those from surface holes.

The emerging pattern shows a reasonable agreement with the earlier one in both the relative magnitudes of adjacent or continuous zones and in the deviation in strike direction of the two main high zones.

The pattern varies, however, in some respects - notably in an apparent displacement of some zones and a greater or lesser continuity in others. These discrepancies are due in part to the probable mislocation of original zones because of deviation of surface holes and to the fact that the crosscut is not exactly at the 4552 foot elevation on which assigned grades were measured. The pattern reasonably substantiates that forecast from surface drill holes.

7.5.5 Conclusions

The compilations and assessment of the sampling by crosscutting and flat hole drilling lead to the following conclusions:

(1) The area tested is typical of a large part of the overall deposit within the proposed pit. The work underground gave no evidence that the crosscut sampled an unusual part.

(2) Flat drill holes give results that for copper are similar to bulk sampling but lower in the case of molybdenum.

(3) Flat holes and the crosscut clearly imply that surface drilling has not over-valued the copper content of the area tested. Copper grades by underground work in the test area are about 9% above that indicated by surface drilling.

(4) The behaviour of molybdenum in the various methods of sampling is inconsistent but the grade established from surface is reasonably upheld. A full understanding of the distribution and content of molybdenum will probably not be acquired from operations lesser in scale than actual mining.

(5) The substantiation of the grade trends and grades, though in part vague, provides a sufficient basis for the calculation of an ore reserve. Any grades so calculated are representative, needing no revision in the light of underground results.

SECTION V111 - MINERAL RESERVES

8.1 GENERAL

Calculations of the reserves within the proposed Pit 15, have been calculated by two methods to give respectively -

(i) An in situ reserve and,

(ii) a mill feed reserve.

Both methods are based on the same data, i.e. copper grade trends derived from surface drill holes. The methods of estimation, factors used and the tonnages and grades established are described and compared below.

A separate estimate is made for the oxide zone.

Prior to the development of Pit 15 an estimate of the total ore grade material indicated by surface drilling was made. This was at a cut-off of 0.25% copper.

342,000,000 tons 0.424% Cu .0126% Mo

Deep drilling has shown no diminution of grade and the overall structure continues and is open beyond the bottom of the presently planned pit.

8.2 IN SITU RESERVE

The in situ reserve is based directly on grade trends for copper. The estimate includes only the sulphide zone.

This reserve represents tonnages and grades of all material, defined into various categories by cut-off grades, occurring, in situ, between limits imposed by trend lines. It incorporates no dilution factors nor compositing effects that might be introduced by mining practices.

This estimate is confined to material above cut-off grades and to waste defined as internal. No tonnage for the overall amount of waste rock within Pit 15 is produced and hence no waste to ore ratio.

For the requirements of Mine Planning a grade of molybdenum was calculated for the material mined in the first mining period (Report No. 4, Mining and Production Plans).

8.2.1 Factors in Estimation

The following are factors and assumptions used in the calculation of the in situ reserves of the Lornex sulphide deposit:

(1) Pit limits to which calculations were made and which are shown on plans and sections in this report are those of Pit 15, supplied by Mine Planning.

(2) Cut-off grade. The reserve estimate is based on the division of material into four categories:

(A) Ore - 0.30% Cu. and greater

(B) Ore - 0.29 - 0.26% Cu.

(C) Sub-ore - 0.25 - 0.20% Cu.

(D) Waste (internal) - 0.19% Cu. or less

The final reserve is based on a cut-off grade of 0.26% Cu.

(3) Internal waste is defined as low-grade material that in plan occurs within the pit and,

(a) has material of categories A, B or C between it and the pit limit along the same line of section or,

(b) is contiguous with blocks of category A, B or C on any three sides.

(4) Ore or waste classification is based solely on copper content - no consideration being given to the molybdenum content or to its copper equivalent on any basis.

(5) All assays used in averages for copper and molybdenum are uncut.

(6) Copper assays used are based on total copper content.

(7) Specific Gravity

Specific gravity determinations of the Lornex mineralized material were made at the Lornex pilot plant, Coast Eldridge Engineers and Chemists Ltd., and by the Lornex geological department.

Lornex Pilot Plant

Specific gravity of the cyclone overflow was determined for nine mill lots. The mean specific gravity was 2.64.

Coast Eldridge Engineers and Chemists Ltd.

Saturated specific gravities of 23 diamond drill core specimens from holes 8 and 9 were determined by the loss of weight in water method. The mean specific gravity determined was 2.58.

Lornex Geological Department

The specific gravities of 975 diamond drill core specimens were determined using the loss of weight in water method. The core specimens were taken at approximately 100-foot intervals in all drill holes.

To arrive at an average specific gravity for the ore, the results from 45 holes were used for a total of 675 determinations. The average specific gravity obtained was 2.58.

To arrive at an average specific gravity of the waste wall rock a total of 11 holes or 92 determinations were used. The average specific gravity of the wall rock was 2.60.

Using the same method, the specific gravities of 18 hand specimens from the east crosscut and 29 specimens from the west crosscut were determined. The average specific gravity for the east crosscut was found to be 2.59. The average specific gravity of the specimens from the west crosscut was 2.57.

The average results obtained by Coast Eldridge and the Lornex geological department were identical at 2.58. This should give the bulk of density of the ore in place. For the reserve estimate the bulk density of 2.58 was accepted and the factor 12.4 cubic feet to 1 short ton (2000 lbs.) used.

The method used by the pilot plant should provide a true specific gravity of the solids in the ore body and is consequently higher at 2.64.

(8) The quoted tonnage and grade of the reserves are defined as Drill Indicated and further as the In Situ reserve.

(9) All tonnages are in short tons (2000 lbs.).

8.2.2 Method of Calculation

Reserve calculations were based on grade trend plans of levels at 80-foot vertical intervals from the 3752-foot (No. 18 level) to the 5112-foot elevation (No. 1 level). Zones on these plans were divided into blocks by lines half way between surface drill hole lines or in the case of lines 5S and 35N continued 200 feet beyond the sections.

The limits of the Pit 15 were drawn on each plan and all blocks within these limits designated as one of the four measured categories depending on the grade shown on the plan.

Tonnage for each block was calculated by measuring the area by planimeter and projecting each block vertically 40 feet above and below the level; except in the case of the 3752 level where only the 40 feet above was calculated.

Limits of oxide and overburden were shown on each plan and measurements made to these limits.

No attempt was made in this estimate to calculate the tonnage or grade of waste other than that defined above as internal waste.

Each block was numbered according to the section on which it was based.

An example of the layout and numbering of blocks is shown on a plan of the 4552 elevation plan (Dwg. No. 130).

All blocks were tabulated by section and plan according to their copper value and total reserves calculated by adding tonnages and calculating weighted averages for copper and molybdenum grades.

8.2.3 In Situ Reserve Estimate

The following are the drill-indicated in situ reserves calculated to occur within Pit 15:

CATEGORY	% Cu.	TONS	TOTAL % Cu.	% Mo.
A	0.30 & greater	251,100,000	.462	.0152
B	0.29 - 0.26	22,900,000	.271	.0091
C	0.25 - 0.20	38,700,000	.226	.0066
D	0.19 and less	46,200,000	.135	.0040
TOTAL ORE	CUT-OFF			
A + B	0.26% Cu.	274,000,000	.447	.0147

For Mine Planning molybdenum grades are calculated as follows:

1st Mining Period - 0.0176% Mo.

From this and the total in situ reserve the single grade figure for the ensuing periods is:

2nd, 3rd and 4th Periods 0.0136% Mo.

8.3 MILL FEED GRADE RESERVE

A reserve which by its method of calculation introduced dilution was calculated on a basis of 80-foot cubes. The inherent grade dilution effect of this method of treatment would compensate to some extent for the various effects which tend to cause a mill feed grade to be less than an in situ grade.

This calculation included all material within the pit limits including waste rock of all categories and enables a waste to ore ratio to be derived. A tonnage was calculated for the oxide zone including weathered rock beyond the confines of the deposit but no grade was assigned to this.

The calculations in this estimate produced only a copper grade. Molybdenum grades are derived by adjustments of those of the in situ reserve.

Treatment of the 80-foot cubes was done twice, by computer and manually. Differences were insignificant. The figures quoted here and in Report No. 4, Mining and Production Plans, are those from the manual count.

8.3.1 Factors in Estimation

Factors used are generally as used in the in situ reserve estimate (8.2.1):

(1) Pit limits are those of Pit 15.

(2) Cut-off grade. Material was divided into three categories:

(a) Ore - cut-off 0.26% Cu.

(b) Oxide zone.

(c) Waste. 0.25% Cu. and less

(3) Internal waste was included in 2(c) above.

(4) Estimate was based on copper content only. No grade for molybdenum was calculated.

(5) All assays were uncut.

(6) Copper assays used are based on total copper content.

(7) Specific gravity. A factor of 12.4 cubic feet to one short ton was used for ore or waste - 13.0 cubic feet to one short ton for the oxide zone.

(8) The quoted tonnages and grades are defined as Drill Indicated and the Mill Feed Grade reserve.

(9) All tonnages are in short tons.

8.3.2 Method of Calculation

Grids of squares of 80 feet x 80 feet were superimposed on each of the copper grade trend plans at 80-foot vertical intervals. This grid was orientated at N 15° W, approximating the general strike of the main mineralized zone.

A single copper grade was then calculated for each square from the copper grade trends. This value was the area weighted average grade of each of the zones falling within the square under consideration. An example of the 80-foot square layout and calculation of grades is shown (Dwg. No. 131). The value derived for the square was considered as applying to the ground vertically 40 feet above and 40 feet below the area represented.

Squares and blocks were formed in this manner to include all the material within Pit 15.

Blocks were then counted level by level and, dependent on its grade, a block was allocated to one of the various categories including oxide material, waste, etc. Total tonnages and average grades of these categories were then calculated.

The example (Dwg. No. 131) shows clearly the effect of smoothing of grades inherent in this method and hence the inclusion of material of waste grade in blocks with an overall above cut-off grade. This has resulted in the smoothing out of most areas of waste within the confines of the deposit as a whole and a tonnage of ore grade material greater than that for the in situ reserve but having a lower average copper grade.

8.3.3 Mill Feed Reserve Estimate

Calculated by the 80-foot cubes the materials within Pit 15 are estimated as:

Ore	-	Cut-off 0.26% Cu.
		292,800,000 tons - 0.427% Total Cu.
Waste Rock	-	0.25% Cu. and less
		217,200,000 tons - No grades
Oxide Zone	-	35,300,000 tons - No grades
Waste Ore Ratio	-	0.862:1

Molybdenum Grade - No grade was calculated by the 80-foot cube method for the molybdenum in the above ore or waste rock but, is derived for the ore as shown below.

Copper grades of the in situ and mill feed reserves compare as follows:

In Situ	-	274,000,000 tons	-	.447% Cu.
Mill Feed	-	292,800,000 tons	-	.427% Cu.

The mill feed copper grade is 4.5% lower than that of the in situ estimate.

This reduction applied to the in situ molybdenum grade of .0147% Mo. provides a grade of .0140% Mo. This latter value is taken as the mill feed molybdenum grade and is used in Report No. 4, Mining and Production Plans.

The ore reserve is thus stated as:

292,800,000 tons - 0.427% total Cu. - 0.0140% Mo.

Similarly molybdenum grades for the mining periods are calculated as:

1st Mining Period	-	.0168% Mo.
2nd, 3rd and 4th Mining Periods	-	.0130% Mo.

8.4 OXIDE ZONE

An oxidized capping of varying thickness exists over the entire mineral deposit. The average thickness of this capping over the eastern part of the deposit is about 200 feet. To the west it thins out and over the western portion of the deposit it is normally less than 50 feet thick.

8.4.1. Factors and Method of Calculation

To delimit this zone, results from percussion drilling, Becker drilling, diamond core drilling and a seismic survey were used. The top of the oxidized zone was reasonably well defined by the percussion and Becker drill results. Down-the-hole salting in the percussion drill method, however, made definition of the base of the zone uncertain. Whenever possible the base of the zone was thus determined from diamond drill core results.

Because of the poorly defined base of the zone the lower limit was considered to occur where the soluble copper content dropped to less than 20% of the total copper. This was subject to the inaccuracies of the sampling method.

To calculate the tonnage of material in this zone, it was divided into three categories:

(1) Greater than 50% soluble, above cut-off of 0.25% Cu.

(2) 20% to 50% solubles

(3) Greater than 50% solubles below cut-off of 0.25% Cu.

The areas falling within these categories were measured on each section of drilling and total tonnage calculated by using the end area method and a factor of 13 cubic feet to the ton.

The tonnage included lies between Sections 3S and 33N and only over those sections of the sulphide deposit which are considered to be of ore grade. It does not include, therefore, all rock which has been oxidized. The addition of further oxidized material in these estimates would simply transfer tonnage from a waste rock category to oxidized waste rock.

8.4.2 Estimate

Tonnage and grade arrived at was as follows:

(1) Greater than 50% soluble. Cut-off 0.25% total Cu.

13,086,378 tons - 0.50% total copper and 0.38% soluble copper.

(2) 20% to 50% soluble with no cut-off:

4,814,000 tons - 0.48% total copper and 0.135% soluble copper.

(3) Greater than 50% soluble but less than 0.25% total copper:

3,583,660 tons

No grade has been assigned to this category since its apparent low grade may be due to dilution of the samples from the overburden above.

8.5 CONFIDENCE LIMITS

8.5.1 Confidence Levels and Checks

Within the limitations of independence imposed by the use of the same base data (i.e. surface diamond drill holes) two independent methods of calculation were used as general checks on the ore reserves previously reported. Both checks were carried out with the assistance of a large capacity digital computer.

The first check had a twofold aim; this was:

i) To obtain confidence levels for the 'within-pit-10 foot interval' sample data for material above a 0.26% Cu. cut-off grade.

ii) To obtain a 'within-pit' average value for all 10 foot samples above a 0.26% Cu. cut-off. This value should approximate to the 'in situ' grade previously reported. However, one might expect the 10 foot sample data to give a slightly higher average value due to the greater precision in locating the cut-off point.

The second check had a single aim; this was:

i) To obtain an independent method estimate of the 'within pit' average grade for the 80-foot block valuation. This was accomplished by utilizing a statistical method to fit values to eighty foot centres throughout the ore zone. The accuracy of the method is to two decimal places.

Results

1. Confidence levels:

± 0.009% Cu. to a 97.5% confidence level for the 'within-pit base data (10 foot sample interval)' for material above a 0.26% cut-off.

2. In situ grade check:

Arithmetic mean (above a 0.26% Cu. cut-off), of the 10 foot samples data lying within the final pit:-

0.456% Cu. ± 0.009% Cu. for a 97.5% confidence limit.

3. 80-foot block mill feed grade check:

Statistical zoning programme, mean value, rounded, 0.42% Cu. ± 0.01% Cu. for a 97.5% confidence limit.

8.5.2 Conclusions

There is no significant difference between the geological grade trend estimate of the 'in situ' grade and the simple arithmetic mean check. Similarly, there is no significant difference between the geological and statistical estimates of the 80-foot block based estimates of the mill feed grade.

8.6 SUMMARY OF RESERVE ESTIMATE

Two methods of calculation give the following estimates of the reserves within Pit 15:

1. Calculated from grade trend blocks.

IN SITU RESERVE	TONS	% Cu.	% Mo.
A. Ore 0.30% Cu. and greater	251,100,000	.462	.0152
B. Ore 0.29% - 0.26% Cu.	22,900,000	.271	.0091
TOTAL A & B 0.26% Cu. and greater	274,000,000	.447	.0147
C. Sub-ore (Stockpile) 0.25% - 0.20% Cu.	38,700,000	.226	.0066
D. Internal Waste 0.19% Cu. and less	46,200,000	.135	.0040

2. Calculated by 80-foot cubes.

MILL FEED RESERVE	TONS	% Cu.	% Mo.
A. Ore 0.26% Cu. and greater	292,800,000	.427	.0140
B. Waste (including oxide)	252,500,000	No Grades	

Waste To Ore Ratio - 0.862:1

TABLE 1

WEST CROSS CUT - CHECK ASSAYS

RND NO.	% Cu. LORNEX	% Cu. COAST ELDG.	% Cu. ELLIOT LAKE	% Cu. LORNEX (2)	% Mo. LORNEX	% Mo. COAST ELDG.	% Mo. ELLIOT LAKE	% Mo. LORNEX (2)
1	0.21	0.21			.007	.011		
2	0.23	0.22			.012	.012		
3	0.18	0.18			.010	.010		
4	0.26	0.26			.008	.007		
5	0.23	0.23			.009	.007		
6	0.55	0.57			.027	.022		
7	0.51	0.60	0.49	0.51	.019	.015	.015	
8	0.46	0.44			.018	.017		
9	0.51	0.57	0.50	0.50	.027	.028	.026	.026
10	0.27	0.25			.024	.020		
11	0.41	0.32	0.39	0.43	.018	.022	.021	.024
12	0.35	0.37			.022	.017		
13	0.40	0.44	0.39	0.40	.037	.029	.031	.030*
14	0.43	0.47		0.41	.027	.025		
15	0.78	0.77	0.73	0.78	.081	.127	.140	.133*
16	0.48	0.45			.032	.027		
17	0.53	0.54	0.50	0.54	.075	.085	.095	.101
18	0.35	0.37			.022	.021		
19	0.52	0.54			.066	.064		
20	0.43	0.50	0.43	0.43	.047	.035	.038	.039*
21	0.88	0.94	0.80		.090	.089		
22	0.58	0.65	0.57		.028	.031		
23	0.50	0.55	0.47		.021	.022		
24	0.97	1.07	0.91		.056	.063		
25	0.84	0.91	0.78		.058	.077		
26	0.30	0.34	0.28		.013	.014		
27	0.32	0.37	0.29		.015	.016		
28	0.74	0.72			.042	.088		.101*
29	0.58	0.59			.059	.059		
30	0.79	0.80			.038	.067		.081*
31	0.55	0.50	0.50	0.53*	.030	.030		
32	0.27	0.32	0.25		.022	.021		
33	0.48	0.49			.090	.087		
34	0.80	0.77			.052	.072		.085*
35	0.78	0.79			.035	.067		.075*

	% Cu.				% Mo.			
RND NO.	LORNEX	COAST ELDG.	ELLIOT LAKE	LORNEX (2)	LORNEX	COAST ELDG.	ELLIOT LAKE	LORNEX (2)
36	1.08	1.09			.043	.049		.044
37	0.90	0.91	0.90		.035	.027	.024	.029*
38	1.18	1.21			.048	.046		
39	0.34	0.39	0.31		.012	.013		
40	0.46	0.49			.014	.019		
41	1.06	1.07	1.00		.018	.028	.028	.034*
42	1.65	1.53			.053	.052		
43	1.32	1.26			.055	.064		.067*
44	0.85	0.84			.050	.053		
45	0.40	0.40			.030	.031		

* Assay substituted for the original

TABLE 11

AVERAGE ASSIGNED GRADES ACROSS MAIN MINERALIZED ZONE

PLAN 4552 ELEVATION

SECTION	WIDTH FT.	% Cu.	% Mo.
35 N	Not calculated - overburden		
31 N	778	.299	.024
27 N	1420	.368	.010
23 N	1506	.400	.011
19 N	1660	.468	.011
15 N	1760	.415	.013
11 N X Cut	1482	.445	.019
7 N	1311	.435	.017
3 N	1055	.454	.016
1 S	955	.365	.015
5 S	1055	.288	.010

PLAN 4872 ELEVATION

11 N	1298	.448	.015

PLAN 4232 ELEVATION

11 N	1580	.429	.015

TABLE 111

GRADE COMPARISON

WEST CROSS CUT vs UG. DDH'S UG-1, 2 & 6

West Cross Cut						UG. DDH		
FROM	TO	FT.	ROUNDS	% Cu.	% Mo.	NO.	% Cu.	% Mo.
96	603	507	12-105	.535	.0255	1	.514	.0216
	(a)							
633	910	277	112-174	.435	.0122	2	.407	.0075
	(b)							
928	1032	104	180-203	.534	.0102	2	.491	.0112
1032	1426	394	204-292	.570	.0130	6	.605	.0119
Average		1282		.524	.0174		.516	.0147

NOT INCLUDED

(a) Gap between UG-1 and 2

603	633	30	106-111	.367	.0058	Not drilled	

(b) Vein in UG-2

910	928	18	175-179	.529	.0129	2.175	.0073

TABLE 1V

GRADE COMPARISON BY ZONES

CROSS CUT vs UG. DDH'S UG-1, 2 & 6

		CROSS CUT			UG. DDH	
FROM	TO	FT.	% Cu.	% Mo.	% Cu.	% Mo.
0	52	52	.238	.0104	Not Drilled	
52	96	44	.446	.0221	Not Drilled	
96	163	67	.512	.0515	.480	.0726
163	224	61	.571	.0446	.516	.0300
CUMULATIVE		128	.540	.0482	.493	.0545
224	290	66	.881	.0508	1.161	.0350
CUMULATIVE		194	.656	.0491	.705	.0474
290	311	21	.370	.0127	.283	.0071
CUMULATIVE		215	.628	.0455	.667	.0436
311	377	66	.503	.0118	.503	.0060
CUMULATIVE		281	.600	.0374	.631	.0348
377	426	49	.468	.0109	.262	.0047
CUMULATIVE		330	.579	.0336	.576	.0303
426	562	136	.466	.0101	.393	.0081
CUMULATIVE		466	.546	.0268	.521	.0221
562	603	41	.401	.0079	.403	.0029
CUMULATIVE		507	.535	.0252	.512	.0221
603	633	30	.367	.0058	.367*	.0058*
CUMULATIVE		537	.525	.0242	.503	.0212
633	677	44	.427	.0080	.361	.0036
CUMULATIVE		581	.518	.0231	.492	.0199
677	720	43	.517	.0131	.502	.0056
CUMULATIVE		624	.518	.0224	.495	.0189
720	802	82	.391	.0091	.315	.0027
CUMULATIVE		706	.503	.0208	.474	.0170

		CROSS CUT			UG. DDH	
FROM	TO	FT.	% Cu.	% Mo.	% Cu.	% Mo.
802	845	43	.443	.0126	.624	.0084
CUMULATIVE		749	.500	.0204	.482	.0165
845	910	65	.436	.0171	.342	.0166
CUMULATIVE		814	.494	.0201	.471	.0165
910	958	48	.546	.0104	.551+	.0167+
CUMULATIVE		862	.497	.0194	.475	.0165
958	1,067	109	.517	.0102	.468	.0092
CUMULATIVE		971	.500	.0185	.475	.0157
1,067	1,228	161	.626	.0115	.706	.0119
CUMULATIVE		1,132	.518	.0175	.508	.0152
1,228	1,269	41	.454	.0132	.432	.0076
CUMULATIVE		1,173	.515	.0174	.505	.0149
1,269	1,400	131	.592	.0173	.610	.0142
CUMULATIVE		1,304	.523	.0174	.516	.0148
1,400	1,426	26	.398	.0045	.455	.0027
CUMULATIVE		1,330	.521	.0171	.514	.0146
1,426	1,487	61	No Cross cut		.553	.0051
CUMULATIVE		1,391				

* 603 - 633 Section not drilled - Cross cut average substituted for continuity

\+ 910 - 958 Includes vein section for which cross cut assays are substituted

TABLE V

GRADE COMPARISON

UG. DDH vs ASSIGNED

SECTION NO.	DDH	FT.	UG. DDH % Cu.	UG. DDH % Mo.	ASSIGNED % Cu.	ASSIGNED % Mo.
15 N	UG 8	550	0.338	0.0054	0.315	.0129
	UG 7*	604	.503	.0143	.447	.0161
	CUMULATIVE	1,154	.423	.0101	.384	.0146
13 N	UG 3*	560	.366	.0241	.378	.0203
	UG 4	580	.400	.0178	.414	.0140
		1,140	.384	.0209	.397	.0171
	CUMULATIVE	2,294	.404	.0155	.391	.0158
11 N	UG 1	506	.514	.0216	.480	.0299
	UG 2+	430	.429	.0082	.415	.0131
	UG 6	457	.597	.0109	.485	.0080
		1,393	.515	.0140	.456	.0175
	CUMULATIVE	3,687	.446	.0149	.416	.0165
9 N	UG 5	565	.482	.0257	.432	.0203
	CUMULATIVE	4,252	.450	.0164	.418	.0170
7 N	UG 10	552	.466	.0255	.448	.0182
	UG 9*	636	.509	.0077	.428	.0117
		1,188	.489	.0160	.437	.0147
	Cumulative	5,440	.459	.0163	.422	.0165

* Includes drill hole station

\+ Includes 30 foot section of cross cut and substitution for high grade vein

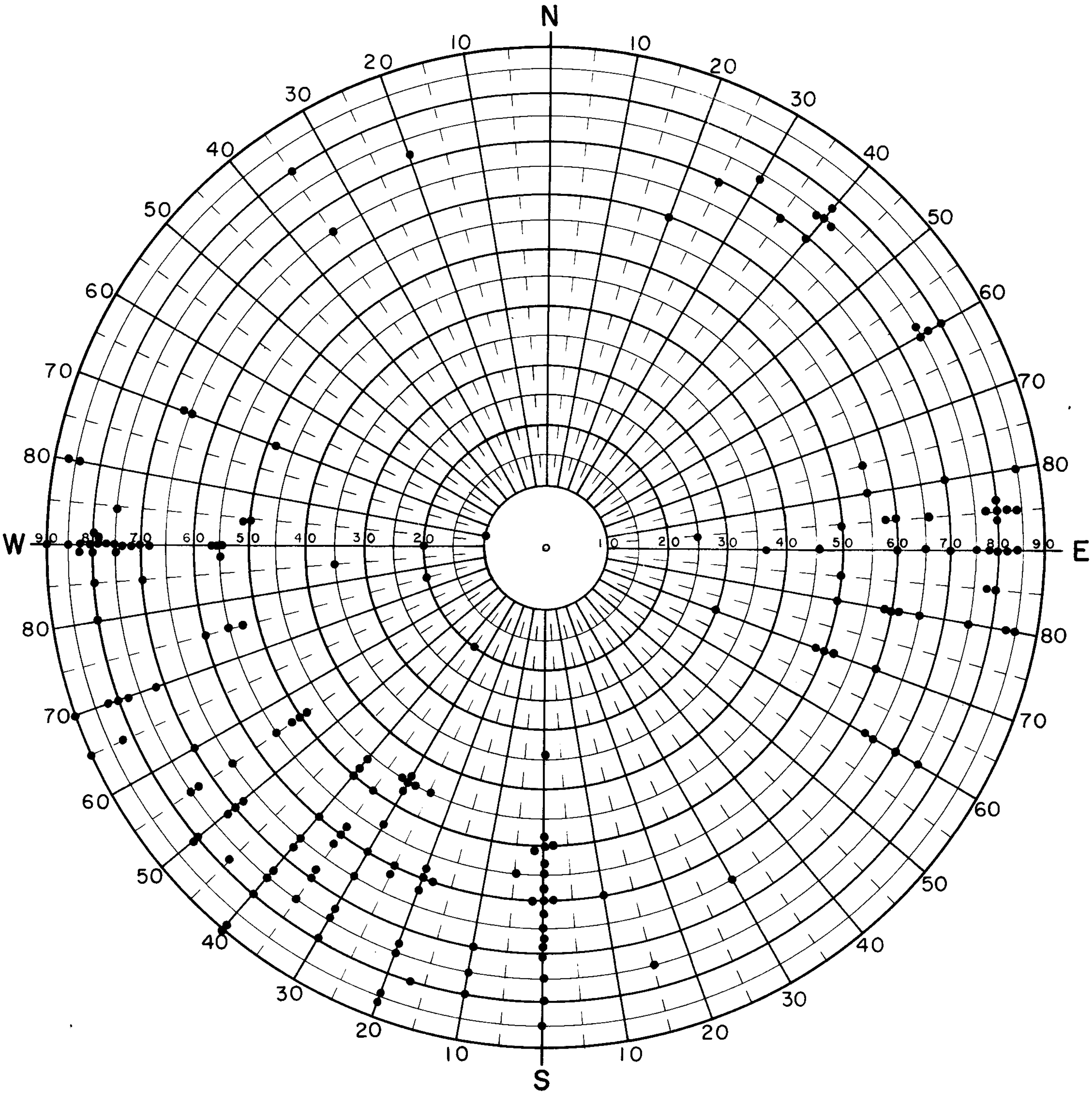


FIG. I

POLAR EQUAL AREA STEREONET

UPPER HEMISPHERE

SHOWING POLES OF FAULTS
IN WEST CROSS - CUT

FIG. 2

FLOWSHEET OF CRUSHING & SAMPLING PLANT

TRUCKS FROM MINE

STORAGE BAYS (1)

FRONT END LOADER (2)

FEED HOPPER OF UNIVERSAL PACEMAKER PORTABLE CRUSHING PLANT MODEL 2930 SET TO CRUSH TO (3)

(3/8)

24" SNYDER SAMPLER (4)

(90%) → # 6 CONVEYOR (13)

(10%)

12 x 24" VIB. FEEDER (5)

12 x 8" ROLLS (6)

3 TPH BUCKET ELEVATOR (7)

14" SNYDER SAMPLER (8)

(90%) → # 6 CONVEYOR (13)

(10%)

8 x 24" VIB. FEEDER (9)

10 x 6" ROLLS (10)

8 x 24" VIB. FEEDER (11)

CONTINUOUS DENVER SAMPLER (12)

(90%) → # 6 CONVEYOR (13)

(10%)

0.10% SAMPLE

TO SAMPLE PREPARATION

6 CONVEYOR (13)

STORAGE BAY

FRONT END LOADER TO MILL BIN OR STOCKPILE

LEGEND TO CRUSHING & SAMPLING FLOWSHEET

1. Six storage bays and a paved area were provided for intermediate storage of shaft rounds ahead of crushing and sampling where each round was treated as a Lot.

2. A 1.5 yard Caterpillar front end loader was used to deliver ore from storage to the crushing plant and to move fine ore after sampling to the mill hopper or storage.

3. This portable plant included an 18 x 24" Jaw crusher, two 24" rolls, two 3' x 8' vibrating screens together with reciprocating feeder, conveyors, etc. It is driven by a 100 HP motor and protected by a suspended electro magnet. Capacity is 20 TPH.

4. 24" Simplex Snyder sampler with two cutters designed to cut 10% cut. 1/2 HP gear motor.

5. 12" x 24" vibrating feeder to smooth out feed to rolls crusher.

6. 12" diameter x 8" face crushing rolls; 5 HP motor with V belt drive.

7. Bucket elevator was used to regain elevation.

8. 14" Simplex Snyder sampler with two cutters designed to give 10% cut. 1/2 HP gear motor.

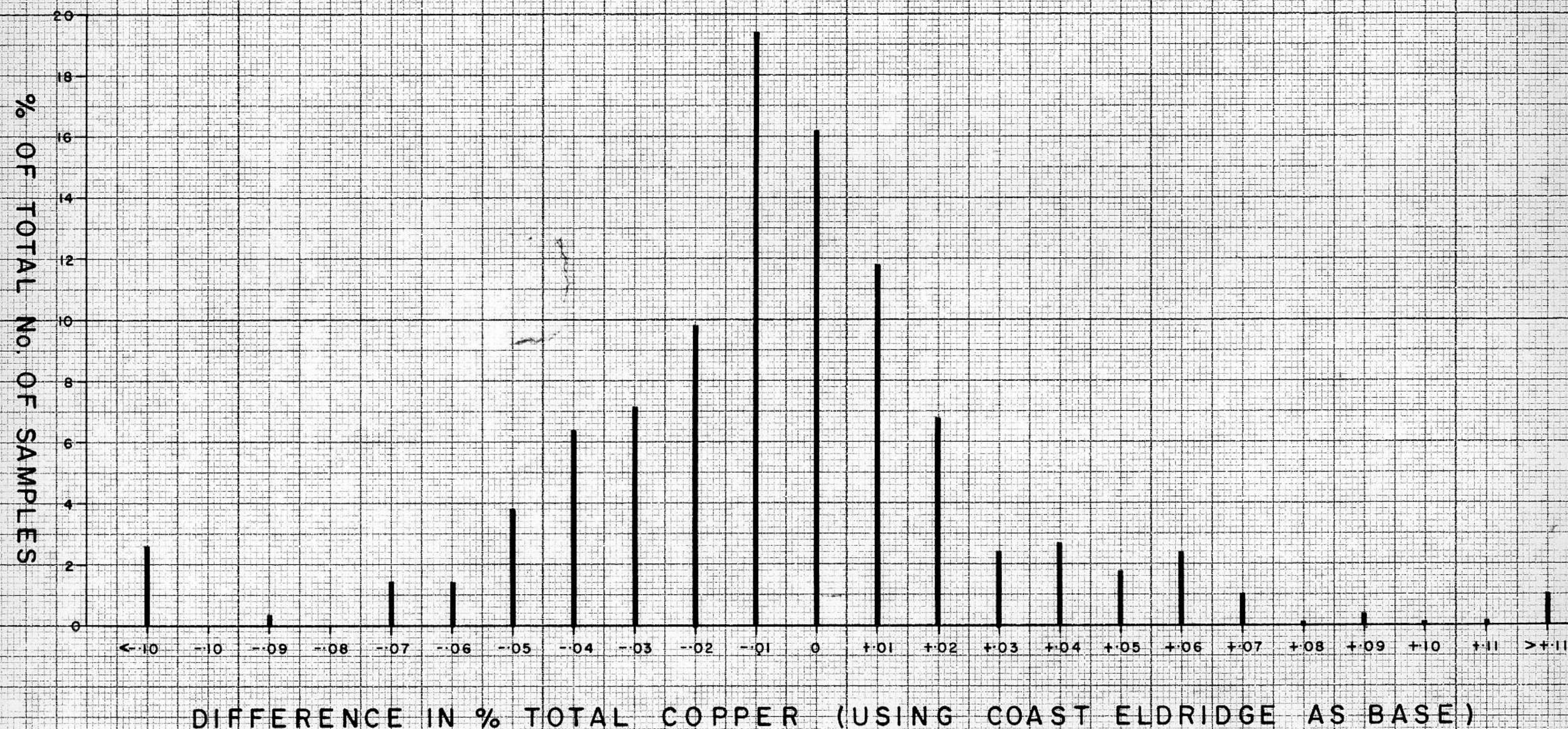
9. Same as (5) above but 8" wide.

10. 10" diameter x 6" face crushing rolls; 3 HP motor with V belt drive.

11. Same as (9) above.

12. Denver continuous reciprocating type sampler. Cutter opening was adjusted to give 10% cut.

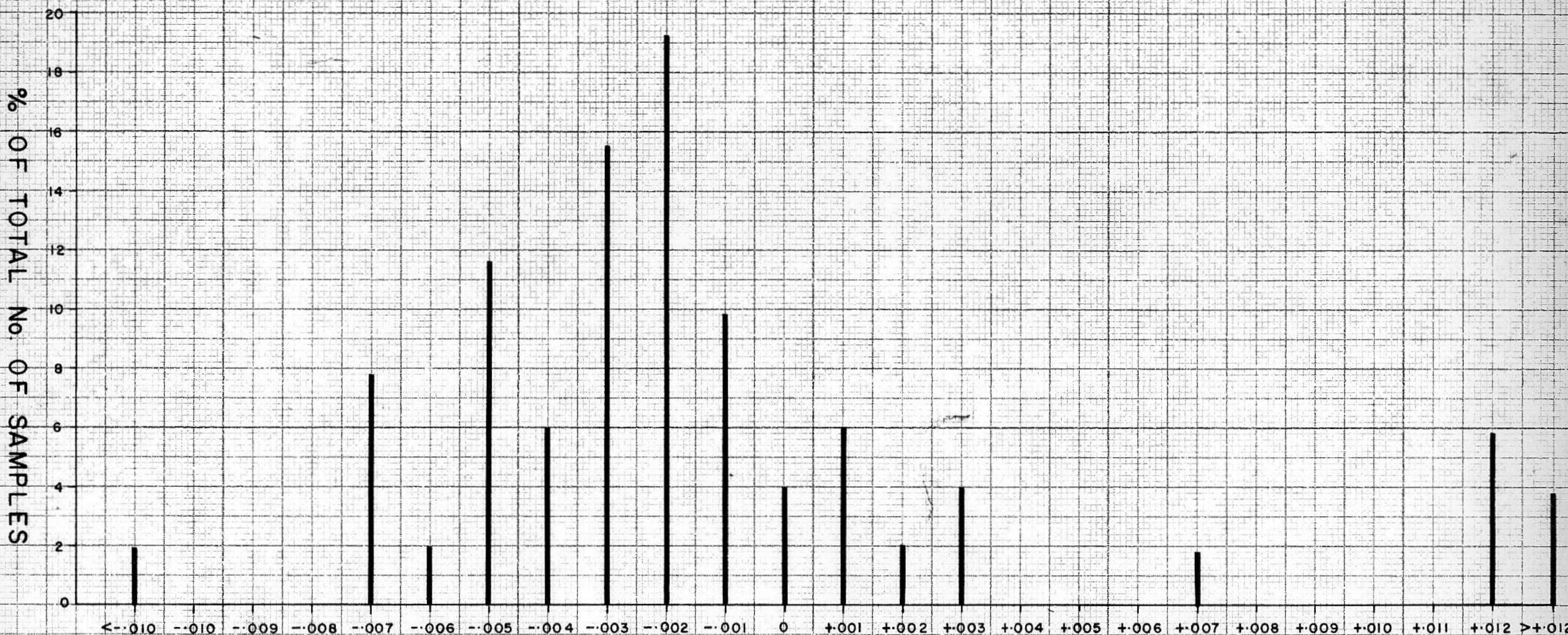
13. No. 6 Conveyor, 18" wide and 27' long picked up the reject products and delivered them to a concrete storage bay for pick-up by front end loader (2) above.



LORNEX — COAST ELDRIDGE

(296 ASSAYS) HOLES 42 - 82 INCLUSIVE

FIG. 3

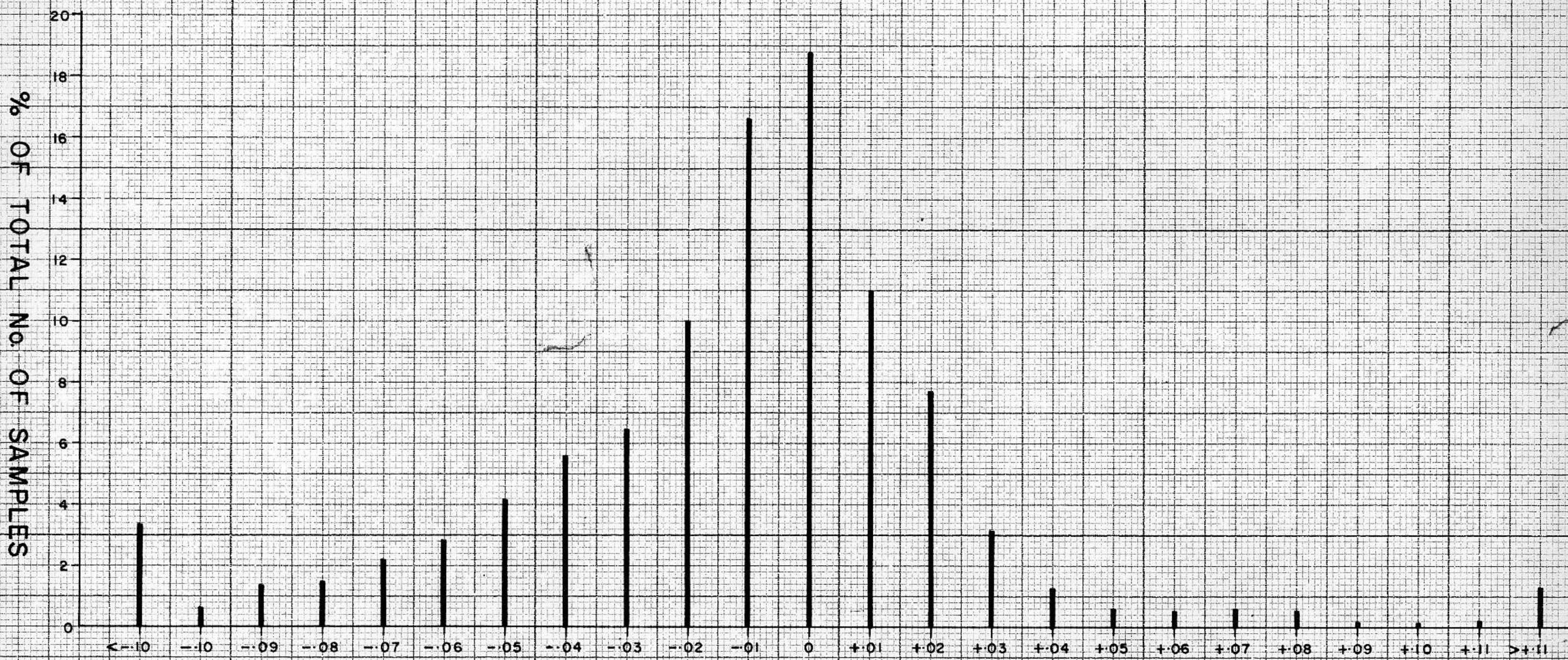


DIFFERENCE IN % M_oS_2 (USING COAST ELDRIDGE AS BASE)

LORNEX – COAST ELDRIDGE

(52 ASSAYS) HOLES 65 – 82 INCLUSIVE

FIG. 4

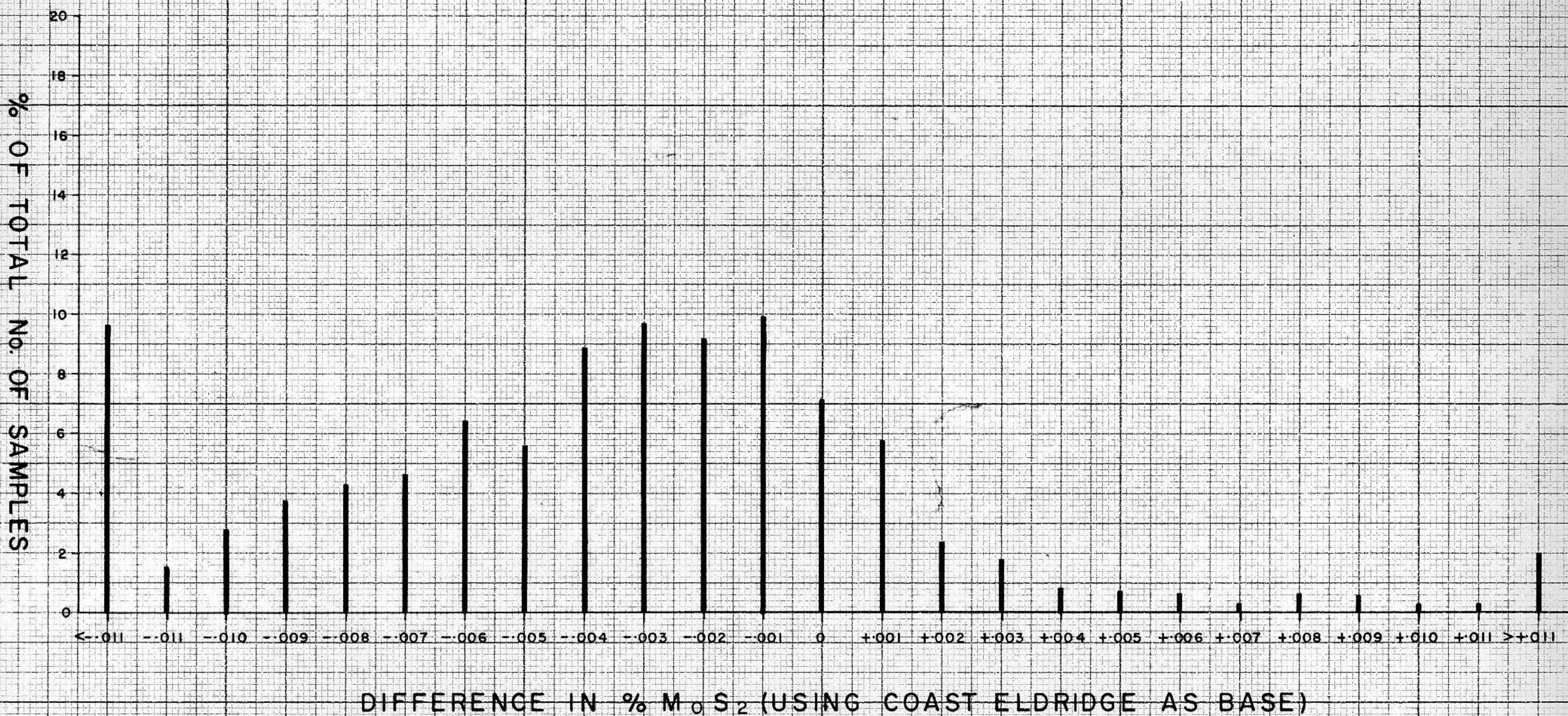


DIFFERENCE IN % TOTAL COPPER (USING COAST ELDRIDGE AS BASE)

ELLIOT - LAKE - COAST ELDRIDGE

(860 ASSAYS) HOLES 8 - 75 INCLUSIVE

FIG. 5



ELLIOT LAKE—COAST ELDRIDGE

(892 ASSAYS) HOLES 8 – 75 INCLUSIVE

FIG. 6

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.

VANCOUVER BRITISH COLUMBIA

LOCATION MAP

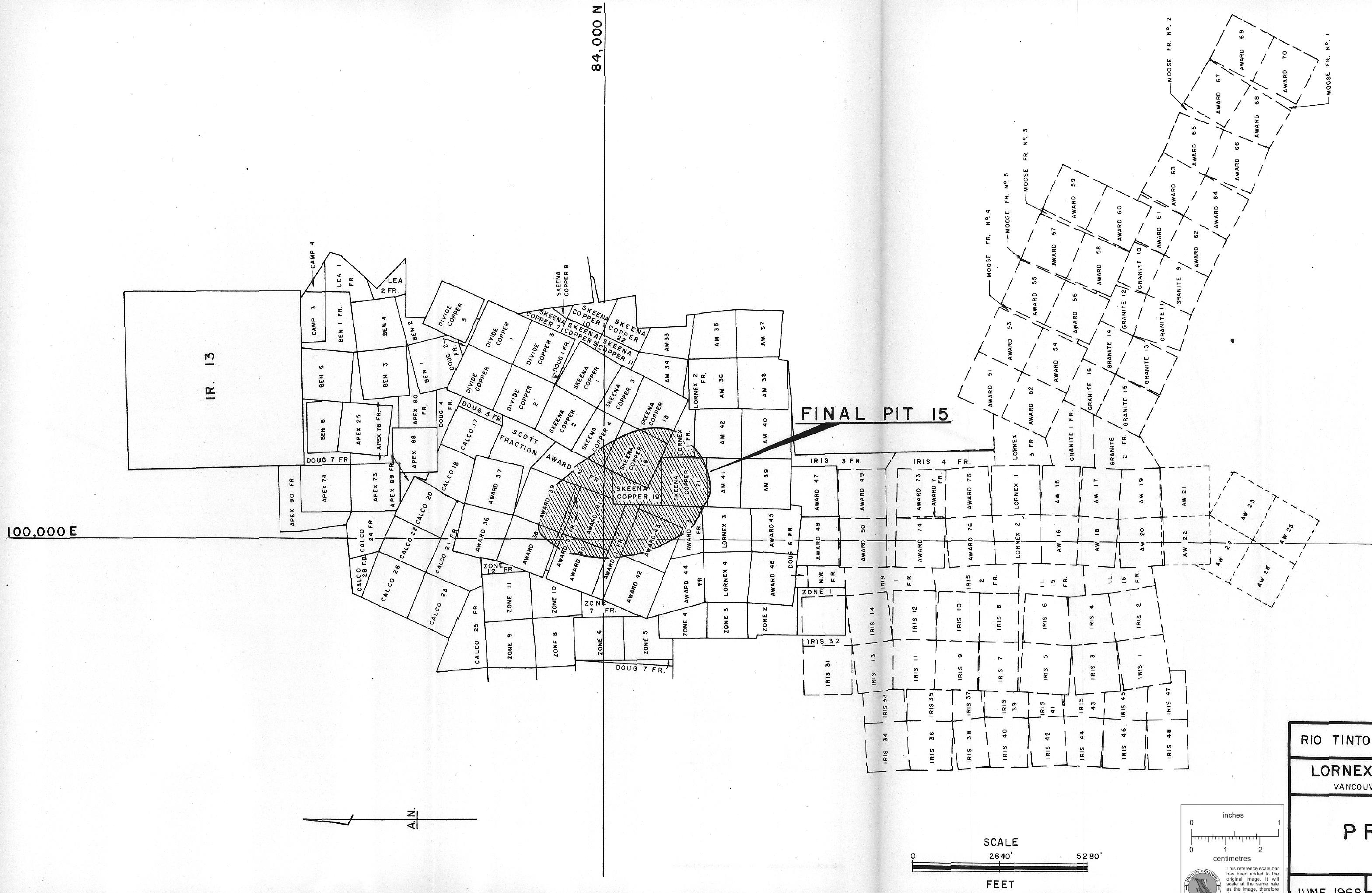
JUNE, 1968	JOB No.	VOL. I 101 DRAWING NUMBER	REV.

inches 0 1 2

0 1 2 3 4 5 centimetres

BRITISH COLUMBIA GEOLOGICAL SURVEY

This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.



inches
0 1
0 1 2
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.

RIO TINTO CANADIAN EXPLORATION LIMITED		
LORNEX MINING CORPORATION LTD. VANCOUVER BRITISH COLUMBIA		
PROPERTY MAP		
JUNE, 1968	JOB No.	VOL. I 102 DRAWING NUMBER	REV.


NORANDA OPTION
102,000 E
Prop. Bdry.
LEGEND:
B BETHSAIDA GRANODIORITE
Q QUARTZ PORPHYRY
Sa SKEENA QUARTZ DIORITE Alteration Zone
S SKEENA QUARTZ DIORITE
OUTCROP
CONTACT
PROMINENT FAULT - ASSUMED
TRENCH
PROMINENT JOINTING
PROMINENT SCHISTOSITY & SHEARING
ROAD
ROAD - SECONDARY
FINAL PIT
I.P. CONTOUR INTERVAL IN MILLISECONDS, USING 800 FT. SPACING & THREE ELECTRODE ARRAY.
LORNEX
AWARD CREEK FAULT
AREA RESERVED BY SKEENA
SKEENA OPTION
AWARD CREEK
84,000 N
NORTH ZONE
Property Bdry.
PURCHASE
SKEENA
SCALE
0' 1000' 2000'
FEET
TRINI LAKE
DISCOVERY ZONE
AWARD
CAMP ZONE
A.N.
LORNEX
Prop. Bdry.
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA
GENERAL GEOLOGY OF PROPERTY
JUNE, 1968
JOB No.
VOL. I 103
DRAWING NUMBER
REV.
inches
0 1
0 1 2
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.

L-35 N
L-31 N
L-27 N
L-23 N
84,000 N
L-19 N
L-15 N
L-11 N
L-7 N
L-3 N
L-1 S
L-5 S
L-11S
102,000 E
REF. LINE
PYRITE
TOTAL COPPER SULPHIDES
BORNITE LESS THAN 33 %
BORNITE BETWEEN 33 % & 50 %
BORNITE GREATER THAN 50 %
AWARD CREEK
FAULT
A.N.
SCALE
0' 300' 600'
FEET
inches
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.
BRITISH COLUMBIA GEOLOGICAL SURVEY
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
BORNITE - CHALCOPYRITE - PYRITE ZONING at 4552 FT. ELEVATION
JUNE, 1968
JOB No.
VOL. I 104
DRAWING NUMBER
REV.


54
68
41, .013
23, .007
33, .009
21, .005
22, .004
5,032'
4,552'
4,072'
3,592'
BETHSAIDA
SKEENA
REF. LINE
LEGEND
68
SURFACE DIAMOND DRILL HOLE
OVERBURDEN
OXIDE ZONE
CONTACT
.21, .005
% Cu., % Mo.
N.A.
NOT ASSAYED
FINAL PIT OUTLINE
inches
0
1
0
1
2
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.
SCALE
0'
200'
400'
FEET
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE 11 SOUTH
JUNE, 1968
JOB No.
VOL. I 105
DRAWING NUMBER
REV.

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA

VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE 5 SOUTH

JUNE, 1968	JOB No.	VOL. I 106 DRAWING NUMBER	REV.


81
55
37
83
34
R.D.H.-5
29
59
5,032'
4,552'
4,072'
3,592'
.27,.005
.24,.003
.27,.032
.51,.027
.07,.016
.10,.001
.12,.002
.23,.008
.35,.017
.71,.027
.45,.003
.23,.001
.34,.002
.15,.004
.05,.004
.62,.032
.23,.005
.14,.003
.23,.011
.49,.018
.34,.004
.30,.010
.36,.004
.13,.002
.16,.002
.51,.027
.19,.007
.24,.004
.30,.008
.62,.042
.14,.001
.13,.003
.46,.005
.25,.004
.19,.010
.32,.052
.20,.017
.41,.010
.32,.019
.21,.009
.29,.004
.31,.005
.19,.010
.41,.019
.19,.007
.40,.013
.29,.004
.57,.023
.36,.010
BETHSAIDA
SKEENA
REF. LINE
LEGEND
81
SURFACE DIAMOND DRILL HOLE
R.D.H.-5
ROTARY DRILL HOLE
OVERBURDEN
OXIDE ZONE
CONTACT
.23,.001
% Cu., % Mo.
N.A.
NOT ASSAYED
FINAL PIT OUTLINE
SCALE
0'
200'
400'
FEET
inches
centimetres
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE I SOUTH
JUNE, 1968
JOB No.
VOL. I 107
DRAWING NUMBER
REV.

50
28
84
25
27
26
41
5,032'
4,552'
4,072'
3,592'
BETHSAIDA
SKEENA
REF. LINE
·05,·001
·22,·005
·07,·007
·15,·004
·15,·008
·07,·005
·44,·044
·38,·011
·29,·006
·17,·005
·32,·014
·26,·012
·50,·025
·56,·026
·44,·022
·52,·036
·26,·012
·44,·005
·35,·004
·28,
·38,·004
·34,·004
·50,·006
·31,·008
·60,·005
·33,·005
·93,·005
·31,·004
·68,·009
·23,·005
·33,·005
·57,·011
·40,·004
·52,·011
·38,·010
·48,·008
·24,·003
·15,·002
·23,·003
·27,·007
·64,·007
·28,·004
·37,·005
·12,·001
·40,·007
·22,·007
·16,·002
·26,·001
·08,·002
·37,·003
·20,·004
LEGEND
50
SURFACE DIAMOND DRILL HOLE
OVERBURDEN
OXIDE ZONE
CONTACT
·26,·001
% Cu., % Mo.
N.A.
NOT ASSAYED
FINAL PIT OUTLINE
inches
0
1
0
1
2
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.
SCALE
0'
200'
400'
FEET
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE 3 NORTH
JUNE, 1968
JOB No.
VOL. I 108
DRAWING NUMBER
REV.


86
39
35
85
31
43
53
5,032'
4,552'
4,072'
3,592'
REF. LINE
BETHSAIDA
SKEENA
·16,·001
·12,·002
·50,·023
·42,·008
·28,·001
·29,·026
·39,·052
·47,·003
·38,·004
·40,·013
·39,·004
·35,·007
·60,·031
·79,·013
·50,·004
·34,·020
·21,·007
·41,·007
·12,·002
·59,·016
·52,·019
·40,·011
·42,·003
·64,·040
·95,·034
·42,·005
·52,·016
·39,·010
·28,·016
·43,·037
·12,·008
·41,·007
·46,·050
·37,·009
·15,·004
·61,·014
·53,·021
·35,·019
·38,·011
·24,·013
·09,·002
·44,·008
·44,·008
·30,·005
·42,·041
·27,·003
·54,·021
·66,·003
·32,·011
·45,·018
·32,·022
·21,·010
·56,·036
·44,·002
·45,·014
·12,·002
·31,·005
LEGEND
39
SURFACE DIAMOND DRILL HOLE
OVERBURDEN
OXIDE ZONE
FAULT ZONE
·95,·034
% Cu., % Mo.
N.A.
NOT ASSAYED
FINAL PIT OUTLINE
inches
0
1
0
1
2
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.
SCALE
0'
200'
400'
FEET
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE 7 NORTH
JUNE, 1968
JOB No.
VOL. I 109
DRAWING NUMBER
REV.


49
12
11
10
8
24
36
9
13
51
5,032'
4,552'
4,072'
3,592'
REF. LINE
LEGEND
SURFACE DIAMOND DRILL HOLE
OVERBURDEN
OXIDE ZONE
FAULT ZONE
% Cu., % Mo.
NOT ASSAYED
FINAL PIT OUTLINE
N.A.
BETHSAIDA
SKEENA
SCALE 1"=200'
0'
200'
400'
FEET
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE II NORTH
JUNE, 1968
JOB No.
VOL. I 110
DRAWING NUMBER
REV.

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA

VERTICAL CROSS SECTION
LOOKING NORTH
GEOLOGY
LINE 11 NORTH

JUNE, 1968	JOB No.	VOL. I 111 DRAWING NUMBER	REV.


69
71
45
40
76
5,032'
4,552'
4,072'
3,592'
LEGEND
SURFACE DIAMOND DRILL HOLE
OVERBURDEN
OXIDE ZONE
FAULT ZONE
% Cu., % Mo.
NOT ASSAYED
FINAL PIT OUTLINE
BETHSAIDA
SKEENA
N.A.
REF. LINE
.64, .004
.33, .002
.41, .006
.20, .003
.31, .002
.62, .006
.40, .001
.55, .002
.40, .003
.69, .020
.40, .040
.59, .039
.42, .020
.52, .008
.38, .050
.47, .004
.26, .003
.39, .007
.31, .015
.43, .042
.34, .001
.17, .002
.34, .005
.32, .005
.19, .004
.33, .019
.35, .002
.23, .002
.12, .005
.33, .029
.18, .007
.41, .005
.14, .004
.46, .025
.24, .027
.39, .013
.22, .005
.42, .016
.26, .009
.62, .028
.31, .010
.15, .002
.67, .064
.33, .005
.18, .006
.27, .019
.13, .008
.37, .021
.16, .012
.30, .002
.17, .002
.47, .013
SCALE
0'
200'
400'
FEET
inches
centimetres
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE 15 NORTH
JUNE, 1968
JOB No.
VOL. I 112
DRAWING NUMBER
REV.

20
18
17
22
15
87
14
44
5,032'
4,552'
4,072'
3,592'
BETHSAIDA
SKEENA
N.A.
REF. LINE
LEGEND
20
SURFACE DIAMOND DRILL HOLE
OVERBURDEN
OXIDE ZONE
FAULT ZONE
·42, ·005
% Cu., % Mo.
N.A.
NOT ASSAYED
FINAL PIT OUTLINE
SCALE
0'
200'
400'
FEET
inches
centimetres
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE 19 NORTH
JUNE, 1968
JOB No.
VOL. I 113
DRAWING NUMBER
REV.


5,032'
4,552'
4,072'
3,592'
42
38
33
75
77
REF. LINE
BETHSAIDA
SKEENA
N.A.
LEGEND
42
SURFACE DIAMOND DRILL HOLE
OVERBURDEN
OXIDE ZONE
FAULT ZONE
·79, ·008
% Cu., % Mo.
N.A.
NOT ASSAYED
FINAL PIT OUTLINE
SCALE
0'
200'
400'
FEET
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE 23 NORTH
JUNE, 1968
VOL. I
114
JOB No.
DRAWING NUMBER
REV.

52
21
19
16
23
47
5,032'
4,552'
4,072'
3,592'
N.V.
N.A.
.13, .002
.31, .002
.46, .012
.08, .004
.07, .002
.40, .020
.33, .002
.22, .016
.36, .015
.67, .016
.44, .010
.61, .027
.22, .015
.77, .007
.31, .004
.53, .074
.87, .022
.55, .016
.23, .004
.21, .015
.35, .007
.42, .023
.36, .004
.53, .015
.26, .002
.55, .011
.27, .006
.18, .002
.79, .034
BETHSAIDA
SKEENA
REF. LINE
LEGEND
21
SURFACE DIAMOND DRILL HOLE
OVERBURDEN
OXIDE ZONE
FAULT ZONE
.67, .016
% Cu, % Mo.
N.A.
NOT ASSAYED
FINAL PIT OUTLINE
inches
0
1
0
1
2
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.
BRITISH COLUMBIA GEOLOGICAL SURVEY
SCALE
0'
200'
400'
FEET
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE 27 NORTH
JUNE, 1968
JOB No.
VOL. I 115
DRAWING NUMBER
REV.

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.
VANCOUVER — BRITISH COLUMBIA

VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE 31 NORTH

JUNE, 1968		VOL. I 116	
	JOB No.	DRAWING NUMBER	REV.


5,032'
4,552'
4,072'
3,592'
63
65
·15,·007
·38,·026
·07,·002
·49,·014
N.A.
·15,·010
·36,·007
BETHSAIDA
SKEENA
REF. LINE
LEGEND
63
SURFACE DIAMOND DRILL HOLE
OVERBURDEN
OXIDE ZONE
FAULT ZONE
·38,·026
% Cu., % Mo.
N.A.
NOT ASSAYED
FINAL PIT OUTLINE
inches
0 1
0 1 2
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.
SCALE
0' 200' 400'
FEET
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA
VERTICAL CROSS SECTION
LOOKING NORTH
COPPER GRADE TRENDS
LINE 35 NORTH
JUNE, 1968
JOB No.
VOL. I 117
DRAWING NUMBER
REV.

SCALE

0' 200' 400'

FEET

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.

VANCOUVER BRITISH COLUMBIA

VERTICAL CROSS SECTION

LOOKING NORTH

COPPER GRADE TRENDS

LINE 39 NORTH

JUNE, 1968		VOL. I 118	
	JOB No.	DRAWING NUMBER	REV.

REF. LINE
303
304
305
301
302
287
282
283
281
280
278
279
275
B 10
263
257
258 A
262
259
260
261
5,032'
4,552'
LEGEND
261
PERCUSSION DRILL HOLE
B 10
BECKER DRILL HOLE
OVERBURDEN
OXIDE ZONE
FAULT
SOLUABLE Cu.
TOTAL Cu.
SCALE
0
100'
200'
FEET
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
PERCUSSION DRILL HOLES
LOOKING NORTH
SECTION 11 NORTH
JUNE, 1968
JOB No.
VOL. I 119
DRAWING NUMBER
REV.
inches
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.
BRITISH COLUMBIA GEOLOGICAL SURVEY

L-39 N
L-35 N
L-31 N
L-27 N
L-23 N
84,000 N
L-19 N
L-15 N
L-11 N
L-7 N
L-3 N
L-1 S
L-5 S
L-11 S
L-15 S
L-21 S
A.N.
102,000 E
102,000 E
REF. LINE
REF. LINE
5112
5032
4952
4872
4792
6·0
8·0
Pit
LEGEND:
INCLINED D.D. HOLE
VERTICAL D.D. HOLE
TEST PIT
PIT OUTLINE
5112 ELEVATION IN FEET
6·0
8·0
I.P. CONTOUR INTERVAL IN MILLISECONDS, USING 800' SPACING & THREE ELECTRODE ARRAY.
SHAFT
UNDERGROUND DEVELOPMENT
SCALE
0'
300'
600'
FEET
inches
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
SURFACE PLAN
VOL. I
120
JOB No.
DRAWING NUMBER
REV.

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA

COPPER GRADE TRENDS
BASED ON
UNDERGROUND DRILL HOLES

JUNE, 1968	JOB No.	VOL. I 121 DRAWING NUMBER	REV.

102,000E

LEGEND

D=2	DENSITY OF MINERALIZED FRACTURES PER FOOT
XCW 60	ROUND NUMBER
·28	% COPPER
·003	% MOLYBDENUN
	FAULT
	MINERALIZED FRACTURE
	MINERALIZED QUARTZ VEIN

SCALE

0 10' 20'

FEET

inches 0 1

centimetres 0 1 2

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.

VANCOUVER BRITISH COLUMBIA

WEST CROSS-CUT

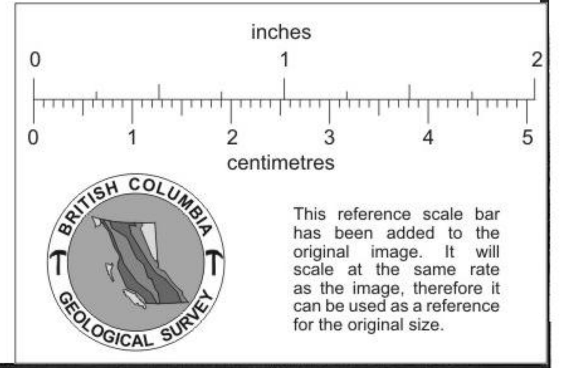
REPRESENTATIVE SECTION SHOWING GEOLOGY OF BACK AND NORTH WALL

JUNE, 1968		VOL. 1 122	
	JOB No.	DRAWING NUMBER	REV.


LEGEND
OVERBURDEN
OXIDE ZONE
B BETHSAIDA GRANODIORITE
Q QUARTZ PORPHYRY
G GRANITE
Sa SKEENA QUARTZ DIORITE Alteration Zone
S SKEENA QUARTZ DIORITE
CONTACT
PROMINENT FAULT - ASSUMED
PIT OUTLINE
L-35 N
L-31 N
L-27 N
L-23 N
84,000 N
L-19 N
L-15 N
L-11 N
L-7 N
L-3 N
L-1 S
L-5 S
102,000 E
REF. LINE
A.N.
AWARD CREEK FAULT
SCALE
0'
300'
600'
FEET
inches
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.
BRITISH COLUMBIA GEOLOGICAL SURVEY
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
GEOLOGICAL PLAN at 4552 FT. ELEVATION
JUNE, 1968
JOB No.
VOL. I 123
DRAWING NUMBER
REV.

LEGEND
OVERBURDEN
OXIDE ZONE
B BETHSAIDA GRANODIORITE
Q QUARTZ PORPHYRY
Sa SKEENA QUARTZ DIORITE Alteration Zone
CONTACT
TREND LINE
PIT OUTLINE
·31, ·010 % Cu., % Mo.
DRILL HOLE - CORED
DRILL HOLE - NOT CORED
UNDERGROUND DEVELOPMENT
L-35 N
L-31 N
L-27 N
L-23 N
84,000 N
L-19 N
L-15 N
L-11 N
L-7 N
L-3 N
L-1 S
L-5 S
102,000 E
REF. LINE
A.N.
Sa
Q
B
Shaft
AWARD CREEK FAULT
SCALE
0' 300' 600'
FEET
inches
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA
COPPER GRADE TRENDS
at
4552 FT. ELEVATION
JUNE , 1968
JOB No.
VOL. I 124
DRAWING NUMBER
REV.


RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
UNDERGROUND TEST AREA
COPPER GRADE TRENDS
from
SURFACE DRILL HOLES
JUNE, 1968
JOB No.
VOL. I 125
DRAWING NUMBER
REV.
SCALE
FEET
L-15 N
L-11 N
L-7 N
REF. LINE
UG-7
UG-8
UG-4
UG-3
UG-6
UG-2
UG-1
UG-5
UG-9
UG-10
NORTH DRIVE
SOUTH DRIVE
WEST CROSS-CUT
EAST CROSS-CUT
SHAFT
% Cu.
% Mo.
83,000 N
102,000 E



RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA

COPPER ASSAYS
E-W X-CUT and UG-I

JUNE, 1968	JOB No.	VOL. I 126 DRAWING NUMBER	REV.

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA

MOLYBDENUM ASSAYS
E-W X-CUT and UG-I

JUNE, 1968		VOL. I 127	
	JOB No.	DRAWING NUMBER	REV.

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA

COPPER ASSAYS
W X-CUT and UG-2, 6

JUNE, 1968		VOL. I 128	
	JOB No.	DRAWING NUMBER	REV.

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA

MOLYBDENUM ASSAYS
WX-CUT and UG-2, 6

JUNE, 1968		VOL. 1 129	
	JOB No.	DRAWING NUMBER	REV.

REF. LINE
L-15 N
L-11 N
L-7 N
15-1
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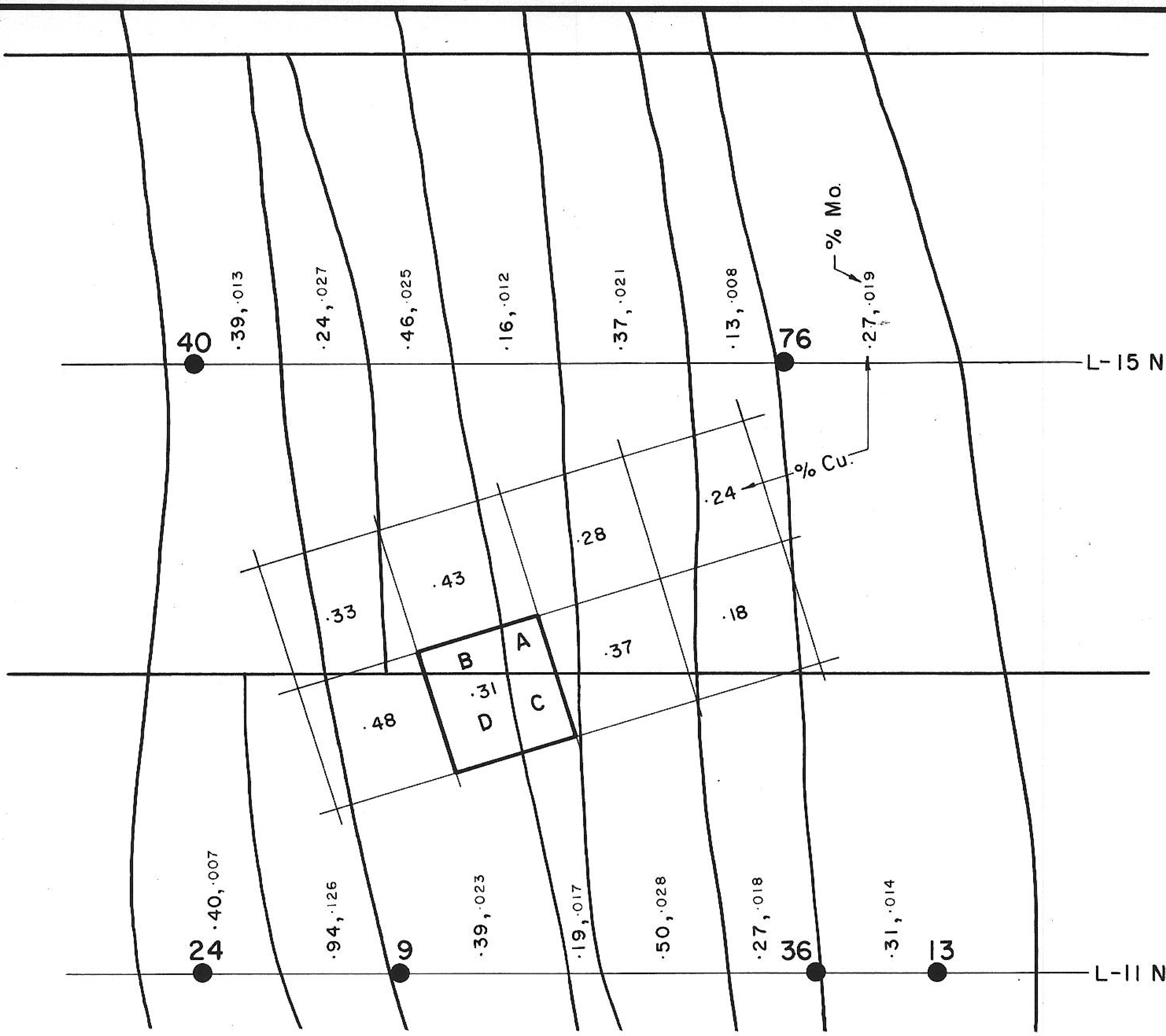
7-18
7-19
7-20
7-21
NORTH DRIVE
SOUTH DRIVE
WEST CROSS-CUT
EAST CROSS-CUT
SHAFT
% Cu.
% Mo.
N.M.
102,000 E
83,000 N
LEGEND:
15-6 - BLOCK NUMBER
A,B - ORE
C - SUB-ORE
D - WASTE
N.M. - NOT MEASURED
- FINAL PIT
SCALE
0
100'
200'
FEET
inches
centimetres
RIO TINTO CANADIAN EXPLORATION LIMITED
LORNEX MINING CORPORATION LTD.
VANCOUVER
BRITISH COLUMBIA
ORE RESERVE BLOCKS
at
4552 FT. ELEVATION
JUNE, 1968
JOB No.
VOL. I 130
DRAWING NUMBER
REV.



CALCULATION OF GRADE OF SQUARE :

A = 17 / 100 x 0·16 % Cu.
B = 18 / 100 x 0·46 % Cu.
C = 22 / 100 x 0·19 % Cu.
D = 43 / 100 x 0·38 % Cu.
100

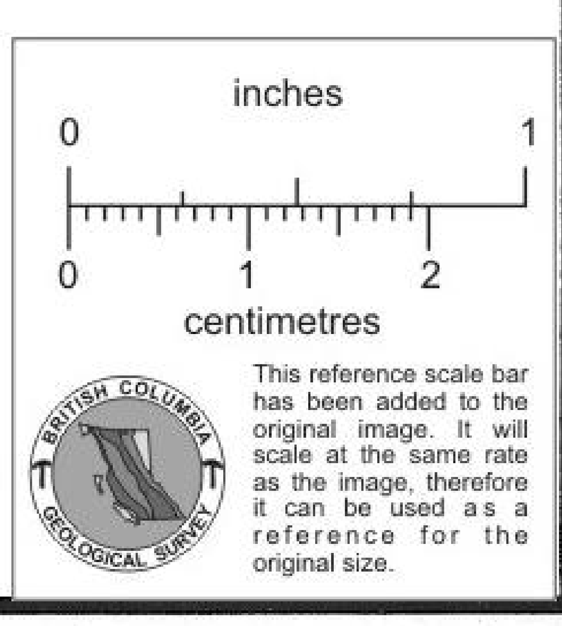
AVERAGE 0·31 % Cu.

RIO TINTO CANADIAN EXPLORATION LIMITED

LORNEX MINING CORPORATION LTD.
VANCOUVER BRITISH COLUMBIA

EXAMPLE OF 80 FT. SQUARE
SHOWING
CALCULATION & SMOOTHING EFFECT

JUNE, 1968	JOB No.	VOL. I 131 DRAWING NUMBER	REV.



APPENDIX "A"

MINERAL CLAIMS IN THE KAMLOOPS MINING DIVISION HELD BY LORNEX MINING CORPORATION LTD. (N.P.L.)

CLAIM	RECORD NO.	LATEST DATE FOR RECORDING FURTHER WORK
Award No. 1 Fr.	42130	April 16, 1972
2 Fr.	42131	April 16, 1972
3 Fr.	42659	May 6, 1975
5 Fr.	53243	December 28, 1975
7 Fr.	57274	June 20, 1975
36	38503	November 15, 1975
37	38504	November 15, 1975
38	38505	November 15, 1975
39	38506	November 15, 1975
40	38507	November 15, 1975
41	38508	November 15, 1975
42	38509	November 15, 1975
43	38510	November 15, 1975
44 Fr.	38511	November 15, 1981
45	38512	November 15, 1975
46	38513	November 15, 1975
47	38514	November 15, 1975
48	38515	November 15, 1975
49	38516	November 15, 1975
50	38517	November 15, 1975
51	38436	October 13, 1975
52	38437	October 13, 1975
53	38438	October 13, 1975
54	38439	October 13, 1975
55	38440	October 13, 1975
56	38441	October 13, 1975
57	38442	October 13, 1975
58	38443	October 13, 1975
59	38444	October 13, 1975
60	38445	October 13, 1975
61	38416	October 13, 1975
62	38417	October 13, 1975
63	38418	October 13, 1975
64	38419	October 13, 1975
65	38420	October 13, 1975
66	38421	October 13, 1975
67	38422	October 13, 1975
68	38423	October 13, 1975
69	38424	October 13, 1975

CLAIM	RECORD NO.	LATEST DATE FOR RECORDING FURTHER WORK
Award No. 70	38425	October 13, 1975
73	38518	November 15, 1975
74	38519	November 15, 1975
75	38520	November 15, 1975
76	38521	November 15, 1975
AM No. 33	31466	March 31, 1975
34	31467	March 31, 1975
35	31468	March 31, 1975
36	31469	March 31, 1975
37	31470	March 31, 1975
38	31471	March 31, 1975
39	31472	March 31, 1975
40	31473	March 31, 1975
41	31464	March 31, 1975
42	31465	March 31, 1975
Granite No. 1 Fr.	57272	June 20, 1975
2 Fr.	57273	June 20, 1975
9 - 16	41124-41131	November 23, 1975
Lornex Nos. 1 - 2	46648-46649	June 18, 1975
3 - 4	46650-46651	June 18, 1975
Iris Nos. 1 - 2	40069-40070	June 29, 1972
3 - 14	40071-40082	June 29, 1975
31 - 32	40099-40100	June 29, 1975
33 - 44	46834-46845	July 15, 1975
45 - 48	46846-46849	July 15, 1972
1 - 2 Frs.	46086-46087	April 30, 1972
3 Fr.	52558	November 12, 1975
4 Fr.	52560	November 12, 1975
AW Nos. 15 - 26	46661-46672	June 25, 1975
Zone Nos. 1 - 6	48297-48302	December 10, 1975
7 Fr.	48303	December 10, 1975
8 - 11	48304-48307	December 10, 1975
12 Fr.	48308	December 10, 1975
Apex No. 25	51940	September 17, 1975
73 - 74	51405-51406	August 31, 1975
76 Fr.	51407	August 31, 1975
80 Fr.	51408	August 31, 1975
88	51409	August 31, 1974
89 Fr.	51410	August 31, 1975
90 Fr.	51411	August 31, 1974
Scott Fr.	52559	November 12, 1974
Lea Nos. 1 - 2 Fr.	53457-8	February 3, 1975

CLAIM	RECORD NO.	LATEST DATE FOR RECORDING FURTHER WORK
Doug Nos. 1 - 2 Frs.	52760-1	November 26, 1975
3 - 4 Frs.	53244-5	December 28, 1975
5 - 7 Frs.	53246-8	December 28, 1975
NW FR.	47940	November 16, 1975
Ben No. 1	47503	October 7, 1975
1 Fr.	52619	November 15, 1975
2 - 4	47504-6	October 7, 1975
5 - 6	52620-1	November 15, 1975
Lornex No. 1 Fr.	46645	June 18, 1983
Lornex Nos. 2 - 3 Frs.	46646-7	June 18, 1975
Camp Nos. 3 - 4	62483-4	December 30, 1975
I.L. Nos. 15 - 16	62481-2	December 30, 1975
Skeena Copper Nos. 15 - 16	24471-2	August 29, 1975
19	24475	August 29, 1975
21	24477	August 29, 1976
Moose Nos. 1 - 5 Fr.	61785-89	November 18, 1975

MINERAL CLAIMS HELD BY SKEENA SILVER MINES LTD. (N.P.L.)
(Now Consolidated Skeena Mines Ltd. (N.P.L.) Subject to Option Agreement)

Skeena Copper Nos. 1 - 4	16494-7	October 5, 1975
7 - 11	16500-04	October 5, 1975
22	38814	March 8, 1975
Divide Copper	14905	June 17, 1975
Divide Copper Nos. 1 - 3	14906-8	June 17, 1975
5	15279	July 12, 1975

MINERAL CLAIMS HELD BY NORANDA EXPLORATION COMPANY LIMITED (N.P.L.)
Subject to Agreement dated July 1st, 1967

Calco Nos. 17		November 26, 1976
19		November 26, 1976
20		November 26, 1976
21 Fr.		November 26, 1976
22		November 26, 1976
23		November 26, 1976
24 Fr.		November 26, 1976

MINERAL CLAIMS HELD BY NORANDA EXPLORATION COMPANY LIMITED (N.P.L.)

Subject to Agreement dated July 1st, 1967

CLAIM	RECORD NO.	LATEST DATE FOR RECORDING FURTHER WORK
Calco Nos. 25 Fr.		November 26, 1976
26		November 26, 1976
28 Fr.		November 26, 1976

MINERAL CLAIMS HELD BY LORNEX MINING CORPORATION LTD. OUTSIDE THE PRINCIPAL GROUP

Pix Nos. 1 - 3	60349-51	September 14, 1969
4 Fr.	60352	September 14, 1969
5	60353	September 14, 1969
6 Fr.	60354	September 14, 1969
8	60355	September 14, 1969
10	60356	September 14, 1969
12	60357	September 14, 1969
14	60358	September 14, 1969

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COCKFIELD, W. E.	Geology and Mineral Deposits of the Nicola Map-Area, British Columbia; Geological Survey of Canada, Memoir 249, 1948.
DUFFELL, S., and McTAGGART, K. C.	Ashcroft Map-Area, British Columbia, Geological Survey of Canada, Memoir 262, 1952.
WHITE, W. H., THOMPSON, R. M., and McTAGGART, K. C.	The Geology and Mineral Deposits of Highland Valley, British Columbia; C.I.M.M. Bull., Vol. 50, No. 544, August 1957.
CARR, J. M.	Porphyries, Breccias, and Copper Mineralization in Highland Valley, British Columbia, Canadian Mining Journal, Vol. 81, No. 11, November 1960.
NEWELL, J. M.	Lornex Mining Corporation, Highland Valley Property, Kamloops Mining Division, British Columbia. Private report in the files of Rio Tinto Canadian Exploration Limited, Toronto 1965.
GOLDEN, BRAWNER AND ASSOCIATES LTD.	Preliminary Evaluation of Pit Slopes and Tailings Dam Stability, Highland Valley, British Columbia. Private report in the files of Lornex Mining Corporation Ltd., 1967.

RIO TINTO CANADIAN EXPLORATION LIMITED

Toronto, Ontario

REVIEW OF METHODS AND RESULTS

OF

CORE AND BULK SAMPLING - SAMPLE ANALYSES - RESERVE ESTIMATION

LORNEX COPPER - MOLYBDENUM PROJECT

ASHCROFT AREA - BRITISH COLUMBIA

July 1968

BEHRE DOLBEAR & COMPANY, INC.

MINING, GEOLOGICAL AND METALLURGICAL CONSULTANTS

NEW YORK

CABLE ADDRESS
BEHRDOL

TELEPHONE
DI 4-5530
AREA CODE 212

BEHRE DOLBEAR & COMPANY
INCORPORATED

MINING, GEOLOGICAL AND METALLURGICAL CONSULTANTS

ELEVEN BROADWAY
NEW YORK, N. Y. 10004

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CONSULTANTS
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T. B. COUNSELMAN

ASSOCIATES
FRANK H. MADISON
R. W. McCALLUM
WILLIAM B. AGOCS
A. A. GUSTAFSON

July 29th, 1968.

Mr. J. A. Sadler, President,
Rio Tinto Canadian Exploration Limited,
120 Adelaide Street, West.
TORONTO 1, Ontario, Canada.

Dear Mr. Sadler:

RE: Lornex Project

In accordance with your request submitted herewith is a summary of observations on the results of the development program carried out by your Company on the Lornex copper-molybdenum deposit some 30 miles south-east of Ashcroft, British Columbia.

The observations are largely confined to matters bearing on the determination of the tonnage and average grade of mineralized material within a proposed open pit (No. 15) designed by personnel of your Company as a basis for a feasibility and economic study.

Opinions are based on study of the results of 66 core holes totalling 74,044 feet drilled from surface, 550 feet of shaft sinking, 1,674 feet of under-ground cross-cutting and 873 feet of other lateral work, and 10 horizontal core holes totalling 5,300 feet drilled from underground workings. Some 211 short vertical percussion and rotary holes totalling 50,056 feet have also been drilled from surface. Since shaft sinking was started in March 1967, the property has been visited on the average of once every six weeks. During each visit the underground workings were inspected; the operations of the bulk sampling plant cutting representative samples from mineralized material from the underground workings were studied; the preparation of pulp samples for chemical analyses from the samples from the bulk sampling plant were observed; a review was made of the methods of chemical analyses used by the laboratory on the property and by those doing check analyses, that is, the laboratory at Elliot Lake, Ontario, operated by your affiliate Company and that by Coast Eldridge, an independent laboratory in Vancouver. Periodically samples were taken by Behre Dolbear during operations in the bulk sampling plant and in the pulp sample preparation room which were analyzed by an independent laboratory in New York. Finally, an estimate of the tonnage and grade of the mineralized material in the proposed pit, based on vertical cross sections was made manually as a check of the estimates made by Riocanex. All the basic data developed since the start of shaft sinking and all the basic data used by Riocanex in calculating its estimates were checked by Behre Dolbear. Cores from surface drilling completed before retention of Behre Dolbear were studied but not resampled.

The program of underground development and bulk sampling was well planned and competently carried out. Everything was done to insure that the information derived from it was as accurate as possible. The results, therefore, may be accepted with confidence. Comments on pertinent aspects of the program are contained in the following pages.

Field work and supervision of this project were done by Armine F. Banfield, Geologist.

The full co-operation of your staff is gratefully acknowledged,

Respectfully Submitted,

BEHRE DOLBEAR & COMPANY, INC.

Armine F. Banfield

Armine F. Banfield,
Vice-President.

C O N T E N T S

CONCLUSIONS

1. Riocanex' Estimates of Tonnage and Grade

Riocanex has made the following estimates of the sulphide mineralized material within the proposed Pit No. 15 designed by the Company, based on analyses of cores from holes drilled from surface:

SULPHIDE MATERIAL

	Category	Short Tons	% Total Cu.	% Total Mo.
For Treatment	0.26% total Cu. & above	274,000,000	0.447	0.0147
Possible Stockpile	Between 0.20% & 0.259% Total Copper	38,700,000	0.226	0.0066
Waste	0.19% total Cu. & below	46,200,000	0.135	0.0040

These estimates were calculated manually from zones of similar grade which had been correlated from hole to hole on vertical cross sections spaced 400 feet apart and horizontal plans spaced 80 feet vertically. The deposit subsequently was also subdivided into cubes 80 feet to a side, the grade of each cube being derived from the horizontal plans. A computer study based on these cubes gave the following estimate:

	Short Tons	% Total Cu.	% Total Mo.
0.26% Total copper and above	292,800,000	0.427	0.0140

Because some of the cubes overlap high grade and very low grade or waste zones, the effect of consolidating into cubes is to dilute the higher grade material. Thus the tonnage of the estimate of 274.0 million tons given above is increased to 292.8 million tons while the average grades are decreased by 4.5 per cent. Since dilution will occur during mining, the estimate of 292.8 million tons with a grade of 0.427 per cent total copper and 0.0140 per cent total molybdenum is probably a more realistic estimate to use in feasibility calculations.

These are acceptable estimates of the over-all tonnage in Pit 15 and may be used with confidence in financial studies. The figures of average grade are considered to be conservative.

2. Grades of Surface Core Holes Compared to Cross Cut

Grades shown by the bulk sampling of 13,000 tons from the 1673 foot cross cut compared to those estimated from cores of eight surface holes cutting the same area are as follows:

	% Total Cu.	% Total Mo.
Cross Cut	0.465	0.0156
Surface Core Holes	0.423	0.0137 (1)
% Difference based on Holes	9.9%	13.8%

(1) After eliminating from 258 core samples in this area, three abnormally high samples representing 30 feet and averaging 0.1842 per cent total molybdenum from Hole 9, from 630 to 660 feet. If these three samples are included the average becomes 0.0168 per cent total molybdenum.

These lower grades in the core holes probably reflect the loss of copper and molybdenum minerals during drilling since core recovery on the average was only 94.3 per cent by length (probably less by weight).

The conclusion is drawn from these figures that the grades shown by the cores from surface holes have not over-valued the deposit but on the contrary are conservative. It would not be sound practice though to up-grade the average grade of the deposit on the basis that the cross cut grades are a certain per cent higher than those of the surface core holes cutting the area. However, assuming that the cross cut has an area of influence extending to 400 feet to the north and to the south (that is to the limits of the horizontal core holes on the cross cut level) and up to 200 feet vertically above and below, then it would have an influence on the grades of about one-third of the tonnage to be mined in the first five years.

3. Accuracy of the Analyses

Both the copper and molybdenum are present in lower amounts than what are referred to as "low grade" deposits of these metals. The accuracy of the methods of bulk sampling, sample preparation and chemical analysis are obviously important. A study of the data from all sources indicates that from the standpoint of bulk sampling, sample preparation and chemical analyses, the average grades should be accurate within 0.01 per cent total copper and 0.002 per cent total molybdenum, or within 2 per cent and 12 per cent of the contained amount of these metals.

4. Dilution

The estimated grades of the sulphide material are subject to dilution during mining. One source of dilution may be the low grade bands of material within the deposit which have been segregated in the estimates from the material that is expected to go to the concentrator. Some of this low grade material of course has been included in compositing the deposit into 80 foot cubes. Whether this is an adequate amount of dilution should be determined by the mining department. There also may be dilution from the walls of the pit, and from near surface areas where oxidation extends to greater depth than estimated. The principal dilution, however, will be from internal waste and its control will depend upon the care taken by the shovel operators and their supervisors, and the grade control personnel.

The skill of the operating staff in controlling dilution may have a greater effect on the grade of material processed than variations due to loss of core in surface holes on which the estimates have been based.

5. Variation in Grade and Recovery of Molybdenum

The eastern part of the deposit has a much higher molybdenum content than the central and western parts. For example, cross cut west rounds 6 to 45 (240 feet) contained an average of 0.038 per cent total molybdenum, while the rounds 46 to 94 (265 feet) averaged 0.010 per cent molybdenum. No flotation tests are reported to have been run on material containing 0.010 per cent total molybdenum or less. Hence the percentage of recovery from material of this grade is not known. This should be considered in feasibility studies before assigning dollar values to the amount of molybdenum that might be recovered from the lower grade parts of the deposit.

Near the surface some, and in places, most of the molybdenum sulphide has been altered by weathering to the "oxide" or "soluble" form, the percentage decreasing downward.

A composite prepared from every tenth round of the cross cut showed a minor amount of molybdenum in oxide form. However, the amounts of molybdenum, in this form, as shown by the results of the pilot plant tests should be used in studies as they would be more representative. The effect of the presence of oxide of molybdenum on production during the first two or three years should be considered since this material is not readily recoverable without special treatment.

6. Oxide Material

The tonnage and grade of the near-surface material in the oxidized or weathered zone overlying the sulphide zone have not been adequately established. The drill holes are too far apart to delineate or accurately sample material that varies so erratically in thickness and grade. This is not a serious matter at this time since an economic method of processing the material has not yet been worked out. The higher grade material should be stockpiled for possible future treatment.

7. Effect of Fracturing on Mining

Geological maps of the cross cut show over 180 faults ranging in width from one to six inches in a distance of 1400 feet, or an average of one every eight feet. Most of these contain gouge, or clay, and may contain molybdenite, a natural lubricant. Both the clay and molybdenite make the rock incompetent and increase the possibility of movement, a tendency that is substantially increased by water. In addition to the faults there are abundant fractures, and as a result of these features, the rock tends to break readily, a fact attested to by the extensive sloughing in the underground workings. The incompetent character of the rock will affect not only slope stability but the construction, location and maintenance of roads.

8. Achievement of Objectives of Program

It is considered that the objectives originally planned for the underground program have been largely achieved as to bulk sampling and geology. Additional core drilling and, or, underground work is not likely to change average grades or confirm to a much greater degree the hole to hole correlation of the mineralization. Some information is lacking in areas beyond the ends of the cross cut particularly on the west side of the deposit but this affects only a minor area.

Some fifteen thousand tons of material from the underground workings have been systematically stockpiled near the pilot plant for future use if required.

GENERAL COMMENTS

1. Sampling and Assaying

Every precaution appears to have been taken in surface core drilling to obtain the highest possible percentage of core recovery. Despite this, core recovery, based on length, averaged only 94.3 per cent. Recovery in the 10 underground holes averaged 89.7 per cent by length but only 83.1 per cent by weight. Mud was used as a drilling fluid because this method resulted in the best core recovery. However, its use meant that sludge samples could not be taken and hence grade could only be determined from core alone. The core loss was due to the presence of considerable clay-like material (gouge, kaolin and fine mica) resulting from faulting or rock alteration, and to closely spaced fractures, particularly in quartz veins.

The cores, 1-7/8 inches in diameter were split, or divided where soft, in half, one half being sent for analysis and the other being retained in core boxes for reference or metallurgical tests. Sample lengths were 10 feet. To establish whether this was an acceptable method the remaining half of 2086 feet of core was also analysed. There was a difference of only 0.02 per cent total copper and 0.002 total molybdenum between the averages of the two halves of the cores, or 5 per cent of the amount of contained copper and 12 per cent of the contained molybdenum.

The conclusion, therefore, is that one half of the cores did provide a representative sample.

Pulp samples for analysis were prepared from the surface cores first at Elliot Lake and later at the mine laboratory. In each place the core sample was reduced to a one or two pound sample which was pulverized to minus 80 or 100 mesh. Two or three samples were taken from this pulp for analyses. From holes 8 to 42 each pulp was prepared and analyzed at Elliot Lake for total copper and MoS_2, the soluble molybdenum having been removed. A duplicate pulp of every fifth sample was sent to Coast Eldridge, an independent laboratory in Vancouver for checking. From holes 42 to 87 the pulps were prepared and analyzed at the mine with every fifth sample being checked by Elliot Lake and Coast Eldridge except that Elliot Lake continued to determine MoS_2 to hole 65. For the 10 horizontal underground core holes, the samples were prepared and analyzed by the mine laboratory with check analysis of every pulp being done by Coast Eldridge with referee samples being done by Elliot Lake. Sample preparation of cores from surface holes appears to have been satisfactory. The average grades of core samples reported by the three laboratories were -

COPPER

HOLE NO.	NUMBER OF SAMPLES	% Total Copper LORNEX	ELLIOT LAKE	COAST ELDRIDGE
8 to 40	566	-	0.379	0.390
41 to 72	174	0.295	0.284	0.306
41 to 87*	216	0.314	-	0.323

*No Coast Eldridge assays from Holes 70, 73 to 77 inclusive. Hence these holes have been omitted.

MOLYBDENUM

			% Mo.	% Mo.
8 to 72	861	-	0.0114 (sulphide only)	0.0138 (Total)

The conclusion is drawn from this table that the average grades calculated from a large number of these core analyses are accurate.

The three laboratories used standard but somewhat different methods of analyses and equipment. The results indicate the average of the analyses reported by the three laboratories for 778 samples showed a maximum difference of 0.02 per cent total copper. About 65 per cent of the total number of analyses differed by 0.02 per cent copper or less while 90 per cent of them differed by less than 0.05 per cent copper. In the case of molybdenum the averages of the analyses by Elliot Lake and Coast Eldridge showed an average difference of 0.0024 per cent molybdenum. About 67 per cent of the analyses differed by 0.003 per cent molybdenum or less while

90 per cent differed by 0.006 per cent molybdenum or less. Part of these differences may be because Elliot Lake up to hole 72 leached out the soluble molybdenum before making the determination. Thus its results show only the molybdenum in sulphide form.

2. Core Drilling

The estimates of mineralized material given on Page 1 were based on 66 core holes totalling 74,044 feet drilled from surface. These holes were inclined mostly at 60 degrees to the west and were drilled on sections oriented across the long axis of the deposit, the sections being spaced 400 feet apart. Most sections contained from four to eight holes, the collars of which were spaced 350 to 400 feet apart. The collars of the holes were thus roughly at the corners of 400 foot squares. Taken in conjunction with the underground bulk sampling and core drilling program it is considered that an adequate number of holes and footage of drilling have been done for establishing the tonnage and grade to the extent required for a feasibility study. Because of the amount of core that is lost in drilling, additional holes would not establish the grade more firmly. The results of the underground core drilling indicate that additional holes along intermediate sections are not likely to change the present hole to hole correlation to any important extent.

3. Overburden and Oxide Zone

The sulphide-mineralized zone is overlain irregularly by variable thicknesses of weathered or "oxide" material. In places, particularly in the western part of the deposit, all the weathered material was removed by the glacier. Overlying all bedrock now is a mantle of glacial debris composed of sand, gravel and clay varying from 20 to 200 feet thick.

Because of the broken character of the rock in the weathered zone, it is very difficult, even with coring, to determine the location of the top and bottom of the oxide zone, or the top of the sulphide zone. For the same reason it was not possible to obtain representative samples for analysis because there was usually some salting by material caving into the hole from above.

The Company used several types of percussion drilling, rotary drilling and core drilling. Of these the core drilling provided the most reliable results. A seismic survey was also run over the deposit in an effort to delineate more accurately the location of the top of the solid rock. While results confirmed in a general way those of the drilling, they added little more accurate data.

Tests run in the pilot flotation plant on oxidized material from the small open pit showed that oxidized material interfered with the flotation of the copper and molybdenum sulphide minerals. For this reason the top of the sulphide zone was taken where the soluble copper content was less than 20 per cent of the total copper in the sample.

Insufficient drilling has been done to establish the tonnage and grade of the oxide material with the accuracy required for a feasibility study. A hole spacing of 100 feet between holes would probably be required as compared to the present 400 foot spacing. However, an economic process for treating the oxide material has not yet been worked out. For the present, therefore, the oxide material should be separately stockpiled for possible treatment in the future. The following estimates are possible:

RIOCANEX' ESTIMATES OF OXIDE MATERIAL

Category	Short Tons	% Total Cu.	% Soluble Cu.
(a) Greater than 50% acid soluble using cut-off of 0.25% Cu. and above.	13,086,000	0.50	0.38
(b) 20 to 50% acid soluble with no cut-off.	4,814,000	0.48	0.135
(c) Greater than 50% acid soluble but less than 0.25% total Cu.	3,583,000	No grade estimated	

4. Bulk Sampling Plant

All of the rock from the shaft, cross cut and some of that from the North and South Drives was put through the bulk sampling plant on surface. In this plant all the rock was crushed to minus 3/8 size and a tenth removed. This tenth was crushed to minus 1/4 inch and a tenth removed. This tenth was then crushed to minus 10 mesh and a tenth cut out for a laboratory sample. This process resulted in a final sample theoretically weighing 0.1 per cent of the head sample. That is, a sample weighing 80 pounds was cut from a 40 ton underground round. In actual practice the ratios varied slightly from time to time and the final sample generally represented from 0.12 to 0.15 per cent by weight of the head sample. Occasionally considerably more than 40 tons was run through as one bench or round due to sloughing off the walls and as a result the final samples weighed as much as 300 pounds. Screen analyses made weekly of the final samples showed that generally over 90 per cent was less than 10 mesh in size. When it was found less than this adjustments were made to the spacing of the rolls. Periodically test runs of a ton or so were run through the plant to make sure that the final cut was in the proper ratio. Four underground rounds were run through the sampling plant twice. Analyses of three samples produced agreed within 0.02 per cent total copper, which was considered a satisfactory check. The two samples from the fourth run - Shaft Bench No. 14, differed by 0.05 per cent copper, the two grades being 0.89 and 0.84 per cent. The bulk sampling plant functioned satisfactorily. Its operations were periodically checked by Behre Dolbear.

It is considered that the samples produced by the bulk sampling plant were very accurate and representative of the rock cut by the cross cut.

The 80 pound or so sample from the bulk sampling plant, after drying, was reduced in volume by a Jones Riffler to about 10 pounds. This sample was then crushed to about 20 mesh in a gyratory crusher. The 20 mesh material was then riffled down to a one to two pound sample which was then pulverized until it all passed through a 100 mesh screen. It was from this pulverized sample that the samples for analyses were taken.

The standard practice was to analyze one sample of the pulp at the mine and to send one to Coast Eldridge in Vancouver for check analyses. If the analyses did not agree within 5 per cent of the total copper content if above 0.20 per cent total copper or 10 per cent if below, and within 10 per cent of the total molybdenum content if above 0.005 per cent total molybdenum or 100 per cent if below 0.005 per cent, then the mine re-analyzed the sample. If its second analysis did not agree with the Coast Eldridge analysis within these tolerances, a sample of the pulp was then sent to Elliot Lake for referee analysis. As a further check Behre Dolbear periodically cut samples from the pulps independently as they were being prepared and had these analyzed by Lucius Pitkin, Inc. in New York. The average grades of 1408 feet of cross cut determined by the mine and Coast Eldridge laboratories differed by only 0.0035 per cent total copper and 0.0003 per cent total molybdenum. The method of sample preparation and the analytical results may therefore be accepted with the confidence that they are representative.

The parts of the North and South Drives that were not put through the bulk sampling plant were car-sampled. A test program run in the main cross cut comparing the results of car-sampling with those of the bulk sampling plant showed a difference of only 0.02 per cent total copper. Car-sampling is therefore acceptable providing there is a sufficiently large number of samples.

The excessive sloughing from the walls made conditions too dangerous for the taking of chip or channel samples. It was also not possible to obtain representative samples of sludge from the face holes because of the excessive amount of clay or gouge in the rock.

This underground program as completed, despite difficulties, is considered to have achieved the original objectives as to bulk sampling and geology.

5. Underground Program

The original underground program envisioned a 550 foot shaft, from which 3000 feet of cross cutting was to be done on the 500 foot level. Two hundred feet of raising was to be done in a high grade zone and 100 feet in a lower grade zone up core holes which had already been drilled from surface. The objectives, from a geological standpoint, were to check the accuracy of the surface diamond drill core sampling results, and to determine particularly whether any bias existed. This underground program had to be modified because the very broken and friable character of the rock necessitated the almost complete support of the workings in order to insure safe working conditions. This of course increased costs substantially and doubled the length of time required to mine a round. The idea of raising had to be abandoned because it was considered too dangerous. Instead, drifts were driven 400 feet to the north to section 15N and to the south to section 7N from the central part of the cross cut. From these drifts 7 horizontal core holes were drilled to the east and west at 200 foot intervals to check the assumed continuity of the geology in a north-south direction. Core holes were also drilled ahead of the shaft (No. 36) and cross cut (Nos. U-1, U-2, U-6) and the cores sampled at intervals corresponding to the rounds in the working.

The program as finally completed consisted of 1674 feet of cross cutting, 873 feet of drifting and 5300 feet of horizontal core holes in an area approximately 1700 feet east-west and 800 feet north-south, in addition to the 550 foot shaft core hole.

All the workings were meticulously mapped geologically and the results plotted on plans and sections on a scale of 1 inch to 10 feet. An attempt was made to channel-sample the walls but this was abandoned because it was too dangerous. Similarly, the taking of sludge samples from the core holes and rock bit holes was found not practical.

The host rock in which the deposit occurs originally appears to have been a relatively homogenous formation. The copper-molybdenum mineralization was introduced into the solid rock along fractures and faults. Geological mapping of the cross cut indicates that there are several sets of faults and fractures with different dips and strikes. The major trend however is in a north-south direction with a steep general dip to the east. Using these directions as a basis, the Company's geologists joined zones of similar grade, as shown by the surface core holes, first on vertical cross sections 400 feet apart and then on horizontal plans 80 feet apart vertically. It was found that definite zones of similar copper grade, from a few tens of feet up to two hundred feet wide, could be traced for the entire length of the deposit from section to section.

Results of the underground horizontal core holes have shown that these correlations, called "Trends" are sufficiently consistent to form the basis for calculating the average grade of the deposit.

The average grade shown by the bulk sampling of some 13,000 tons from the cross cut was 0.465 per cent total copper and 0.017 per cent total molybdenum. The grade of this area as calculated from the eight surface core holes cutting across it is 0.423 per cent total copper and 0.0137 per cent total molybdenum[(1)]. The grades shown by the bulk sampling of the cross cut are considered to be representative. Hence copper and molybdenum grades calculated from the surface core holes are too low, probably due to the loss of valuable constituents in the core lost in drilling.

The three core holes drilled ahead of the cross cut and sampled in the same intervals as the cross cut rounds showed an average grade of 0.514 per cent total copper and 0.0146 per cent total molybdenum over a length of 1330 feet. The grade of the cross cut over the same length was 0.521 per cent total copper and 0.0171 per cent total molybdenum. These figures are

(1) The basic data on which this figure is based include some analyses made by Elliot Lake which did not include soluble molybdenum as described on Page 9.

in close agreement and indicate these core hole results are reasonably accurate. The holes were drilled under supervision and as carefully as possible. While there was core lost, it apparently was soft material that did not affect the grade. Core recovery was 89.7 per cent by length and 83.1 per cent by weight.

A comparison of the results from the horizontal core holes drilled east-west on section 15N (400 feet to the north of the cross cut) and on section 7N (400 feet to the south) with those calculated from the surface core holes is as follows:

Section	E-W Distance	% Total Cu.	% Total Mo.
15N - Underground holes	1154'	0.423	0.0101
Surface Holes	1154'	0.384	0.0146
Difference		0.039	0.0045
7N - Underground Holes	1188'	0.489	0.0160
Surface Holes	1188'	0.437	0.0147
Difference		0.052	0.0013

Considering all 10 underground holes and assigning the grades of the bulk sampling for gaps between holes, the comparison is as follows:

		% Total Cu	% Total Mo
Underground Holes -	5439'	0.459	0.0163
Assigned (calculated from surface holes)	5439' (2)	0.422	0.0156 (1)
	Difference	0.037	0.0007

All evidence therefore indicates that the grades shown by surface holes in the areas investigated by the underground work are low in both copper and molybdenum if the cross cut grades are assumed to be accurate.

6. Comments on Riocanex' Method of Estimating Tonnages and Grades

Riocanex first developed the grade or trend zones from the vertical cross sections along which the surface core holes had been drilled, by joining groups of analyses of similar grade. The sections were 400 feet apart north-south. The holes were collared at intervals of 350 to 400 feet in the sections and drilled westward at an angle generally of 60 degrees. The trend lines were then drawn on horizontal plans spaced at 80 feet apart

(1) After eliminating three abnormally high samples, representing 30 feet of core and averaging 0.1842 per cent total molybdenum, from Hole 9, from 630 to 660 feet. If these three samples are included the average becomes 0.0165 per cent total molybdenum.

(2) The 5439 feet represent the horizontal equivalent of an approximately equal footage in core holes which has been projected on to the cross cut level.

vertically. The estimates given on Page 3 were based on these basic data as applied to proposed Pit No. 15.

Behre Dolbear made an estimate by measuring areas on the vertical cross sections and estimating the grade on the basis of the trend zones. The tonnage and grade on each section was then assumed to extend 200 feet to the north and 200 feet to the south. Behre Dolbears' average grades agreed with those of Riocanex. The tonnage was 256 million tons as compared to the 274 million tons reported by Riocanex. The Riocanex estimate of tonnage is the more correct.

Riocanex' estimates given on Page 1 may therefore be accepted with confidence.

All these estimates have been based on the analyses from cores of the surface holes and hence are accurate only to the extent that these analyses are representative. This has been discussed on Pages 19 to 21.

7. Volume Per Ton

Riocanex has based its calculations on the assumption that one short ton occupies 12.4 cubic feet. This factor has been based on a specific gravity of 2.58 which is the average of 675 determinations carried out on cores and character samples from underground. A factor of 13 cubic feet has been used for oxidized material.

These factors are considered satisfactory from a practical standpoint.

8. Reduction of High Analyses

High analyses have not been reduced by Riocanex in estimating average grades. This is considered sound practice as high analyses are very seldom abnormally high. In all Behre Dolbears' estimates only three copper analyses and three molybdenum analyses, all from core holes, were considered so abnormally high as to be eliminated from calculations. In each case the core hole followed a narrow high grade vein for an abnormal distance.

New York, New York
July 29, 1968.