FINAL REPORT

TROUT LAKE MOLYBDENUM PROJECT

Revelstoke Mining Division
British Columbia
Canada

by

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January 31, 1983

Newmont Exploration of Canada Limited
Newmont Mines Limited
Vancouver, B. C.

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SUMMARY AND CONCLUSIONS

At the Trout Lake property, located 84 km by road south of Revelstoke in SE B.C., a program of approximately 38,850 m of surface and underground diamond drilling and 2,000 m of underground workings has outlined a molybdenite deposit of 48.7 million tonnes, averaging 0.193% MoS_2 at a 0.10% MoS_2 cutoff grade. Within this tonnage, several higher grade zones are estimated to contain 11.7 million tonnes of 0.362% MoS_2 calculated to a 0.20% MoS_2 cutoff. The deposit remains open at depth, and several areas are suggested for future exploration.

Mining the deposit would require underground methods. A preliminary mining study, using the 0.20% cutoff, suggested open stoping with delayed cemented fill, utilizing the existing adit for mining above the 960 m Level with a shaft and hoisting system below this level. A significant tonnage of better grade material would likely remain in partially unrecoverable crown and sill pillars. As this pipe-like body extends over a vertical interval of at least 1,000 m, unit development costs would be high.

An extensive program of metallurgical testing was carried out on drill core from long surface holes and from bulk samples of the mineralized zone obtained while driving the adit. Results of tests on the core indicated about 90% recovery of the molybdenite in a 90 - 92% MoS_2 concentrate with a relatively simple flow sheet. In tests on the bulk samples the concentrate grade decreased significantly as the recovery increased. This was attributed to fine coatings of molybdenum on gangue particles in the silicified schist, a rock type comprising about 80% of the bulk samples. Test results on the other rock types approximated the results obtained from the core samples.

Although the test results obtained on the silicified schist are disappointing, this problem may not be too significant in considering the whole deposit. An analysis of the mode of occurrence of this rock type suggests that much of it lies outside the potential mineable reserves (0.20% MoS_2 and above). Further test work, which could be done on core samples now at the Danbury lab, is required to determine if all silicified schist has the same characteristics. Also, tests on the bulk sample material grouped by grade would be helpful.

Electric power could be supplied by B.C. Hydro and would require construction of a transmission line approximately 74 km in length. Adequate water is available from Wilkie Creek, possibly supplemented by flow from the adit. Preliminary investigations identified suitable sites for tailings storage, close to the mine. A townsite and related facilities to accommodate most of the employees would be required and would likely be located at Trout Lake.

Environmental and socio-economic studies sufficient to establish a data base for a Stage 1 Report, as required to obtain government approval, were carried out. Assessment of specific impacts would have to be considered when final details of the project are completed. In general it was concluded that the mine could be developed without serious detriment to the existing environment, provided suggested mitigative measures were incorporated during all phases of the project. It is also believed that little or no public opposition to the project would be encountered.

Although no detailed feasibility study was made, based on rough estimates of capital and operating costs the Trout Lake property is not economically attractive at presently projected prices for molybdenum. Further studies of the effect of other cut-off grades and possibly other mining schemes at higher tonnages are warranted.

INTRODUCTION

General

This report describes the results of the underground exploration program carried out at the Trout Lake molybdenum property during 1979 to 1981. The work was done by Newmont Exploration of Canada Limited and Newmont Mines Limited on behalf of a joint venture between Newmont Mines Limited and the Esso Minerals Canada division of Esso Resources Canada Limited.

The report also includes summaries of other work undertaken for the joint venture during the 1979-82 period. Considerable background material on some topics is recorded for future reference whenever the project is reactivated. H. Craig Boyle, project geologist, has written the chapters on Geology, Mineralization, Reserve Estimate, Future Exploration Targets, Underground Program, Bulk Sampling and Assaying. J. Harvey Parliament, President of Newmont Mines Limited, has summarized the sections on Mining, Tailings Storage, Power, Water, Housing, Environmental Studies, and Hydrology and Climate. S. W. Nabbs has provided the summary on Metallurgy. The supporting studies of Newmont staff members F. T. Hancock, W. G. Martin, and K. F. Dahlke are presented in the Appendices. T. N. Macauley organized and edited this report. The individual reports for each study, as well as the background data bases, are on file in Newmont's Vancouver office. The particulars of these reports, together with those pertaining to earlier exploration programs, are provided in the References.

The exploration work on this project was carried out under the direction of T. N. Macauley, Exploration Manager, and R. F. Sheldon, President of Newmont Exploration of Canada Limited. J. H. Parliament and R. S. Mattson of Newmont Mines Limited directed the mining and operations-related studies.

K. F. Dahlke was property manager. Mining engineer F. T. Hancock was involved in the reserve estimate, bulk sampling and mining study. Design of the adit and related facilities was undertaken jointly by Hancock and the contractor, Canadian Mine Services Ltd. All metallurgical testing on this project has been done at the Danbury Connecticut laboratory of Newmont Exploration Limited under the direction of W. C. Hellyer.

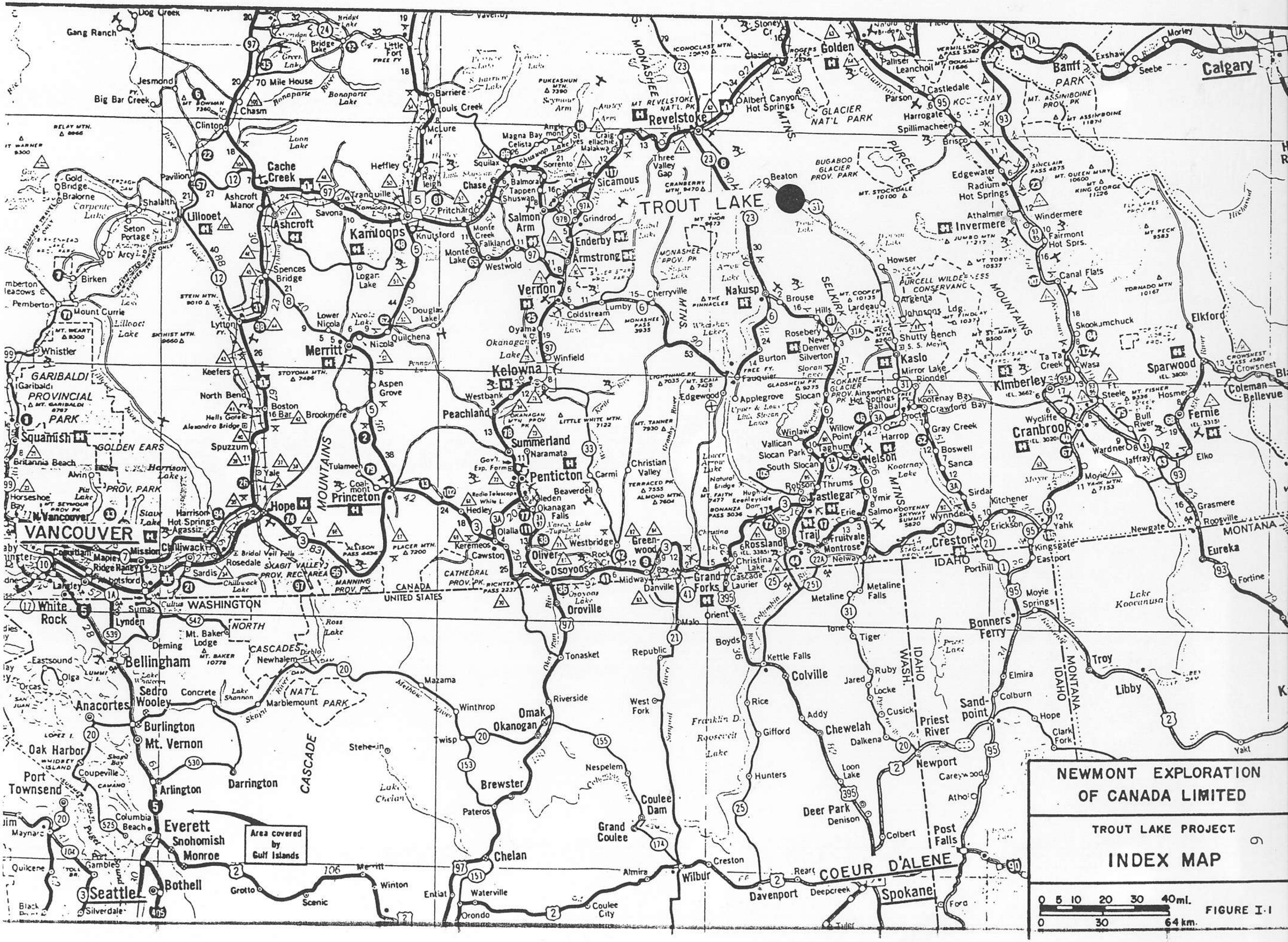
Location and Access

The property is located in the heavily forested Selkirk Mountains of southeastern British Columbia, 3 km west of Trout Lake Village (Figs. I1 and I2). It is within NTS map sheet 82K/12E at latitude 50° 38'N and longitude 117° 36'W.

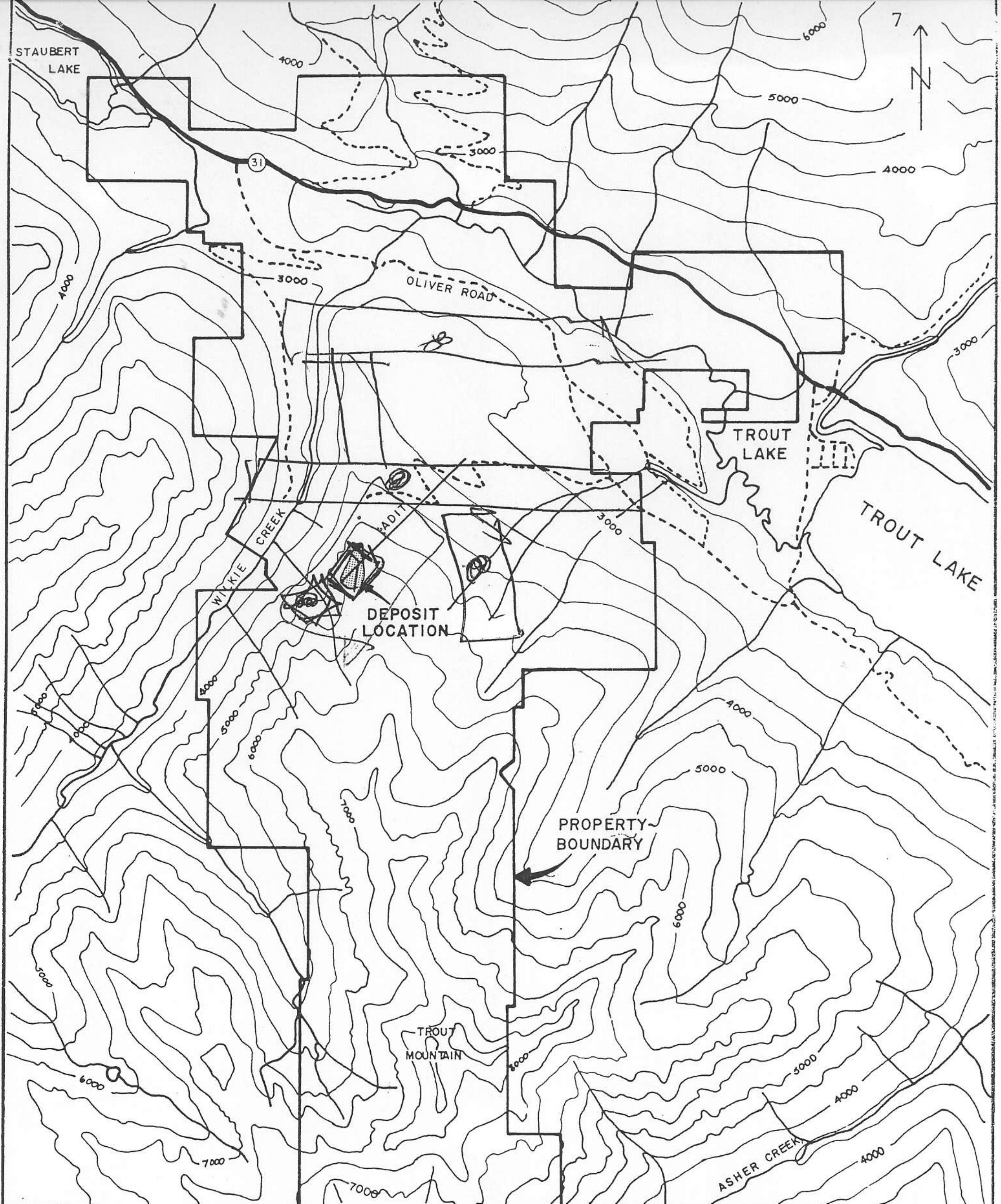
Access to the property is via 84 km (52 mi) of road from either Revelstoke (on the Trans Canada Highway and C.P.R. main-line) or Nakusp (C.P.R. branch line) to the village at Trout Lake. From there, a 6 km logging road leads to the adit portal and campsite (Fig. I2).

History

The property was staked as the "Lucky Boy" and "Copper Chief" claims in 1897 and 1901. Early work concentrated on quartz veins, with 490 tons of Ag-Pb ore being shipped from



NEWMONT EXPLORATION OF CANADA LIMITED
TROUT LAKE PROJECT
INDEX MAP
FIGURE I·1
0 5 10 20 30 40 ml.
0 30 64 km.
TROUT LAKE
Area covered by Gulf Islands



CONTOURS (contour interval = 500 ft.)

ROADS, major secondary

CREEK

0 0.5 1 2 km.

0 1/2 1 mi

NEWMONT EXPLORATION OF CANADA LTD.

TROUT LAKE PROJECT

PROPERTY LOCATION MAP

SCALE	LOCATION	DATE
1:50 000	82 K/12 E	DEC / 82
SURVEY BY	DRAWN BY	NO. FIG I 2

the Lucky Boy. In 1942-43, 23 tons of tungsten ore were sorted and shipped from the Lucky Boy dump.

Molybdenite was reported as early as 1917, but was not the object of exploration until 1969 when Cascade Molybdenum Mines, a subsidiary of Scurry-Rainbow Oil Ltd., optioned the property from prospector Alan E. Marlow. Their bulldozer trenching and 1,000 m of diamond drilling in seven holes outlined a small granodiorite stock with associated molybdenite mineralization. They returned the property to Marlow in 1970 after delimiting the near-surface extent of the deposit, and in the face of rising option payments and a falling molybdenum market.

Newmont personnel had examined the property in 1953 as a tungsten skarn prospect, in 1958 as a silver vein, and in 1969 as a molybdenite prospect. In 1974, Newmont prospector S. W. Barclay obtained Cascade Molybdenum Mine's report and recommended the property for optioning. Individual agreements with owners A. E. Marlow and B. M. Oakey on their separate claim groups were concluded in 1975. The option to purchase Mrs. Oakey's claims was completed in NO ~~1971~~, and payments on the Marlow option are still being made. 1981 ? In 1976, after one season of surface work by Newmont, a joint venture agreement was entered into between Newmont Mines Limited and Esso Minerals Canada. Under this agreement, 15,747 m (51,660 ft) of diamond drilling in 32 drill holes was carried out from 1976 to 1979, outlining a significant molybdenite deposit. In 1979 a decision was made to continue the exploration of the deposit by means of an adit and drifts and a substantial underground drilling program.

Underground Program

The underground program was the next step required in making a preliminary evaluation of the economic feasibility of the Trout Lake Moly deposit. The adit and drifts were designed to penetrate the core of the deposit for the specific purposes of:

1. providing access for concentrated diamond drilling to define reserves,
2. providing bulk samples of the deposit for comparison of their grades to drill hole grades,
3. providing material for metallurgical testing,
4. establishing controls and continuity of mineralization,
5. making a study of mining methods and costs,
6. allowing exploration of previously inaccessible areas.

The mining portion of the program comprised a total of 2,000 m (6,560 ft) of adit, crosscut and drifts that tested the deposit at a depth of 490 m (1,600 ft) below its surface outcrop. Of the 22,120 m (72,570 ft) of underground diamond drilling, 15,357 m detailed the mineralization indicated by surface holes, and 6,259 m explored the areas around and under the deposit for additional reserves. Drifting began with the construction of the portal in September 1979 and was completed in April 1981. Diamond drilling ran from November 1980 to November 1981. At the height of the program the on-site personnel totalled 40. They consisted of manager, 3 geological staff, 2 surveyors, 2 labourers and 32 contractors' employees.

Bulk samples were taken from 189 drift rounds with a total length of 687 m through the deposit. They were processed through a sampling tower which reduced a 25 tonne sample from the 100 tonne drift round to 90 kg (200 lb) of material crushed to minus 6 mm (¼ in). Of this amount, 22 kg were shipped to the Newmont lab and 68 kg were stored on the property. The assays from the bulk samples when compared to those from the pilot diamond drill holes provided useful criteria for extrapolating drill hole grades to large volumes of rock.

A preliminary mining plan has been developed on the basis of the combined underground and surface diamond drilling and the experience of the underground drifting. A convenient tailing storage area was recognized on the property.

Throughout the period of the underground program, consultants were engaged in the investigation and monitoring of environmental and climatic conditions, water supply, and a socio-economic impact (townsite) study.

Claims

The Trout Lake property consists of 6 Crown Grants, 1 Mineral Lease, 18 located "two post" claims, 12 fractional claims and 24 "Modified Grid System" claims totalling 172 units, all either covered by the option agreement with A. E. Marlow or owned by Newmont Mines Limited directly. A complete listing of the claims is given in Table I-I.

The property outline is shown on Fig. I2 and individual claims on the large map (Fig. I3). Assessment work has been filed to the maximum level permitted, so that all claims are in good standing to 1992.

It should be noted here that Ash 1 and 2 claims at the south end of the property lie beyond the 3 mile perimeter

TABLE I-I

TROUT LAKE CLAIMS

Claim Name	Type of Claim	Lot Lease or Record No.	No. of Units	Expiry Date
CH	Crown Grant	L4741	NA	Taxes Due
XYZ	" "	L4742	"	
CD	" "	L4743	"	July 1 of
Blue Jay	" "	L4744	"	
Doubtful	" "	L4745	"	Each Year
LB	" "	L5423	"	
Horseshoe	Mineral Lease	24	NA	Rental Due Aug. 23 of Each Year
Anex	"2 post" located claims	182	NA	Sept. 5, 1992
Lucky Jay No. 1 & 2		9889, 9890	"	Aug. 8, 1992
Lucky Jay No. 3	"	9916	"	Sept. 16, 1992
Lucky Jay No. 6	"	9968	"	Sept. 26, 1992
Lucky Jay No. 7	"	9969	"	Sept. 26, 1992
Lucky Jay Nos. 9, 10 & 11	"	9971-9973	"	Sept. 26, 1992
Rover Nos. 2,3,4,5,6 &7	"	10002-10007	"	Nov. 6, 1992
Copper Chief Moly	"	5657M	"	Sept. 13, 1992
Copper Chief Moly 1 & 2	"	5658M, 5659M	"	Sept. 13, 1992
LB Fraction	located fractional claims	4246	"	Sept. 21, 1992
Czar Fraction		683	"	July 20, 1992
Suzy Fraction	"	681	"	July 20, 1992
Kimo Fraction	"	685	"	July 20, 1992
Francis Fraction	"	680	"	July 20, 1992
Brigitte Fraction	"	682	"	July 20, 1992
Irene Fraction	"	684	"	July 20, 1992
U.P.I. Fraction	"	814	"	Nov. 8, 1992
Sliver 1 Fraction	"	815	"	Nov. 8, 1992
Snow No. 1 Fraction	"	838	"	Dec. 19, 1992
Linda Fraction	"	969	"	June 7, 1992
TL 21 Fraction	"	1520	"	Oct. 29, 1992
Fog 1	Modified Grid System	430	4	Dec. 22, 1992
Fog 2		431	4	Dec. 22, 1992
Fog 5	"	751	2	Oct. 12, 1992
TL 1	"	26	4	Oct. 2, 1992
TL 2	"	27	6	Oct. 2, 1992
TL 3	"	28	1	Oct. 2, 1992
TL 4	"	414	10	Dec. 1, 1992
TL 5	"	415	14	Dec. 1, 1992
TL 6	"	416	15	Dec. 1, 1992
TL 7	"	417	4	Dec. 1, 1992
TL 8	"	418	2	Dec. 1, 1992
TL 9	"	419	4	Dec. 1, 1992
TL 10	"	440	2	Feb. 23, 1992
TL 11	"	441	8	Feb. 23, 1992
TL 12	"	442	4	Feb. 23, 1992
TL 13	"	443	20	Feb. 23, 1992
TL 14	"	444	6	Feb. 23, 1992
TL 15	"	445	6	Feb. 23, 1992
Ash 1	"	446	10	Feb. 23, 1992
Ash 2	"	447	15	Feb. 23, 1992
TL 17	"	829	8	Nov. 22, 1992
TL 18	"	879	8	Feb. 19, 1992
TL 19	"	1177	9	Mar. 9, 1992
TL 20	"	1178	6	Mar. 9, 1992

pertaining to the Newmont-Esso agreement. They were staked and explored under the Kuskanax Survey, and therefore both equity and costs in these two claims are divided equally between Newmont and Esso.

Claims held by other parties include the Kodiak No. 1 to 5 fractions held by Kodiak Resources Ltd., all within the Trout Lake claims but not of critical importance. Oakey Holdings Ltd. holds the Oakey and Fog 4 claims and the Oakey 4 and Snow 2 fractions. An agreement has been negotiated with this owner to allow the use of the surface of these claims during any mining operations on the Trout Lake property. The Hum 1, 2, 7, 8, 9, 10 and 11 are held by Cominco; and Amax owns the Lemar 3 claim.

Cost Summary

Total costs of the Trout Lake Project from its inception in 1975 up to December 31, 1982 are $14,901,928. These may be subdivided as follows:

Surface Program 1975 - 1979 Mainly diamond drilling; also includes geological mapping, geochemical and magnetometer surveys, prospecting, trenching and property payments.	$ 2,266,692

Underground Program 1979 - 1982		
Adit	$ 6,483,183	
Diamond Drilling	4,055,177	
Bulk Sampling	309,008	
Metallurgical Testing	128,079	
Consultants' Studies (environmental, water supplies, etc.)	272,990	
Other (property payments surveying, management fees)	1,386,799	$12,635,236
Combined Programs		$14,901,928

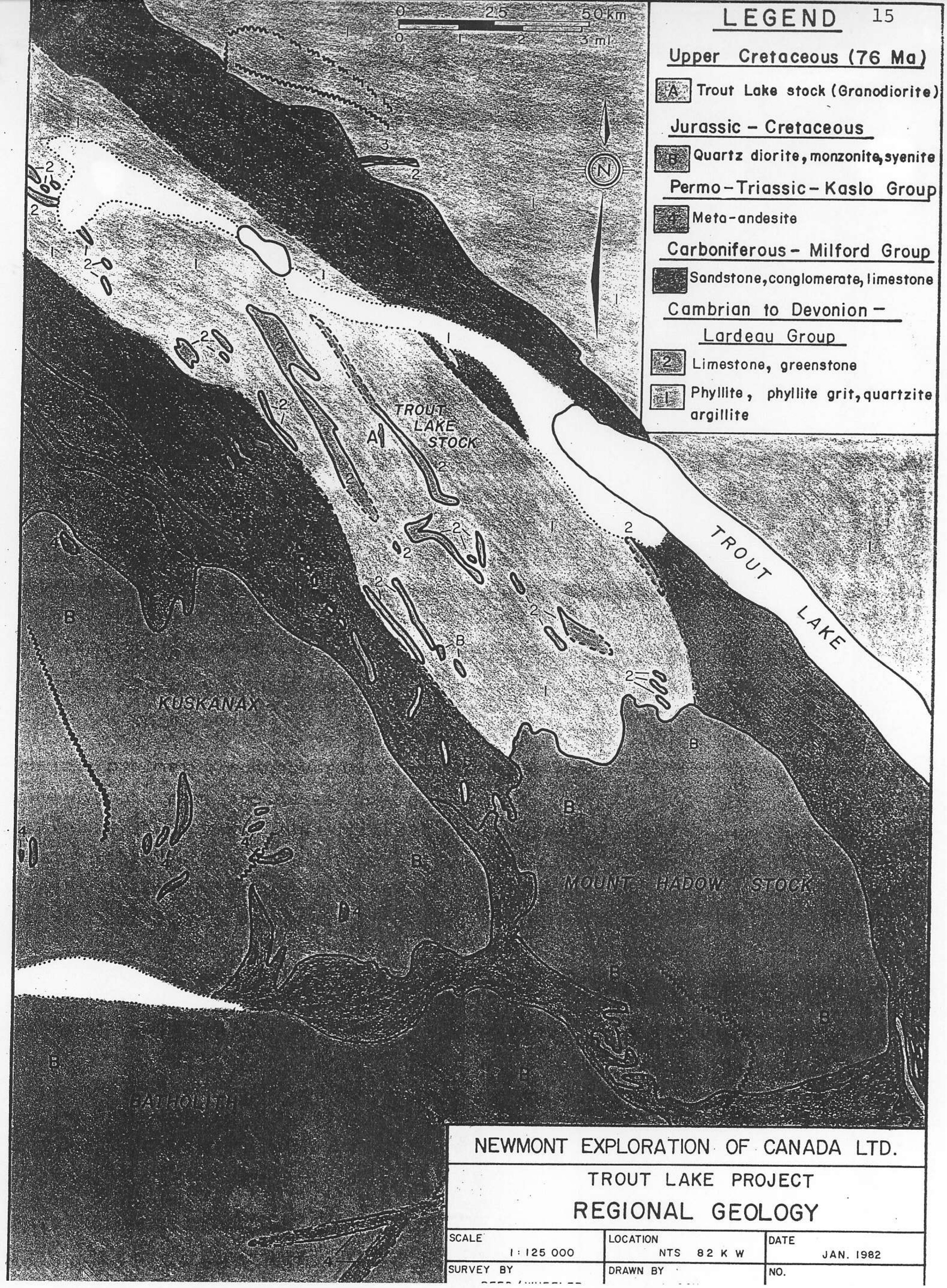
Property payments to the two claim owners to December 31, 1982 have been $55,500; another $59,500 remains to be paid on the one purchase option still outstanding.

GEOLOGY

Regional Geology

The geology of the area has been reported on by Brock (1903), Emmens (1914), Gunning (1929), Holland (1952, 1953), and Fyles and Eastwood (1962). The most recent work is mapping and compilation of the Lardeau, West Half, Sheet by Read and Wheeler (1976), from which Fig. Gla is taken. The geology in the immediate vicinity of the deposit is based on detailed mapping by Macauley, Boyle and Waterhouse (1976-1979) and more regionally by Hill (1980) and Psutka, Fyles and Read (1981). Considerable advantage was taken of Drs. Fyles' and Read's backgrounds in Lardeau area geology.

The property lies near the north end of the Kootenay Arc, a belt of highly deformed, heterogenous sedimentary rocks bowed around the E margin of the Nelson and Kuskanax batholiths. The oldest rocks of the district around the deposit are the unfossiliferous Lardeau Group phyllites, quartzites and schists, with minor limestone and greenstone. Regionally, these rocks overlie the Lower Cambrian Badshot Formation and are unconformably overlain by the Upper Mississippian Milford Group consisting of a basal conglomerate, limestone, phyllite, and micaceous metasandstone. The Jurrasic Kuskanax Batholith, an aegirine-augite leucoquartz monzonite dated at 178 million years (Ma), lies 5 km to the S of the property. A series of calcalkalines stocks of Jurrassic to Cretaceous age (150 - 74 Ma) occur in the Kootenay - Upper Arrow Lake area, several of which have associated molybdenite mineralization. The Trout Lake stock, dated by K/Ar on biotite as 76 Ma, is one of these stocks and forms the focus of the Trout Lake deposit.


0 2.5 5.0 km
0 1 2 3 mi
LEGEND
Upper Cretaceous (76 Ma)
A Trout Lake stock (Granodiorite)
Jurassic - Cretaceous
B Quartz diorite, monzonite, syenite
Permo-Triassic-Kaslo Group
4 Meta-andesite
Carboniferous - Milford Group
Sandstone, conglomerate, limestone
Cambrian to Devonion - Lardeau Group
2 Limestone, greenstone
1 Phyllite, phyllite grit, quartzite argillite
N
TROUT LAKE STOCK
TROUT LAKE
KUSKANAX
BATHOLITH
MOUNT HADOW STOCK
NEWMONT EXPLORATION OF CANADA LTD.
TROUT LAKE PROJECT
REGIONAL GEOLOGY
SCALE 1:125 000
LOCATION NTS 82 K W
DATE JAN. 1982
SURVEY BY
DRAWN BY
NO.

Local Geology

The surface geology of the property is shown in Fig. Glb. The subsurface geology is shown in a series of sections and level plans from Fig. G2 to Fig. G16. Detailed underground mapping and drill core logging is presented in Fig. G2 of the 960 Level Plan.

The geology of the property is basically that of intersecting dykes of granodiorite to quartz diorite composition cutting the steeply NE dipping Lardeau rocks and coalescing downwards into a larger, ill-defined, granodiorite stock.

A series of near vertical NW trending faults divide the Lardeau into uncorrelatable panels in the vicinity of the deposit. The N trending Z Fault appearing in one of these panels, bounds the deposit (as presently known) on the east side. It probably possesses a vertical displacement in the hundreds of meters.

Alteration of the enclosing country rocks by a combination of thermal and hydrothermal processes has resulted in the development of schist and skarns and silification in the region around the Trout Lake stock. Closer to and associated with the contacts between the dykes and the country rock are intense quartz stockworks which carry most of the molybdenite mineralization.

Rock Types

The rocks at Trout Lake can be divided into the Lardeau Group rocks and intrusive rocks, and the altered equivalents of both. Lardeau rocks can be further divided into clastic, carbonate and meta-volcanic units.

a) Lardeau Group Clastics - These rocks are highly variable clastic sediments which are traceable for only a few 100 metres. In addition, most of these units contain irregular knots and lenses of granular quartz with chlorite and frequently pyrrhotite. The major rock types are listed below.

Argillites	- aphanitic to very fine grained dark and light grey argillites, usually displaying a clear lamination.
Slate	- dark grey to black shaley slate, frequently with an uneven flaggy cleavage.
Schists	- weakly developed, fine grained, limy or quartzitic chlorite and biotite schists.
Quartzites	- dark to light grey or white, phyllitic or micaceous, fine to medium grained quartzites or quartz grits.
Silicified Schists	- fine to medium grained chlorite, sericite and biotite schist infused with varying amounts of silica to produce a hard well indurated rock as a product of thermal and hyrdothermal alteration associated with the Trout Lake stock and deposit.
Biotite Hornfels	- very fine grained, massive, hard, brown to purplish-brown biotite hornfels associated with carbonate units. Frequently cut by quartz pyrrhotite veins with chlorite envelopes. These rocks usually occur within skarn zones or calc-silicate schists.

b) Lardeau Group Carbonates - Carbonate rocks are interbedded with various members of the clastic units. Though they provide the clearest markers, correlation is extremely difficult because of their number, variability and structural complexities.

Limestones	- vary between massive white or black with white calcite stringers to grey or blue-grey and white banded, occasionally interbedded with various clastic units.
Dolostone	- massive, white to grey, fine to coarse grained dolomite, frequently developed within limestone horizons.
Calc-silicate Schist	- light green to purple calc-silicate with chlorite and biotite schists in a "zebra" like texture associated with carbonate horizons.
Skarn	- includes a variety of rocks from a green grey mottled calc-silicate hornfels to a light and dark green diopside garnet skarn, frequently carrying abundent pyrrhotite and occasionally scheelite. Development of the skarns is clearly associated with the intrusive and hydrothermal activity.

c) Lardeau Group Meta-Volcanics - These are of very limited extent, occurring as narrow slivers in surrounding rocks and only occasionally forming units of significant size. They are correlated with the meta-volcanics of the Jowett Formation, a member of the Lardeau Group.

Greenstone	- dark green limy chlorite schist and amphibolite, frequently with fine grained euhedral disseminations of magnetite. Outcrops are frequently stained a dark rusty purple from iron and magnesium oxides.

The degree to which these rocks have been affected by the Trout Lake deposit is unclear.

d) Intrusive Rocks - Intrusive rocks associated with the Trout Lake stock vary in composition from quartz diorite to granodiorite and aplite. The intrusive rocks are all thought to be genetically related, but their relationship is confused by contradictory evidence and alteration. However, the apparent sequence from oldest to youngest follows the order presented below.

Granodiorite - grey, medium grained, porphyritic rock with 10% large euhedral (to 0.5 cm diameter) quartz "eyes" in a seriate groundmass of euhedral plagioclase phenocrysts (35%), anhedral quartz (35%), K-feldspar (10%), and sericitized/chloritized biotite relics (10%). In most cases the granodiorite has undergone further hydrothermal alteration, resulting in a pale yellow-green colour due to saussuritization of feldspars and development of sericite, frequently with the accompaniment of disseminated fine, euhedral pyrite. In some cases, the alteration is so intense as to leave only quartz, sericite and pyrite in a porphyritic felted texture. Near contacts, the granodiorite occasionally displays a compositionally banded texture of high and low biotite content parallel to contacts. This texture suggests flow banding and contamination by wall rock inclusions during emplacement.

Aplite - these generally consist of quartz and white K-feldspar in a fine to medium grained sugary texture, occasionally with some minor sericite. It is often transitional to quartz feldspar pegmatite veins and on occasion has fine to coarse grained disseminated molybdenite in euhedral flakes and rosettes.

Quartz Diorite A - dark grey to grey porphyritic rock with phenocrysts of quartz and finer plagioclase in a finer, more distinct, biotite bearing groundmass than that of the granodiorite. It contains slightly less quartz (35%), more plagioclase (45%), less K-feldspar (5%), more biotite, with rare hornblende phenocrysts and late magmatic K-feldspar porphyroblasts. It generally has a fresher, salt and pepper appearance and frequently contains biotitic inclusions of altered silicified schists.

Quartz Diorite B - transitional between the granodiorite and the quartz diorite A, tends to be lighter in colour than quartz diorite A but darker than the granodiorite, texture is less porphyritic but still has a finer grained groundmass and more prominent biotite than the granodiorite. Quartz diorite B less commonly has inclusions of altered silicified schist.

The distinction between the intrusives, particularly the quartz diorites, can be subtle and when viewing them individually, their identification tends to be subjective. Only when seen side by side in crosscutting relationships do the characteristics of the various intrusives stand out.

Structure

The structural geology of the Trout Lake area is dominated by the regional NW trending fabric of this part of the Cordillera. This involves large scale regional folding, with at least two phases in the Lardeau rocks, accompanied by strong faulting. Superimposed on these structures is a network of dykes and veins resulting from the emplacement of the Trout Lake stock and its associated mineralized zone.

Foliation throughout the area, as measured from surface outcrops and from the underground workings, strikes NW and dips moderately to steeply (50° - 90°) to the NE. Occasional reversals in dip in the 70° - 90° SW range have been noted, but the lack of a consistent pattern suggest that these are simply local variations.

The country rocks of the Lardeau group have been deformed repeatedly but first phase pre-Mississippian folds have been completely obliterated in all but the rarest cases by second phase Middle Jurassic folds and accompanying metamorphism (Psutka et al. 1981). The second phase folds are moderately closed to isoclinal, slightly overturned, display a reverse-N geometry when viewed down plunge, and have undulating fold axes which vary from sub horizontal in the SE, to plunging 25° - 45° NW in the NW. Second phase folds are most clearly outlined by the carbonate horizons, and the interpretation of combined diamond drill hole data and surface mapping has indicated a thick skarny carbonate unit showing the scale and geometry of second phase folds in the area of the Trout Lake deposit. This folded carbonate unit can be seen on Sections 1 through 12 in Fig. G5 to G16, abutting against the steeply SW dipping Ethel Fault.

The area in which most exploration and diamond drilling has been done to date seems to be less strongly folded to the SW of the Ethel Fault (at least at the scale of these folds). There are however numerous areas of minor folds of a few meters scale which display second phase fold geometry within this exploration area.

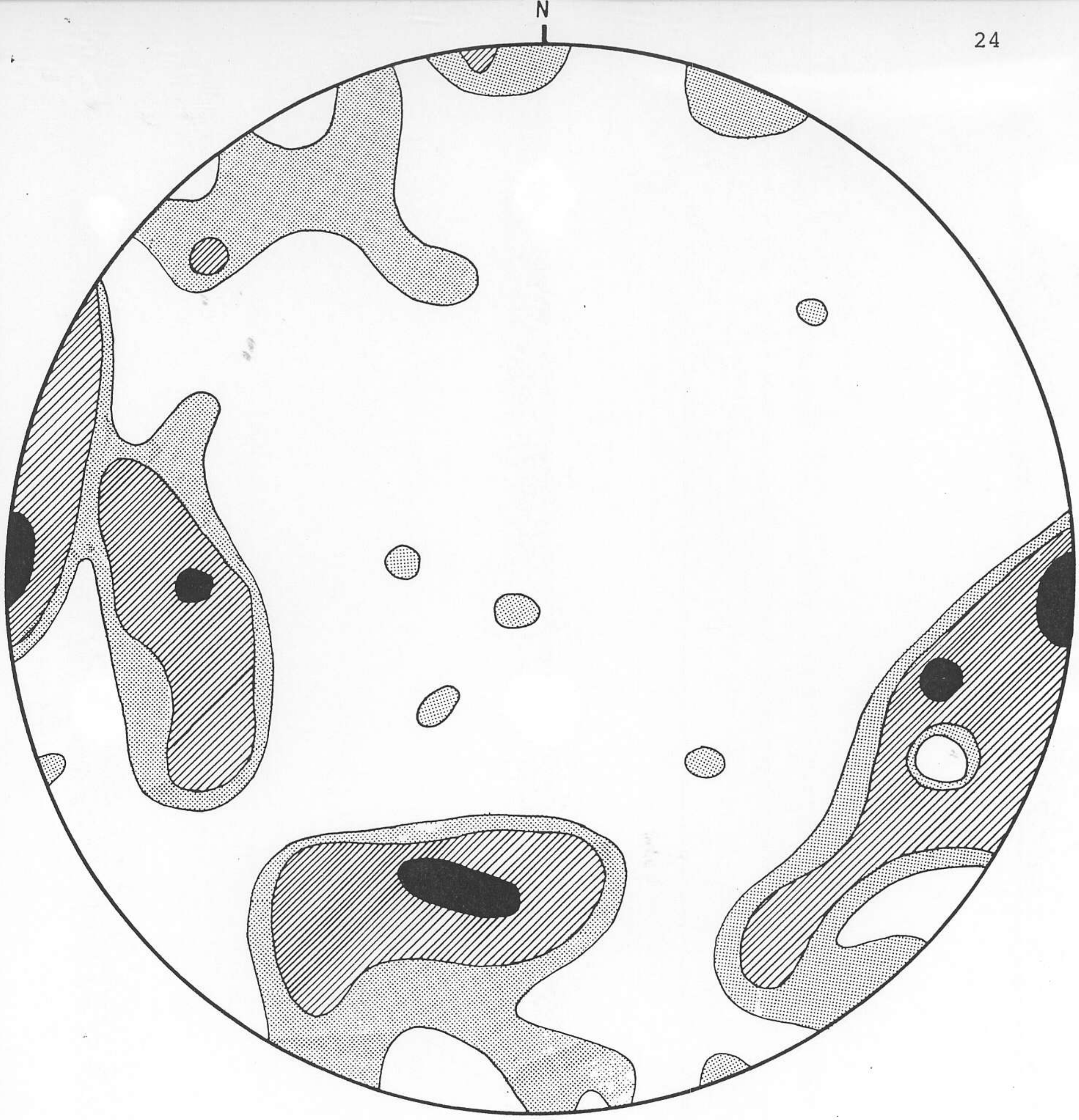
The Ethel Fault is one of five NW striking steeply dipping post deformational faults that divide the area into five panels. Because the Lardeau rocks are so heterogeneous and repetitive, it is difficult to determine the magnitude and direction of offsets on these faults. However, structural considerations such as that suggested across the Ethel Fault in the twelve sections, and interrupted metamorphic and alteration patterns, indicate displacements are large and normal dip-slip (Psutka et. al., 1981).

The strongly developed and economically important Z Fault strikes about 10°E of N through the area of the Trout Lake deposit with a near vertical dip. It appears to be a branching extension between two of the dominant NW trending faults, twisting to the NW into the Adit Fault at the northerly end and to the SE into the Ethel Fault at the southerly end. Structural constraints, interrupted metamorphic and alteration patterns, and offset intrusive contacts, all suggest a dip-slip movement downwards on the east side of the Z Fault in the 100's of metres. (Slickensides give numerous conflicting directions of movement with no dominant pattern). While there has definitely been movement on the Z Fault after emplacement of the Upper Cretaceous Trout Lake stock, it is entirely possible that the Z Fault could have been present before this and possibly provided a zone of weakness as a focus for the intrusion of the stock.

The area of mineralization is cut by numerous small faults and shears which appear to have a similar pattern as, and perhaps genetically related to the mineralized quartz veins. As seen in the underground workings and drill core, these faults vary widely in appearance and intensity. They can be wide zones filled with rubble and wet clayey gouge, to dry and shattered. They can be as narrow as a few centimetres or as wide as several metres. Sometimes they contain mineralized rubble and other times they are barren; they may cut a mineralized zone off in one area and be well mineralized in an otherwise barren area. In other words, the minor shears can be so varied in their appearance, especially in drill core, that it is difficult to correlate between them and only general patterns are suggested by a detailed survey of shears mapped in the adit and drifts.

A lower hemisphere stereonet plot of poles to shears is shown in Fig. G17. Such a plot is constructed by projecting an imaginary line which is perpendicular to the plane of the structure in question (a "pole"), downwards to intersect an imaginary half sphere or "lower hemisphere" at a point. The point is then projected vertically upwards to a horizontal plane. For example, a structural plane, such as a shear, vein, bedding, joint, etc., striking NW-SE and dipping to the SW would be represented by a point plotting in the NE quadrant of the stereonet plot. Its location falls along a line which is at 90° to the strike of the structure. The dip of the structure determines the precise location of the point, varying between 0° dip at the center of the circle to 90° dip on its circumference.

As any particular attitude results in a unique point, a number of particular structures such as shears can be represented on a single stereonet plot. The frequency and relative



Lower Hemisphere Stereonet Plot of Poles To Shears
101 measurements

LEGEND

- Low density
- Medium density
- High density

NEWMONT EXPLORATION OF CANADA LTD.		
TROUT LAKE PROJECT		
STEREONET PLOT: POLES TO SHEARS		
SCALE	LOCATION	DATE Aug 10/82
SURVEY BY H. C. Boyle	DRAWN BY J. Goh	NO. G 17

importance of a particular attitude can then be determined by contouring the number of points which fall within a specific area, usually 1% of the area of the entire stereonet plot. In the data presented here, and in the following section on quartz vein distribution, the results are presented only in broad relative proportions (high, medium and low density). This is because of the variability of the data base as the measurements were taken from place to place throughout the underground workings.

The pattern of poles to shears outlines a pattern of steeply dipping shears (60° and greater) striking in a semi-random distribution. The areas of high concentration do, however, indicate slightly more prominent sets of shears. One is an EW striking set, dipping 60°N. The other set strikes N-S to just east of N, dipping between 70°E and 70°W. This set includes the strongly developed Z Fault and indicates a local pattern of minor faulting related to the Z Fault.

This pattern apparent in the stereonet reinforces the impressions of faulting seen on the cross sections as splays and branches off the Z Fault. In the underground workings the minor shears seem to have small offsets associated with them resulting in displacements of only a few metres or tens of metres. They appear to be relatively minor responses and readjustments to the stresses generated by the emplacement of the Trout Lake stock and its associated mineralization. Thus, though they may have acted as focuses for intrusive dykes and channel ways for mineralizing fluids, these minor faults have not displaced large zones of mineralization or affected the shape of the deposit.

The Trout Lake stock, as mentioned earlier, has probably been emplaced along the Z Fault, as well as having been subsequently displaced along it. The numerous dykes and dyklets also seem to be influenced by the foliation fabric and minor faults. Most of the granodiorite dykes appear to strike NW, sub-parallel to foliation with near vertical dips and apparent steep NE dips in a few cases. Vertical, NW-striking orientations are also common for the two types of quartz diorite dyklets, particularly the 'A' type; but these dykes, as well as the aplite dykes, are dominated by NE sets that also frequently have shears associated with them.

The orientation of faults and shears as well as the fabric of the surrounding country rock and emplacement of the intrusive have a correlation with the stockwork system of quartz veins which constitute the mineralized zones. However, a discussion of this is deferred to the section on mineralization.

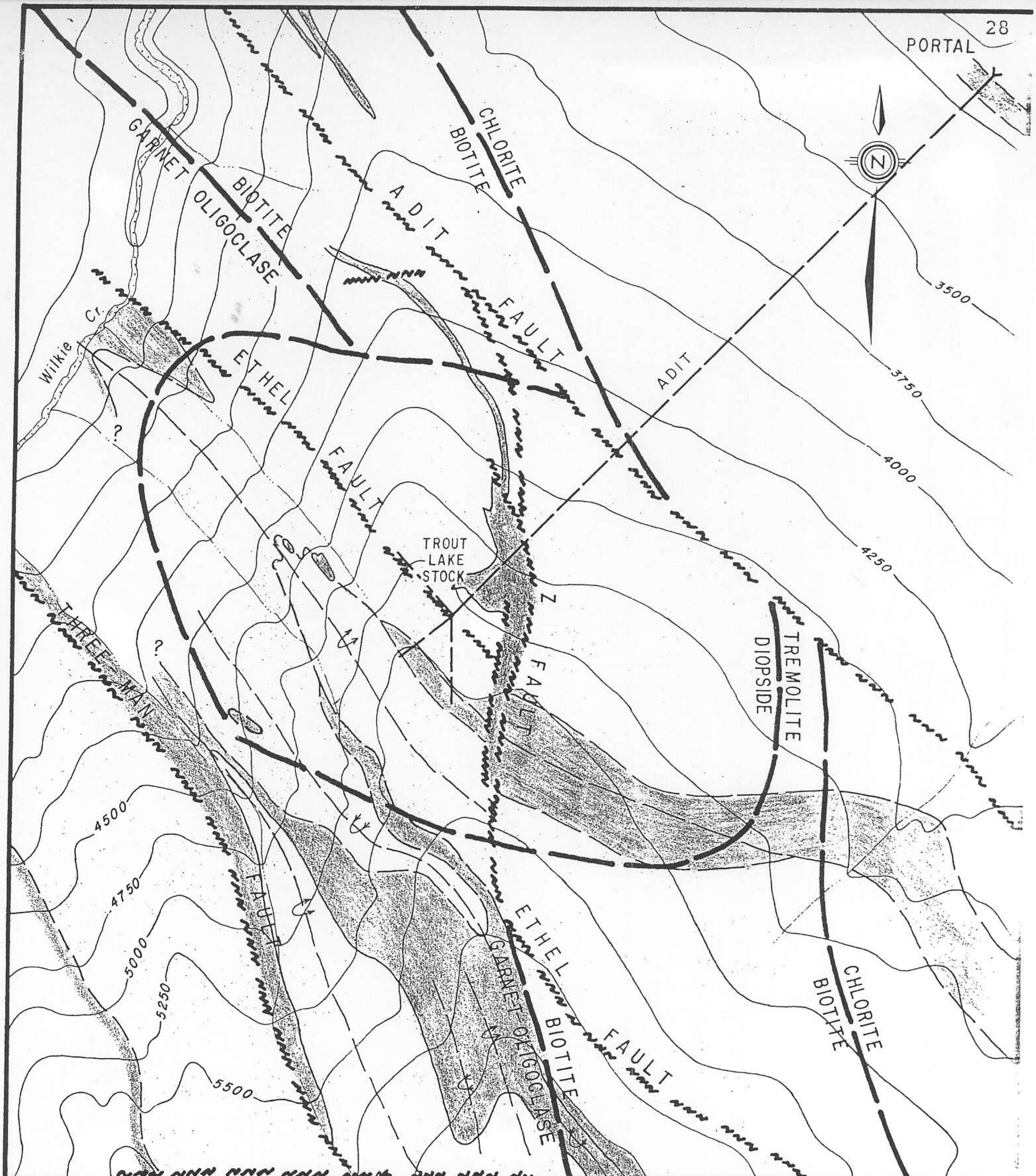
Metamorphism

A study of the regional metamorphism was undertaken in an effort to better define and restrict in a regional sense the best areas for exploration. Contact metamorphism and hydrothermal alteration present excellent guides to mineralized targets, but in the Trout Lake area the pattern of these guides is unclear because they overprint regional metamorphic patterns with similar mineralogy. The object of establishing the regional metamorphic pattern, therefore, was to identify and segregate those contact metamorphic and alteration effects which reflect the presence of Trout Lake stock and its associated mineral deposits.

Regional metamorphic patterns have been established through mapping and petrographic work correlated with broad patterns within the Kootenay Arc. This includes several seasons work by a number of Newmont geologists supplemented by two weeks of surface and underground mapping by geologists from Geotex Consultants Ltd. under the direction of Dr. P. B. Read. Petrographic studies were based on 104 thin sections of rocks collected by Newmont and Geotex over a 15 km^2 area around the deposit.

Regional metamorphism in the area is associated with a mid-Paleozoic, low grade metamorphic event that was overprinted by mid-Jurassic metamorphism developed after the second phase deformation. It increases from E to W, with steeply dipping regional metamorphic isograds. The lowest grade chlorite zone extends for over 20 km, from well out of the Trout Lake area to just E of the deposit. It is succeeded westwardly by the narrower biotite zone, which is only 600 m wide and characterized by fine grained, foliated, dull brown biotite. The biotite zone is succeeded by the highest grade garnet-oligoclase zone which extends W from the approximate location of the Trout Lake stock for more than 3 km to the Kuskanax Batholith. The boundary of the garnet-oligoclase zone was somewhat difficult to locate as it depends on the presence of either of two critical assemblages, one involving almandine garnets and one involving oligoclase. These were never observed together. Consequently, the actual location of this zone is indefinite. The position of the isograds may be seen in Fig. G18 which is taken from Geotex's report and simplifies the geology in the immediate vicinity of the Trout Lake stock.

Contact metamorphism associated with the emplacement of the Trout Lake stock and the stockwork molybdenum deposit is represented in Fig. G18 by the diopside zone, defined by the first appearance of diopside and outlined by the tremolite-



LEGEND

- GRANODIORITE, QTZ DIORITE
- PHYLLITES, SCHISTS, GRITS, QUARTZITES
- LIMESTONE-DOLOMITE; SKARN, CALESILICATE SCHISTS
- METAVOLCANICS AMPHIBOLITIC
- METAMORHIC ISOGRAD
- FAULT
- OVERTURNED ANTIFORM
- OVERTURNED SYNFORM

NEWMONT EXPLORATION OF CANADA LTD.

TROUT LAKE PROJECT

METAMORPHISM

0 100 200 300 400 500 600 700

SCALE IN METRES

SCALE: 1:10,000	LOCATION: NTS 82K/12E	DATE: AUG. 26, 1982
SURVEY:	DRN. BY:	DWG. No.:

diopside isograd. This tremolite-diopside reaction is dependent on suitable bulk rock composition and is not valid in rocks deficient in either quartz or calcite. The diopside is thus better developed in the carbonate horizons than in the more extensive phyllitic rocks. Also, the carbonate horizons may be more altered since they may have acted as better conduits for channeling the hydrothermal fluids. In any event, this diopside alteration zone is one indicator of the emplacement of the Trout Lake stock. Centered as it is on the stock, it possibly delimits the area most favourable to exploration.

The diopside zone as shown in Fig. G18 (adopted from Geotex's report) was drawn very crudely, based on only a limited number of rock samples taken in this area. It overstates the zone on the E side of the Z Fault and masks the offsetting effects of the fault. In their report, Geotex state when discussing the Z Fault that movement on the fault is "dominantly dip-slip, and the limited amount of skarn and hornfels on the E side of the fault implies that the rock E of the fault moved relatively down". Here the "skarn and hornfels" correlate to the "diopside zone" with hornfelsing defined by an area in which the fine grained micas have an "unoriented" texture. The report further clearly states that, "the diopside zone is offset by the Z Fault...." and continues that "the diopside isograd should be defined better in plan and section.... partially because it.... may assist in determining the displacement on the Z Fault". If the diopside alteration is strongly vertically oriented, this may explain why it is difficult to discern any effect by the vertically offsetting Z Fault.

Alteration associated with the Trout Lake deposit is not restricted to the development of the diopside zone described above. It also includes the more widely recognized zones of phyllic and potassic alteration characterized by the development of quartz-sericite in the former and K-feldspar (alkali feldspars) plus or minus biotite in the latter. These show a close relationship with mineralization and will be discussed more thoroughly in the following chapter on that subject.

MINERALIZATION

General

The Trout Lake deposit, as outlined by drilling and drifting to date, is a stockwork molybdenite deposit of 48.7 million tonnes of 0.193% MoS_2. This mineralization occurs in four zones (A, B, D, F) defined by a 0.10% MoS_2 cutoff. They constitute an irregular, steeply SW plunging, pipe-like body that extends from a small surface exposure down to a depth of 980 m (3,215 ft). Maximum horizontal dimensions are 280 m by 160 m (920 x 525 ft). Included in this reserve is 11.7 million tonnes of plus 0.20% MoS_2 material with an average grade of 0.362% MoS_2. This high grade material occurs as cores within the four zones, with the bulk of it occurring in two closely spaced bodies within the largest zone (B). Together these two make up 9.3 million tonnes at 0.357% MoS_2, extending for 565 m (1,850 ft) from the 615 (2,020 ft) to 1,180 m (3,870 ft) elevations. They have a maximum combined horizontal dimension of 110 m by 200 m (360 ft by 660 ft).

The Trout Lake deposit is classified as a stockwork molybdenite deposit of the general Climax type, sharing many characteristics with this and other well known molybdenum deposits in Colorado and B.C. It has an extensive quartz stockwork system associated with multiphased intrusions of intermediate to acid composition, a well developed alteration sequence centered on the deposit and a simple mineralogy with an apparent halo of associated metals. There are, however, aspects of each of these characteristics which are unique to Trout Lake. The principal aspects in which Trout Lake differs from other deposits are the composition of the intrusive phases and the influence of the enclosing metasediments. The more common association in other deposits is with intrusive rocks of granite to quartz monzonite composition, while at Trout Lake the stock is composed of granodiorite to quartz diorite. A strong structural control by the metasediments has resulted in a preferred orientation for the important granodiorite dykes and one set of quartz veins. The steeply dipping foliation at Trout Lake probably contributed to the strong vertical attenuation of the deposit. These influences are not generally apparent at other molybdenum deposits.

Mineralogy

The Trout Lake deposit is mineralogically simple, with molybdenite the only economic mineral. Pyrite and pyrrhotite accompany the molybdenite mineralization and constitute the most abundant sulphides, averaging about 1 to 2% throughout the deposit and running as high as 10 to 15% combined in the higher grade zones. Pyrite predominates on the margins of the deposit whereas pyrrhotite is the more abundant sulphide in the centre of the deposit. Very minor amounts of chalcopyrite (only up to 0.5% in concentrates) and rare traces of sphalerite, galena and scheelite have been noted within the mineralized zones. Economically significant concentrations of these minerals are only found in vein systems or skarn zones outside the molybdenum deposit.

Molybdenite occurs principally as fine to medium grained (rarely coarse grained) flakes and rosettes disseminated in quartz veins along their margins or as trains within the veins. The iron sulphides occur as disseminations or aggregated masses and knots, and molybdenite is frequently intimately associated with them. Limited polished section work indicates that the molybdenite replaces both pyrite and pyrrhotite.

Also accompanying the sulphides in the quartz veins are accessory silicate minerals, noted in about half the observations. Fine white to green sericite is the most common, followed by white alkali feldspars. These feldspars range in composition from albite (Ab_{100} to $Ab_{90}Or_{10}$), to an intermediate alkali feldspar between albite and orthoclase ($Ab_{90}Or_{10}$ to $Ab_{70}Or_{30}$), to orthoclase proper ($Ab_{60}Or_{40}$ to Or_{100}). Less common are chlorite and carbonate; biotite (presumably secondary) is rare. The white alkali feldspar is usually distributed along the vein walls and sometimes coats them completely, creating a zoned quartz-feldspar vein. Occasionally, the veins will contain so much feldspar that they become transitional to aplite dyklets. Sericite occurs along the walls, associated with the white feldspar and in fractures. Molybdenite, with or without pyrite/pyrrhotite, is usually intimately associated with the feldspar and/or sericite, particularily when the silicates are coarser grained.

Chlorite in molybdenite bearing quartz veins is almost always associated with fractures, and the molybdenite and chlorite occur intergrown with pyrrhotite/pyrite. Biotite in the quartz veins is so rarely seen that no definite relationships, particularly with respect to molybdenite, can be established.

In some of the larger veins, ribbony textures arise from the apparent rhythmic deposition of quartz and molybdenite, sometimes with white feldspar as well. These textures suggest regeneration of the vein system during sequential episodes of mineralization. In one location, at 36 m in No. 3 Drift, there is an example of "brain" texture, which consists of crenulated quartz veins interbanded with granodiorite. This is a refinement of, and further evidence for, a rhythmic pulsating period of molybdenite deposition. None of the above relationships are persistent throughout the deposit or indeed through any particular vein. Vein mineralogy can be seen to vary unsystematically from place to place and crosscutting relationships between veins of distinct mineralogy at one lccation can be reversed at another location. Thus, it is impossible to determine a sequence of mineralizing episodes of quartz veining based on vein mineralogy.

Of unusual but significant occurrence are zones of molybdenite mineralization associated with intrusive bodies know as High Grade Dykes. The molybdenite is associated with numerous micro fractures and large (greater than 10 cm thick) quartz veins in granodiorite dykes, is commonly intimately mixed with pyrrhotite as blades up to a few centimetres long, and appears as if it were a constituent mineral of the dyke. Grades can be as high as 5.0% MoS_2. Minor chalcopyrite has also frequently been noted with this high grade mineralization. The most significant of these High Grade Dykes was intersected in the crosscut at 1,492 m to 1,503 m striking across the crosscut for a drill indicated length of 60 m. Its vertical range may be as much as 200 m. A second much smaller (2 m thick) High Grade Dyke was encountered in the crosscut at 1,525 m and two other short intersections were encountered in drilling at depth to the SW.

Peripheral to the molybdenite deposit are skarn zones or small erratic fracture veins carrying tungsten, copper, lead, zinc, and silver. The skarn occurrences, such as the Copper Chief, consist of fine to medium grained scheelite as disseminations in a diopside pyrrhotite skarn or associated with quartz veins and garnet zones within the skarn. The skarns also carry chalcopyrite, molybdenite, sphalerite and galena.

The quartz fracture veins, such as the Lucky Boy, carry pyrite/pyrrhotite, black sphalerite, galena, argentiferous tetrahedrite, and euhedral scheelite. A few small quartz, galena, and red sphalerite veins occur at the margins of the molybdenite zones.

Alteration

The Trout Lake deposit, as is typical of stockwork molybdenite deposits, displays a complex but discernible alteration pattern associated with the mineralization. Because the mineralization and its alteration is strongly controlled by fracturing and veining, and may vary widely in type and intensity over short distances, it is difficult to see alteration patterns in drill core or hand specimens. The alteration patterns were defined, however, by applying quantitative X-ray diffraction techniques developed by Hausen (1979) and supported by visual observations made in the underground workings. Measurements by XRD of the alteration mineralogy were made on 15 to 30 m composites of surface diamond drill hole pulps. Despite some gross lithologic contrasts (e.g. between schists, carbonates and intrusives) a clear alteration pattern can be seen from the analyses.

The principal zoning established (Fig. G19 a-d) ranges from a strong silica-potassic zone with MoS_2 at the centre, outwards to a quartz-sericite-pyrite (phyllic) zone and possibly an outer zone where ankerite and chlorite are more prevalent, although chlorite concentrations rise again in the unaltered core of the intrusive mass. Both the central silica high, measured by total percent quartz, and the potassic en-

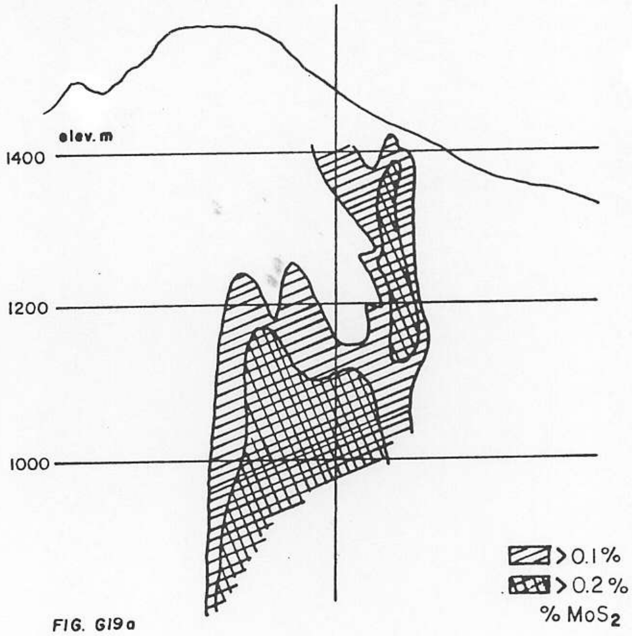


FIG. G19a

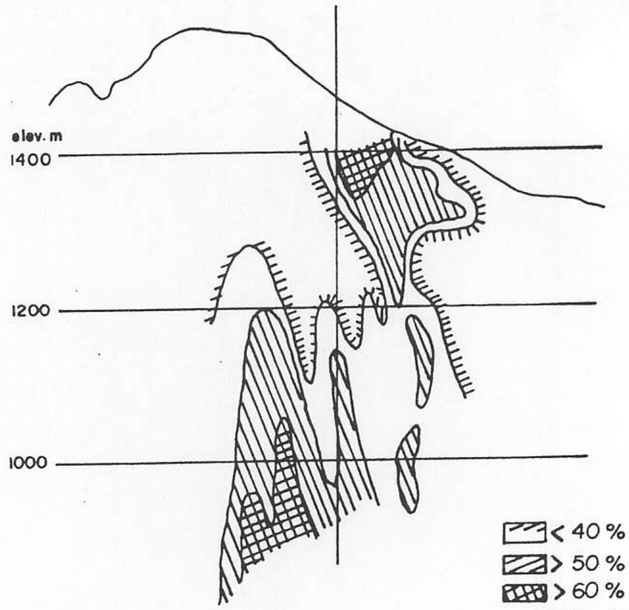


FIG. G19b

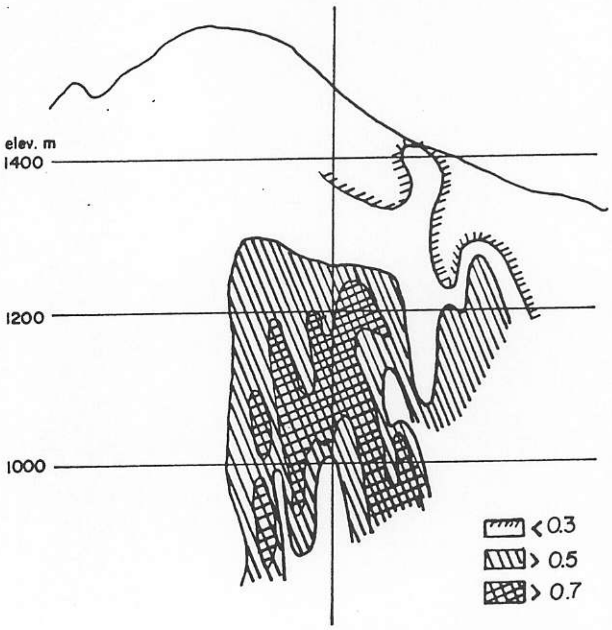


FIG. G19c

K-spar/Plagioclase Ratio

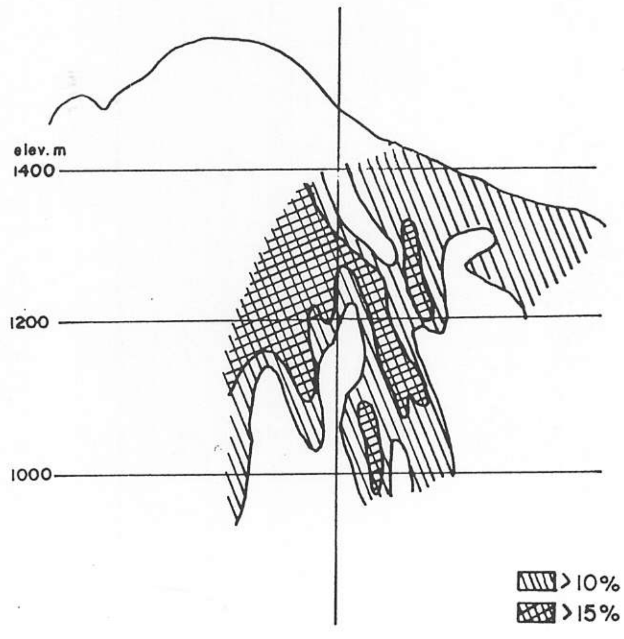


FIG. G19d

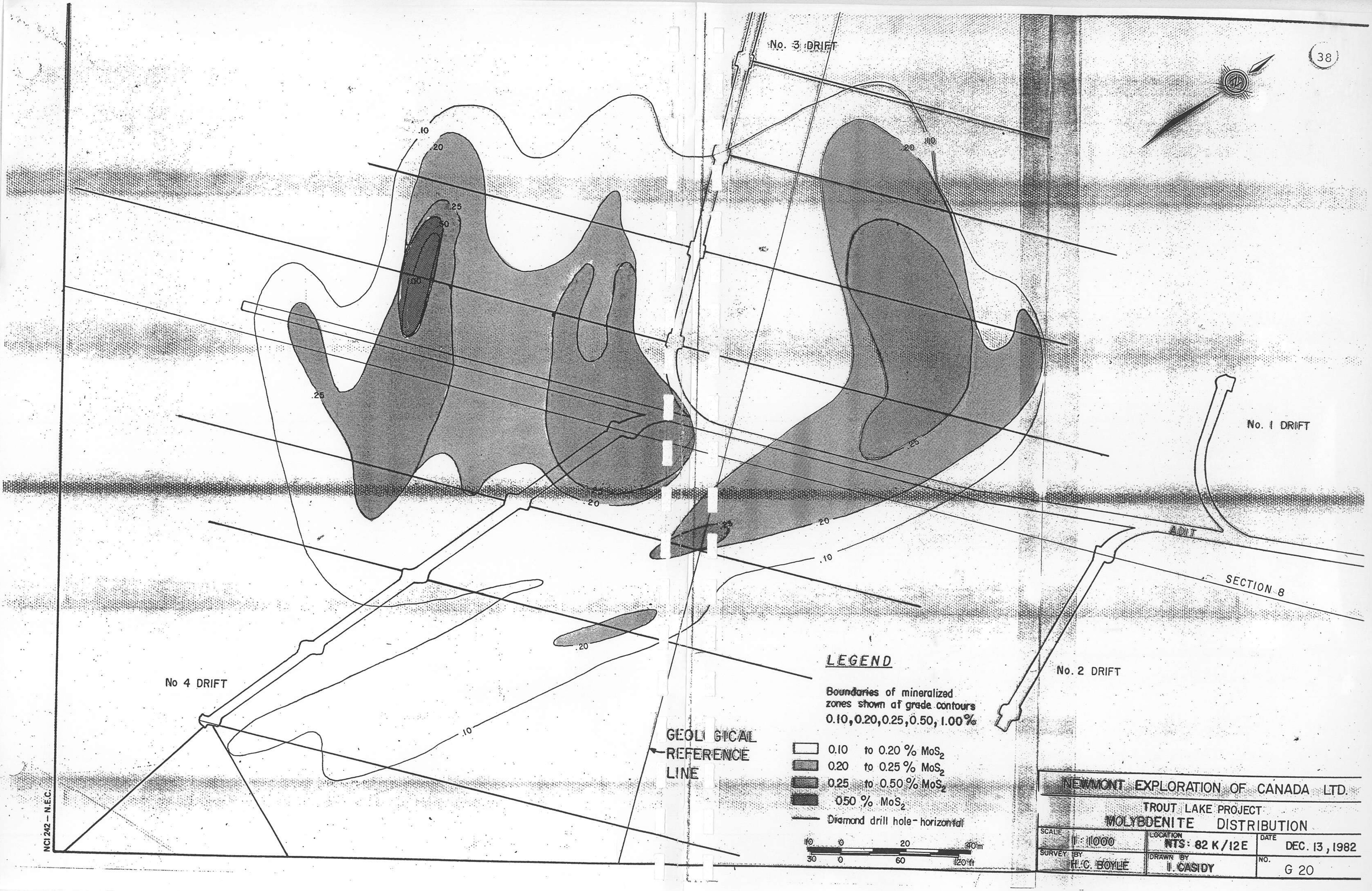
%Sericite

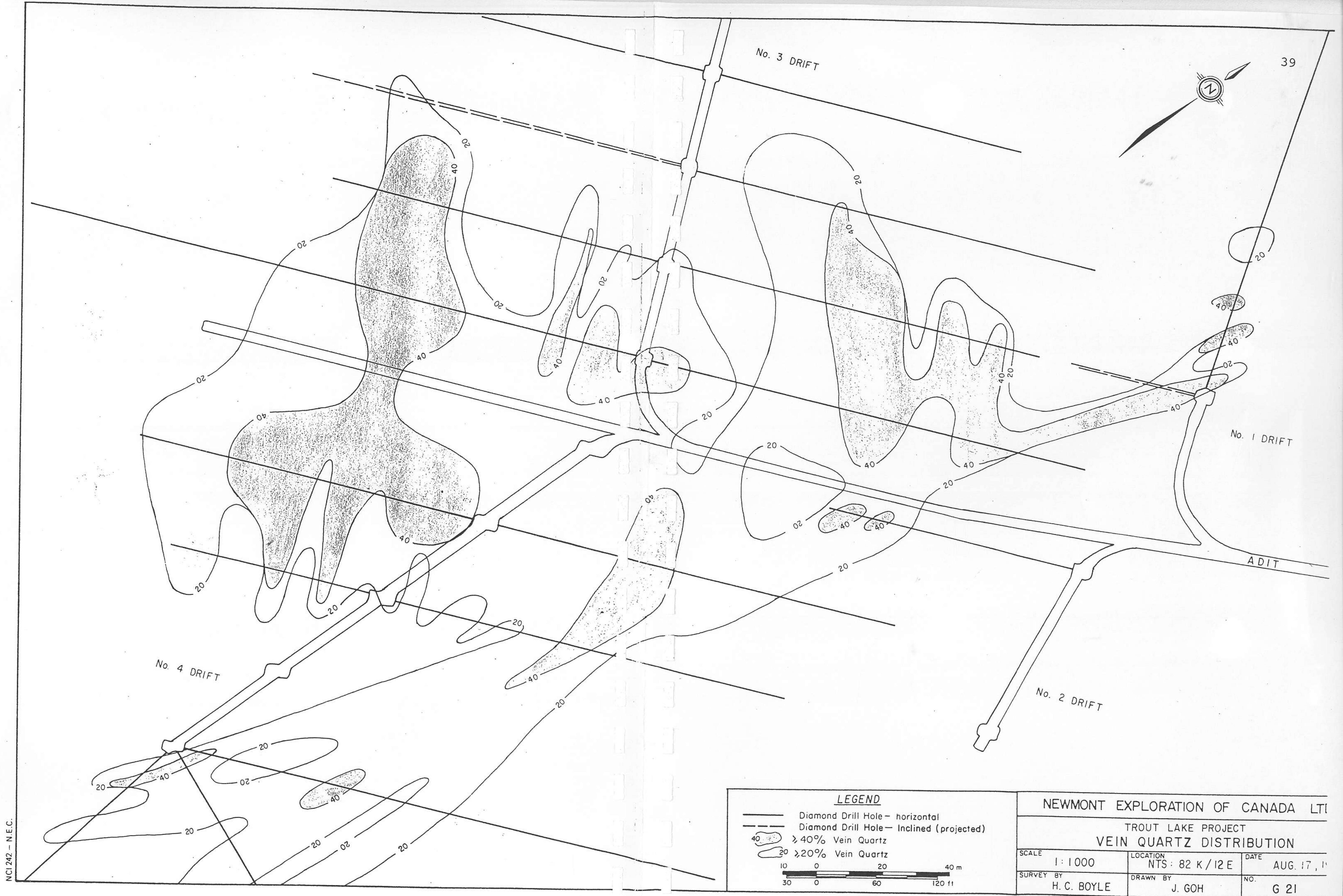
NEWMONT EXPLORATION OF CANADA LTD.		
GENERALIZED XRD ALTERATION PATTERNS		
SCALE 1: 10000	LOCATION TROUT LAKE	DATE JUNE 1982
SURVEY BY	DRAWN BY L CASIDY	NO. FIG G19

richment, measured as a ratio of K-feldspar to plagioclase, correlate well with the best molybdenite grades (Hausen, 1981). It should be noted that, as at many other molybdenum deposits associated with calcalkaline intrusives, the secondary feldspars at Trout Lake include not only K-feldspar but also alkali feldspars transitional to albite (Leitch, 1981), and these cannot be separated from "plagioclase" by the XRD method.

Visual observation, particularly in the underground workings, has tended to confirm this pattern. Ankerite and chlorite bearing veins and fracture coatings are encountered well away from the deposit, and distinct tan sericite envelopes (grading to prevasive alteration) are developed up to 300 m (1,000 ft) from the deposit. Within the deposit the altered schists contain dark, possibly secondary biotite; definite secondary biotite envelopes can be seen on a few quartz veins. An even more detailed correlation between mineralization and vein quartz can be seen in Fig. G20 and Fig. G21. This is a contour plot on the 960 m Level of the visually estimated percent vein quartz seen in the crosscut and drifts and the diamond drill holes, and an overlay of the contour plot of percent MoS_2 from diamond drill core assays. As can be seen, the correlation is almost direct. The figure also reflects the high quartz vein content of the east bounding Z Fault.

Petrologic examination of a selected suite of thin sections from surface and drill core specimens has also helped to elucidate the alteration patterns and their complexities. In detail many local fluctuations, reversals and retrograde minerals are observed. For instance, on a microscopic scale molybdenite flakes are often intergrown with sericite, quartz, and even calcite rather than K-feldspar (although this apparent relationship may only be due to later alteration of K-feldspar to sericite and quartz). In the central "potassic"


No. 3 DRIFT
No. 1 DRIFT
No. 2 DRIFT
No 4 DRIFT
ADIT
SECTION 8
GEOLOGICAL REFERENCE LINE
.10
.20
.25
.50
1.00
LEGEND
Boundaries of mineralized zones shown at grade contours 0.10, 0.20, 0.25, 0.50, 1.00%
0.10 to 0.20 % MoS2
0.20 to 0.25 % MoS2
0.25 to 0.50 % MoS2
0.50 % MoS2
Diamond drill hole - horizontal
10 0 20 40m
30 0 60 120 ft
NEWMONT EXPLORATION OF CANADA LTD.
TROUT LAKE PROJECT
MOLYBDENITE DISTRIBUTION
SCALE 1:1000
LOCATION NTS: 82 K/12E
DATE DEC. 13, 1982
SURVEY BY H.C. BOYLE
DRAWN BY I. CASIDY
NO. G 20
NC1242 - N.E.C.


No. 3 DRIFT
No. 1 DRIFT
No. 2 DRIFT
No. 4 DRIFT
ADIT
LEGEND
Diamond Drill Hole – horizontal
Diamond Drill Hole – Inclined (projected)
40 ≥40% Vein Quartz
20 ≥20% Vein Quartz
10 0 20 40 m
30 0 60 120 ft
NEWMONT EXPLORATION OF CANADA LT
TROUT LAKE PROJECT
VEIN QUARTZ DISTRIBUTION
SCALE 1 : 1000
LOCATION NTS : 82 K / 12 E
DATE AUG. 17, 1
SURVEY BY H. C. BOYLE
DRAWN BY J. GOH
NO. G 21
NC1242 – N.E.C.

zone, much of the alteration feldspar in the vein selvages is actually an alkali feldspar; true K-feldspar is often restricted to the vein itself. The pervasive alteration feldspar replacing plagioclase phenocrysts (away from veins) is albite. These same veins often contain true K-feldspar (not albite) well outside the potassic zone in the altered schists. The relationships of biotite, sericite and chlorite are very complex due to the presence of:

1. regional metamorphic sericite, chlorite and biotite,
2. thermal (hornfels) biotite developed around the stock, on which has been superimposed,
3. hydrothermal sericite and biotite, both related the stockwork vein system,
4. retrograde chlorite as the system cooled.

In a similar fashion to that described by Jambor and Beaulne (1978) and Sheppard (1977), it appears that the quartz-sericite-(pyrite) alteration at Trout Lake was later than the "potassic" alteration which accompanied the molybdenum mineralization. That is, the phyllic alteration envelopes on many quartz veins cut and replaced earlier feldspathic alteration, implying that as the hydrothermal system cooled and collapsed inwards on itself, cooler meteoric waters became an important part of the system.

Only traces of green fluorite have been seen at Trout Lake and topaz is notable by its absence. This is typical of a calc-alkaline molybdenum system (Westra and Keith, 1981).

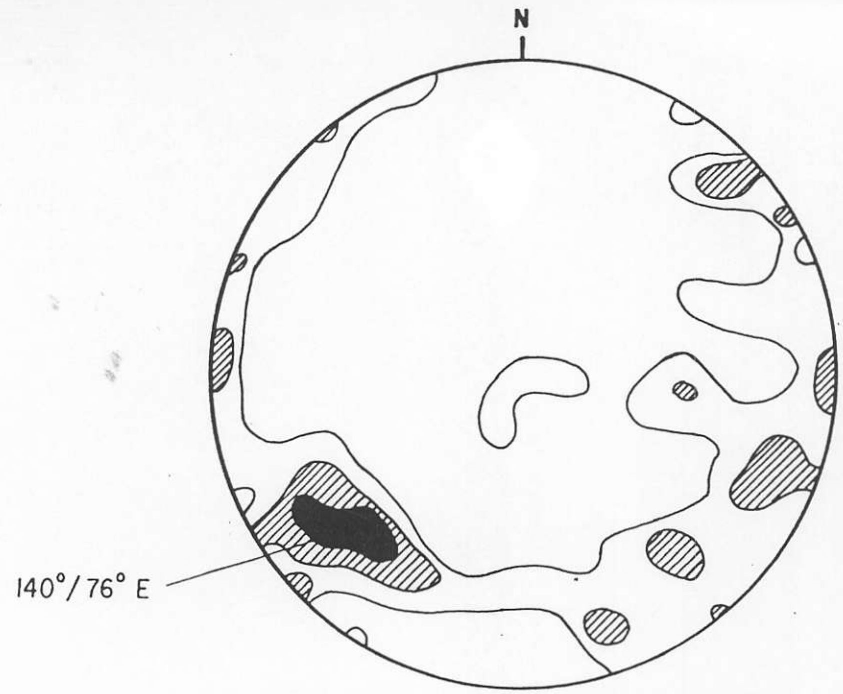
Controls of Mineralization

The controls of mineralization at Trout Lake are clearly related to veining associated with intrusive activity. This is apparent from the superposition of the contoured molybdenite grades and the percent vein quartz contours on the 960 m Level over the geology plan for this level (Fig. G20 and Fig. G21). The best MoS_2 grades coincide with high percentages of vein quartz and both are distributed around or within intrusive dykes or areas of intense and complex dyking. The contact area between the granodiorite stock and the enclosing metasediments, marked by complex dyking and veining, is thus the most important control on the concentration of mineralization. The best example of this importance of dyking and veining is the High Grade Dyke itself. This dyke, fractured and veined as it is, is the location of the most persistent strong molybdenite grades within the deposit. In addition, it is at least partially enveloped by a zone of increased quartz veining and mineralization. This increase in veining and grade is also noticeable around the dykes near the junction of the Crosscut and Nos. 3 and 4 Drifts. It is particularly clear and distinct in the area to the north in the upper levels of the B zone on Sections 4, 5 and 6 where mineralization drops off sharply once the main mass of the granodiorite stock is entered.

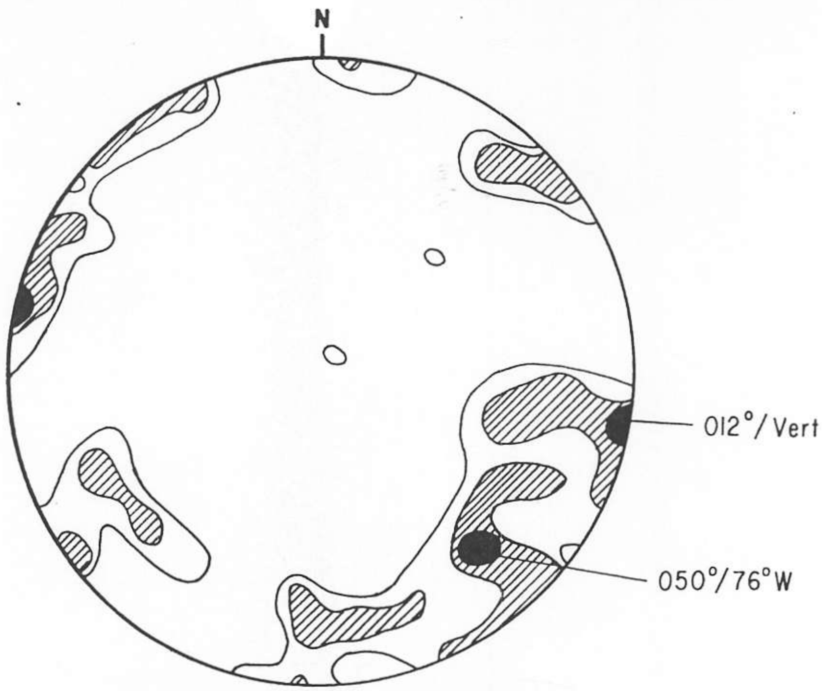
Whereas contact zones appear to be acting as an overall control of mineralization, most of the molybdenite is actually in quartz veins, and thus their density and orientation are of interest. A detailed survey of vein orientation was conducted by R. Linnen in the underground workings in the area of the deposit. The survey was conducted at nineteen stations of 10 m length (except for one station in each of No.'s 1 and 2 Drift, which were 20 m long because of a much lower vein density). There are nine stations through the Crosscut, one each in the No.'s 1 and 2 Drifts and four each in the No.'s 3 and 4 Drifts. The orientation of the veins was measured along with their thickness, and observations on their composition, mineralization (or lack thereof), crosscutting relationships or any other unusual features were recorded. In total 1962 measurements were taken.

The more important results are summarized in Fig. G22 and Fig. G23. Both these figures are of contoured lower hemisphere stereonet plots of poles to quartz veins. (The technique is described in the section on Structure, pg. 23). Fig. G22 demonstrates that the quartz veins can have virtually any strike (i.e. random strike). The vast majority have steep dips of 60° or greater and there are two prominent orientations that appear to be distinguishable by their intensity of mineralization. The upper plot indicates that moderately mineralized veins (defined as veins that would result in an overall grade of approximately 0.10 to 0.20% MoS_2) concentrate around an orientation of 140° strike and 76° E dip, which is sub-parallel to the foliation of the enclosing metasediments. In the lower plot of Fig. G22 are plotted the stronger mineralized quartz veins (those that would result in an overall grade of about 0.20 - 0.50% MoS_2). These veins concentrate around two orientations, one of 12° strike and vertical dip, approximately the same as the major Z Fault, and the other of 50° strike and 76° W dip, nearly perpendicular to the metasedimentary foliation and the quartz vein set described earlier. This earlier set (striking 140° with steep dips) can also be seen, though less strongly, in the lower plot of well mineralized quartz veins.

The following Fig. G23 presents the summary of the full survey's results. The survey fairly well covers the full area of mineralization, denoted by the 0.10% MoS_2 contour, as well as areas outside the mineralization. Consequently the vein



Quartz Veins With Moderate Mineralization
(230 measurements - 11.7 % of total measurements)



Quartz Veins With Strong Mineralization
(118 measurements - 6.0 % of total measurements)

LEGEND

Lower hemisphere stereonet plots
- Poles to quartz veins

- Low density
- Medium density
- High density

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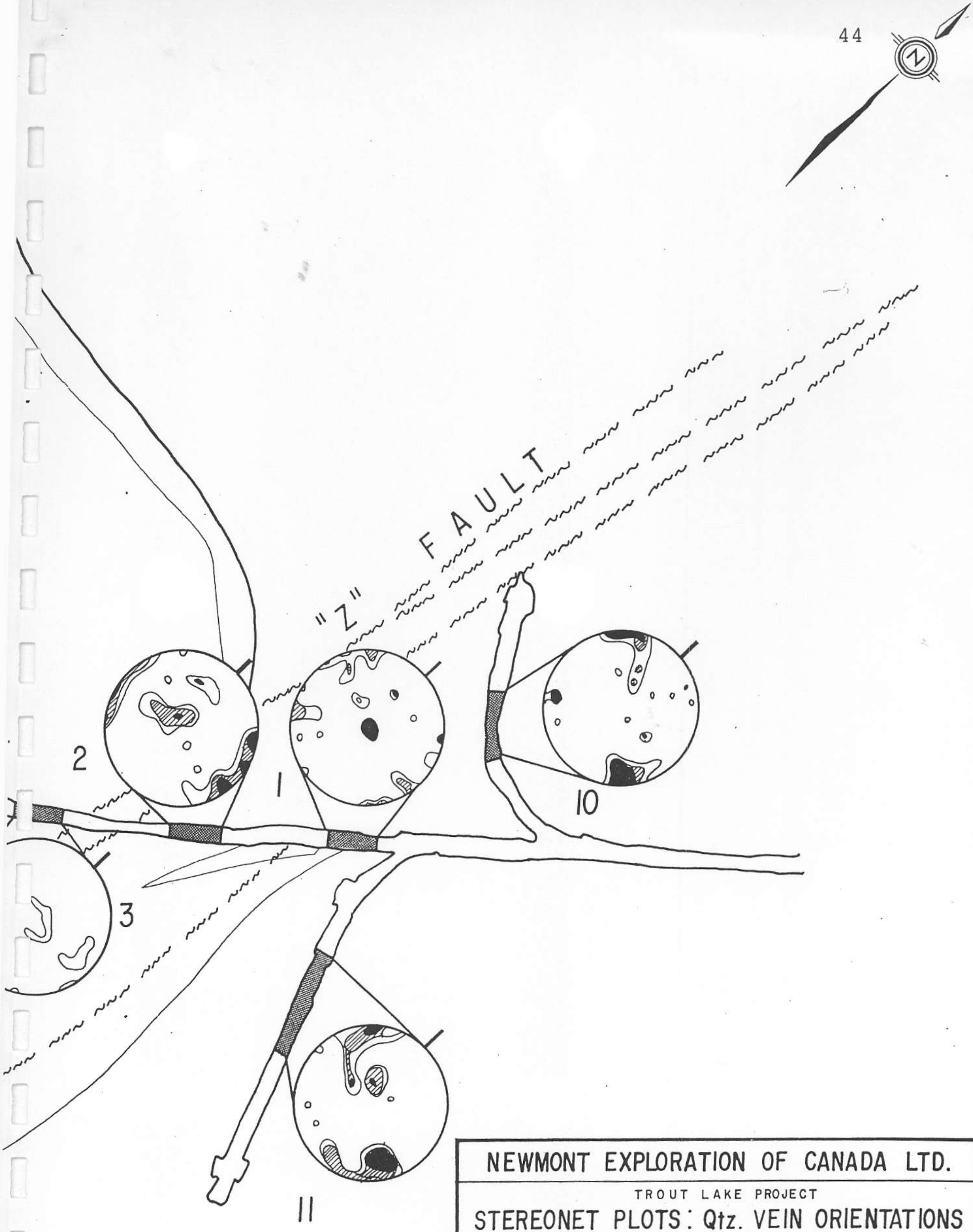
TROUT LAKE PROJECT

STEREONET PLOT - POLES TO MINERALIZED QUARTZ

SCALE	LOCATION	DATE Aug. 10/82
SURVEY BY	DRAWN BY	NO.



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TROUT LAKE PROJECT

STEREONET PLOTS: Qtz. VEIN ORIENTATIONS

0 10 20 30 40 50 60 70

SCALE IN METRES

SCALE: 1:1000	LOCATION:	DATE: AUG. 16, 1982
SURVEY:	DRN. BY: R WILKINSON	DWG. No.: G 23

density and resulting number of measurements varies greatly from station to station, from a low of 51 measurements in Stations No. 2 and 10 to a high of 195 measurements in Station No. 6. Significantly, Stations No. 1, 2, 10, 11, 14 and 15 all lie outside the 0.10% MoS_2 contour and contain fewer than 70 measurements each. The remaining stations lie within the 0.10% MoS_2 contour and contain greater than 75 measurements each; in fact, all but Stations No. 13 and 16 contain more than 100 measurements. Station No. 19 is the exception to this, lying outside the contour and containing 102 measurements. This may be because it lies close to the Z Fault on the west side.

Looking at the pattern of the contour plots through the deposit some interesting trends can be detected. Moving from the NE to the SW along the Crosscut, Stations 1, 10 and 11 all show strong development of the NE striking, steep dipping veins and an uncommon shallow-to-flat dipping set. Station 1 also shows a slight development of the NW striking veins. The distribution in Station 2 is slightly different, with a more even distribution between NW and NE striking and shallow dipping veins. This may be because the station is entirely with granodiorite. However, the vein density in these stations is low and few of them carry significant mineralization; consequently, they lie outside the mineralized zone, to the east of the Z Fault.

Crossing the Z Fault, the pattern immediately changes to one dominated by NW striking veins with relatively few at other orientations. Mineralization in this area is moderate. Continuing to the SW we see the development in Stations 5, 6, 7 and 9 of the wider variety of attitudes, suggesting a more complete development of the stockwork, but concentrations around the principal directions are still apparent. The plot for Station 8 reflects the strong influence of the High Grade Dyke in this area, where veins at an orientation sub-parallel to that of the dyke dominate.

Going NW along the No. 3 Drift, initially, the two dominant NE and NW striking sets clearly stand out in Station 12. The distribution becomes more widespread in Stations 13, 14 and 15 but not in an entirely random manner. To the NW, a pattern develops of the quartz veins distributed in a semi-continuous band, indicating quartz veins all with a NE strike but having any dip about this strike. The NW striking vein set is less prominent, but it is not absent through this transition. The low grades through the areas represented by Station 14 and 15 are related to low vein frequency as well as the vein orientations.

The trends along the southerly No. 4 Drift are similar to those seen in the No. 3 Drift but with respect to the opposite vein sets. Station 16 shows two good strong concentrations around the two principal attitudes of NW and NE striking veins with a third smaller concentration around an E-W strike indicating a well developed stockwork system.

Proceeding S along the drift, Stations 17, 18 and 19 show the development of a band showing NW strikes, again with completely variable dip. The NE striking vein set is present, but less strongly. There is even a suggestion of a distribution of dips similar to the above, but it is weak. Again, it is interesting to note that the area represented by Stations 18 and 19 (for the most part) are outside the mineralized zone as defined by the 0.10% MoS_2 contour. This is also an area dominated by only one of the two prominent vein sets and is consequently poorly mineralized. The fact that the dominating set is the NW one, associated more closely with only moderately mineralized veins, may account for the low grade, despite the significantly higher vein density (percent vein quartz) as compared with the end of No. 3 Drift.

The above evidence substantiates the argument that the best grades result where the two more prominent sets of NW and NE striking veins coincide, and occur with comparable frequency. Areas that are dominated by one set or the other are less likely to grade above 0.10% MoS_2. Also, the evidence suggests that the mineralization as a whole is associated with steeply dipping structures. When the coincidence of intense quartz veining surrounding intensive dykes is recalled and combined with the vein orientation evidence, the two most influential controls on mineralization are seen to combine to produce the high grade zones. In short, the best grade mineralization occurs where complex intrusive dykes invade the enclosing metasediments, producing a zone of intense stockwork veining. This stockwork is dominated by a more extensive, moderately mineralized set of NW striking veins intersecting strong, but more dispersed sets of N and NE striking, well mineralized quartz veins.

Trace Elements

The zonation of trace elements has been studied to a limited degree by X-ray fluorescence analyses (Hausen, 1981). Analyses for Sn and W were done on 15 m composite drill core samples over selected holes, and indicated that although Sn values are very low (10 - 20 ppm), close to the detection limit of 5 ppm, they may be centrally zoned with the Mo. Tungsten (with values up to 300 ppm) is zoned outside the Mo and is associated with skarned carbonate horizons. Analyses for As suggest that it is concentrated in a zone outside the tungsten with values up to 500 ppm compared to 50 ppm in the central zones. Traces of chalcopyrite occur throughout the system, with geochemical analyses running up to 200 ppm Cu; but copper was not analysed for routinely, so no Cu zonation can be determined.

Bismuth minerals and sulfosalts have not been identified, and Bi, Sb, Ag and Au do not show discernible patterns (at levels close to their detection limits in each case). No patterns were detected from the few composites analysed for Mn, F, and U. Mn and F values both ranged from 300 - 400 ppm and U up to 2 ppm. Rhenium values were less than the detection limits at 2.5 ppm in drill core composites. (These results of trace element analyses are from twelve composite samples from surface diamond drill holes, as presented in the "Report on 1977 Work Program" by H. C. Boyle and T. N. Macauley, 1978.)

Semi-quantitative XRF analyses for ZrO_2 (0.005 - 0.017%), SrO_2 (0.001 - 0.05%), and Rb_2O (0.001 - 0.04%) showed very low values and no detectable patterns.

The limited study of trace elements to date indicates that the mineralization at Trout Lake is simple, consisting essentially of molybdenite with associated iron sulphides. Contaminants such as Pb, Zn, Bi, As and Sb either occur in very low concentrations or are restricted to zones of peripheral mineralization which do not overlap with the concentrations of molybdenite mineralization.

RESERVE ESTIMATE

General

The exploration program to date has outlined a geologic reserve of 48.7 million tonnes with an average grade of 0.193% MoS_2 in four distinct zones defined by the 0.10% MoS_2 contour. Contained within this reserve is 11.7 million tonnes of mineralization averaging 0.362% MoS_2 at a 0.20% MoS_2 cutoff. The four zones involved as defined by the combined results of the surface and underground drilling, are referred to as the A, B, D and F Zones. Of these, the B Zone is the largest and, as the principal focus of the exploration program, the best defined. The A Zone is the smallest, closest to surface, and reasonably well defined by drilling. The D Zone is second in size to the B Zone with its broad characteristics outlined by drilling but its details still uncertain. The F Zone has only been indicated by two exploratory drill holes and its dimensions and grade are only approximated.

Reserve Estimate

The reserve estimate is tabulated below in Tables R-I and R-II. The estimate is based on drilling done on twelve vertical sections through the deposit on an azimuth of 050°/230°. No significant reserves were encountered on Sections 1 and 2 and consequently they do not enter into the calculation. Only minor mineralization was located on Section 3, and that used in the reserve estimate is presented on the Geology Section 3 in Fig. G7. The clear overlays (Fig. R1 to R9) represent the

TABLE R-I

RESERVE ESTIMATE BY SECTION

Section	Cutoff	Drill Defined Tonnes	Drill Defined Grade ($\%MoS_2$)	Drill Defined Tonnes x Grade	Drill Indicated Tonnes	Drill Indicated Grade ($\%MoS_2$)	Drill Indicated Tonnes x Grade
3	0.10	87,312	0.139	12,116			
	0.20						
	0.25						
4	0.10	353,655	0.245	86,494	681,850	0.130	88,625
	0.20	124,277	0.435	54,085			
	0.25						
5	0.10	6,547,750	0.163	1,065,702	833,708	0.287	239,325
	0.20	1,018,859	0.273	278,595	392,986	0.452	177,789
	0.25				141,821	0.400	56,728
6	0.10	4,832,110	0.187	902,680	3,891,750	0.166	646,495
	0.20	1,641,386	0.331	543,381	139,128	0.423	58,826
	0.25	355,858	0.504	179,456	139,128	0.423	58,826
7	0.10	8,514,237	0.238	2,025,070	1,531,633	0.177	270,527
	0.20	3,333,121	0.385	1,282,820	95,309	0.497	47,368
	0.25	1,161,334	0.557	646,659	95,309	0.497	47,368
8	0.10	6,976,727	0.266	1,856,631	2,786,396	0.146	407,600
	0.20	3,457,316	0.393	1,357,690	186,293	0.279	52,063
	0.25	2,196,922	0.490	1,075,696			
9	0.10	4,016,115	0.178	714,411	1,911,563	0.126	240,042
	0.20	999,765	0.279	278,820			
	0.25	539,622	0.319	171,938			
10	0.10	2,091,981	0.154	323,178	444,312	0.149	66,241
	0.20	293,761	0.344	101,087	22,848	0.372	8,500
	0.25	151,858	0.434	65,832	22,848	0.372	8,500
11	0.10				1,301,112	0.134	173,728
	0.20						
	0.25						
12	0.10	342,965	0.159	54,367	1,593,077	0.151	241,296
	0.20						
	0.25						
Total	0.10	33,762,852	0.209	7,040,649	14,975,401	0.159	2,373,879
	0.20	10,868,485	0.359	3,896,478	836,564	0.412	344,546
	0.25	4,405,594	0.486	2,139,581	399,106	0.430	171,422
Grand Total	0.10	48,738,253	0.193	9,414,528			
	0.20	11,705,049	0.362	4,241,024			
	0.25	4,804,700	0.481	2,311,003			

TABLE R-II

RESERVE ESTIMATE BY ZONE

		Drill Defined			Drill Indicated		
Zone ~~Section~~	Cutoff	Tonnes	Grade ($\%MoS_2$)	Tonnes x Grade	Tonnes	Grade ($\%MoS_2$)	Tonnes x Grade
A	0.10	2,656,979	0.240	637,929	2,534,171	0.161	408,226
	0.20	1,056,639	0.389	410,653	299,309	0.501	150,044
	0.25				48,144	0.602	28,983
now C A (Displaced?)	0.10	1,267,249	0.153	193,904	2,692,229	0.145	389,577
	0.20	201,308	0.305	61,404			
	0.25	59,405	0.440	26,149			
B	0.10	26,526,966	0.211	5,602,113	1,791,855	0.181	324,054
	0.20	9,142,725	0.354	3,235,719	186,293	0.414	77,211
	0.25	4,253,736	0.488	2,073,749	186,293	0.414	77,211
D	0.10	3,311,658	0.183	606,703	5,519,835	0.161	886,843
	0.20	467,813	0.403	188,702	350,962	0.334	117,291
	0.25	92,453	0.429	39,683	164,669	0.396	65,228
F	0.10				2,437,311	0.150	365,179
	0.20						
	0.25						
Total	0.10	33,762,852	0.209	7,040,649	14,975,401	0.159	2,373,879
	0.20	10,868,485	0.359	3,896,478	836,564	0.412	344,546
	0.25	4,405,594	0.486	2,139,581	399,106	0.430	171,422
Grand Total	0.10	48,738,253	0.193	9,414,528			
	0.20	11,705,049	0.362	4,241,024			
	0.25	4,804,700	0.481	2,311,003			

data used to calculate the bulk of the reserve estimate. A reserve overlay for the 960 Level Plan at a scale of 1:500 is presented in Fig. R10. These overlays, when used in conjunction with their corresponding geologic sections and plan, also illustrate the relationship between mineralization and geology.

The reserve estimate was calculated manually using standard polygonal methods modified by the geological interpretation. After the geological interpretation was completed, assay contours were developed based on assay grades encountered, their continuity from drill hole to drill hole, and conformity to the geologic interpretation. The geologic factors which exercised the most control on mineralization, as summarized in the section on "MINERALIZATION", are:

1. contact between the granodiorite stock and the metasediments,
2. areas of dyking,
3. strong vertical preference for mineralized structures,
4. limiting relationship of the Z Fault.

Once the contours were drawn, at 0.10, 0.20, 0.25, 0.50 and 1.00% MoS_2, the outlined zones were divided into polygons, for the most part based on individual hole intercepts. These polygons, and the drill hole intervals and associated composite grades upon which they are based, are presented on the reserve sections. Most polygons are based on single drill hole intercepts falling between two assay contours but some exceptions are apparent. In areas around drill stations, such as polygon No. 32 on Section 7, where the drill hole density is high, a weighted average grade was calculated for a zone that was defined by a grade contour and/or a line drawn from drill hole to drill hole where their separation increased to about 30 m. Other exceptions are cases where a polygon is defined not by a dill hole within the polygon but by a drill hole(s) on an adjacent section(s) on either or both sides; such a hole being closer than any drill hole on the section of the polygon itself. Extensions of geologic trends are also used to define the polygons. In these instances, the grade of the so defined polygon is determined by the weighted average of the drill hole(s) on the adjacent section(s). An example of this is polygon No. 73 on Section 8.

Once all polygons had been defined and appropriate grades assigned to each, volumes for each of the polygons were generated by planimetering the polygons and projecting the area 15 m either side of the section. The resulting volume, calculated in cubic meters, was multiplied by the specific gravity of the rock of 2.72. This was the average of specific gravity determinations on three representative samples of Trout Lake core. The tonnage, in metric tonnes, with its grade of MoS_2 was thus determined for each polygon defined on the ten sections from Section 3 to Section 12.

A standard 15 m projection either side of the section plane was used instead of trying to adjust the length of projection between sections for the details of the contoured zones. This latter alternative would have created an unwieldy mathematical problem of accounting between sections where a particular zone was not continuous. As well, level plans that would be used to make these adjustments were developed from the vertical sections, (except for the 960 m Adit Level Plan) and the information was less detailed than the information on the sections. It was felt that the effort required to accomodate these modifications could not be justified by the limited increase in precision of the reserve estimate that might result. In addition, there would be a significant opportunity for introducing calculation errors. To check the accuracy of the tonnage estimate based on simple 15 m projections, volumes were generated for the various zones based on level plans at 50 m intervals. Table R-III presents the results of these calculations and compares them to the results obtained using the sections. Volumes only were used in this check as it was not possible to reasonably assign grades to zones outlined in level plan, since the drilling was done on vertical sections.

The comparison between those estimates is what would be expected. The larger and better defined by drilling is the zone, the better the agreement. For instance the largest and most thoroughly drilled zone, the B Zone at the 0.10% MoS_2 contour, agrees within 3%, while the F Zone, poorly understood with only two drill holes, has a discrepancy of 52% between the two figures. Likewise, the small zones of plus 0.25% MoS_2 tend to show wider variations pointing out the uncertainty with which they are defined.

The grade and tonnage having been determined for each polygon, they were then organized as to the grade and zone which contained them and as to whether they could be considered as defined or indicated. A polygon was classified as defined if it had a drill hole through it and was supported by similar intercepts in surrounding holes or by well established geologic trends. Indicated polygons are those drawn from single drill holes with no firmly established geologic controls for guides, or with no drill holes within them and based solely on the extension of trends derived from the geological interpretation. In a few instances, where the trends were strong and compelling, it was felt to be justified to include in the defined category a polygon without a drill hole intercept.

TABLE R-III

COMPARISON OF RESERVES AS CALCULATED FROM SECTIONS AND PLANS

Zone	Cutoff	Tonnes From Sections	Tonnes From Plans	Percent Difference From Section
	0.10	5,191,150	5,560,034	+ 7%
A	0.20	1,355,948	1,163,344	-14%
	0.25	48,144	41,480	-14%
A (Displaced)	0.10	3,959,478	3,613,124	- 9%
	0.20	201,308	97,172	-52%
	0.25	59,405	-	-
	0.10	28,318,821	29,048,067	+ 3%
B	0.20	9,329,018	8,944,680	- 4%
	0.25	4,440,029	4,144,591	- 7%
	0.10	8,831,493	7,978,420	-10%
D	0.20	818,775	722,963	-12%
	0.25	257,122	380,077	+48%
	0.10	2,437,311	1,178,440	-52%
F	0.20			
	0.25			
	0.10	48,738,253	47,378,085	- 3%
Total	0.20	11,705,049	10,928,159	- 7%
	0.25	4,804,700	4,566,148	- 5%

Comparison With Previous Estimates

When comparing the reserve estimate as defined by the combined results of the surface and underground programs to the reserve estimate as defined by the surface program alone, it is remarkable to note their close agreement in overall tonnage and grade. The surface exploration had suggested a reserve figure of 48.4 million tonnes at 0.225% MoS_2 at 0.10% MoS_2 cutoff, which included 10.7 million tonnes at 0.348% MoS_2 at 0.20% MOS_2 cutoff. This compares to 48.7 million tonnes at 0.193% MoS_2 and 11.7 million tonnes at 0.362% MoS_2 for corresponding cutoffs in the combined programs estimate. These figures agree to within 10% except for the grade figures for the 0.10% MoS_2 cutoff, which differ by 17%.

The close agreement in the overall reserve figures somewhat masks the improvement in the understanding of the Trout Lake deposit. The underground program has demonstrated that the B Zone diminishes much more rapidly to the NW than had been appreciated from the surface drilling. It also increased the size and significance of the D Zone and outlined an apparently offset portion of the A Zone on the east side of the Z Fault. The underground drilling also discovered the new F Zone deep to the SW. The details of the more economically significant high grade zones (plus 0.20% MoS_2) are more clearly understood as a result of the program. The most significant improvement, however, is the far higher reliability of the material in the defined category of the combined estimate compared to the surface program estimate of indicated reserves. Furthermore, several new areas with potential for hosting additional reserves were discovered.

FUTURE EXPLORATION TARGETS

General

Although an appreciable amount of exploratory drilling was done simultaneously with the definition drilling, there remain several areas that are recommended for further exploration. They were not adequately tested in the present program because they were at the limit of equipment on site, would have required additional drill drifts and stations, and would have constituted a major addition to the scope and budget of the program. The full significance of one area was not appreciated until the required information had been thoroughly analyzed after the conclusion of the program.

Three areas are recognized at this point as having potential for adding significant new reserves to the Trout Lake deposit. It is worth noting that in each case, the area for additional potential is well within the property claim boundaries and any future exploration would be entirely contained within the present claim group. They all are at considerable depth, approximately the 600 m elevation and below, and occur to the NW, the SW and E of the known deposit.

The Northwest Area

Further exploration is suggested by DDH's 81-8 and 81-61, seen on Section 4. Due to a recalculation of the relative locations of these two holes, it is now clear that there is a trend towards better grades going from DDH 81-61 to 81-8 which has not been cut off at depth. In addition, the mineralization continues from the metasediments into the intrusive, which shows multiple phases. This suggests that there may be a mineralizing source at depth in this area that has not yet been penetrated by any drill hole.

The Southwest Area

The potential for this area is suggested by a number of points. The first and most obvious is the intersection of the F Zone on Sections 8 and 6 by DDH's 81-15 and 81-63. The zone requires more extensive definition which may lead to an expansion of the zone with possible improvement in grades. The potential is reinforced by the general trend of the mineralization at Trout Lake, plunging steeply to the SW from the A Zone at surface to the F Zone itself at depth. Also the occurrence of additional, though small, High Grade Dykes deeper and to the SW of the High Grade Dyke in the B Zone suggest a continuation of this trend. And finally, fracturing, veining and alteration in DDH's 81-63 and 81-64, particularily in the skarny altered rocks, is finer, more closely spaced, and more intense than that usually seen in the B Zone. This may be the result of a mineralizing source separate from and possibly even stronger than that responsible for the B Zone.

The East Area

The potential of this area had been appreciated even before the start of the underground program. It is situated on the east side of the Z Fault on Sections 7 to 12 and 500 metres or more below the adit level.

The DDH's 81-29 and 81-39, on Sections 8 and 10, both penetrated granodiorite lying east of the Z Fault (itself a favourable indication), and encountered well altered and silicified schist with short sections of significant molybdenite mineralization. These intersections are sometimes similar in appearance to intersections obtained above the High Grade Dykes, particularly along the contact of the intrusive with the schists. Combining this with earlier surface drilling, there is a clear increase downwards in area of alteration, presumably associated with a hydrothermal system, and of the extent and grade of mineralization.

DDH 81-84 on Section 8 never got out of intrusive though it was drilled through the Z Fault and down to an elevation of about 400 m. It did, however, encounter alteration of the granodiorite at the bottom of the hole consisting of intense sericitization and pyritization with weak finely disseminated molybdenite flakes. It also contained quartz veins carrying molybdenite which gave individual assays up to 0.7% MoS_2, and zones of ankerite veining. The texture, mineralogy and style of mineralization of this deep intersection in 81-84 resembles most closely core taken from surface hole 76-3. This suggests that there may be a down drop on the east side of the Z Fault in the order of 1,000 m, and that a mineralized zone similar to the B Zone may exist below DDH 81-29, possibly by some 200 m.

UNDERGROUND EXPLORATION PROGRAM

General

The underground exploration program at Trout Lake was undertaken after five years of surface work, including four seasons drilling, had indicated a significant deposit of molybdenite mineralization. Because the surface drilling indicated a strongly vertically oriented zone at considerable depths, an underground program was proposed to intersect the most promising zone at its core and conduct extensive drilling to define the mineralization sufficiently for preliminary mine planning. At the same time, further exploratory drilling was to be done from underground to clarify zones with only sketchy drill information and to test new areas too deep to be drilled from surface. In addition to drilling, the underground program was designed to bulk sample the mineralization through the heart of the deposit.

The underground program consisted of the 1,550.5 m (5,087 ft) adit-crosscut and four drifts totalling 448.4 m (1,471 ft). Regularly spaced drill stations were cut in the drifts and 22,119.4 m (72,570 ft) of drilling completed. Most of the drilling was done on 30 m sections in rings of up to ten holes, supplemented by crosscutting inclined holes. The objectives of the drilling were: to define the most promising zone of mineralization for the purposes of preliminary mine planning, to further explore zones inferred from surface drilling, and to explore new areas too deep to be reached from surface. Pilot holes were also drilled in advance of the mining through the mineralized zone for comparison of their grades with the bulk sample results.

Mining

In July 1979, the decision was made to go ahead with an underground program. A site was selected for the collaring of the portal on the basis of overall length of the adit, room for a future mill plant and permanent campsite, and provision for tailings disposal. The contract for the undergound excavation was awarded to Canadian Mine Services Ltd. Cal-Van Canus Camp Services Ltd. supplied a mobile trailer camp and kitchen facilities for 40 men.

a) Performance - Work on the adit portal began on September 26, 1979 and completed with the excavation of the final drill station on April 26, 1981. Mining conditions, reflected in the contractor's performance, contrasted strongly between the ground excavation in the approach to the deposit and within the deposit, separated by the Z Fault at 1,311.6 m (4,303 ft) in the adit. The adit was driven at a grade of +0.35% throughout the workings to allow for gravity drainage. However, because of high initial water flows the natural drainage was assisted with pumps at 490 m and 880 m (1,600 ft and 2,900 ft). In the first part of the program, the adit was driven at 3.7 m x 4.6 m (12 ft x 15 ft) to 1,276.1 m (4,157 ft) to allow its use as a future haulage way. Beyond that point, the adit and four drifts were driven at 3.0 m x 3.7 m (10 ft x 12 ft). The overall performance from collaring the portal to reaching the Z Fault was 1,311.6 m (4,303 ft) in 393 days (allowing 14 days for a break at Christmas) for an average daily advance of 3.34 m (10.9 ft) per day. If the collaring of the adit is excluded, the rate of advance increases to 3.55 m (11.6 ft) per day.

The reason for the slow advance in the approach to the deposit was the intersection of numerous water bearing fracture zones. These zones resulted in water flows of up to 27,000 litres per minute (6,000 GPM) causing delays due to poor working conditions, drilling of relief holes, caved blast holes and stuck drill steel. As well as the delays in the loading and blasting operation the broken zones necessitated installation of ground support consisting of steel sets and heavy timber cribbing at 13 separate locations, including the portal area, and a total of 91 steel sets. Many of these were relatively minor, requiring only 2 to 4 sets, but several involved major water inflow areas and support works. These occurred at 687 m, 941 m, 1,021 m, 1,055 m, 1,072 m and 1,087 m; with 687 m and 941 m being the two most serious. The fractured ground at 941 m halted progress for 17 days while a raise was driven and bridging installed across the zone before the adit could be advanced.

Rock bolting and steel strapping were used only sparingly in the adit because, except for the major fracture zones or some minor broken sections, the rock was competent and stood up well without support.

b) Rate of Advance - There was a marked improvement in ground conditions and progress, once the Z Fault was encountered and passed. The rock was very competent due to siliceous alteration and fracture zones being narrow and tight. As well, the ground was relatively dry. The advance from 4,303 m to the end of the adit (referred to as the "crosscut" on the W side of the Z Fault) and in the four drifts totalled 689.8 m (2,263 ft), and all drill stations amounted to a slash equivalent of 79.2 m (260 ft) for a total of 772.1 m (2,533 ft). This was completed by April 26, 1981 in 159 days (again allowing a 14 day Christmas break at the end of 1980) for an average rate of advance of 4.84 m (15.9 ft) per day.

Besides the Z Fault, there was only one other zone of fractured rock in all the crosscut and drifts which required major ground support. The Z Fault itself required 5 square sets consisting of 10 in x 10 in x 10 ft posts and 8 in I-beam caps. Square sets were used to provide more room for ventilation ducting and other services. The second fracture zone at 1,356 m (4,448 ft) was also dry, and required only 3 steel arch sets.

Again, rock bolting and steel strapping was used only sparingly. Wire mesh was used principally in the drill stations, which were excavated to 5.5 to 8.2 m (18 to 27 ft) backs, to prevent rock falls while drilling was being done.

Combining the advance for the adit approach, the crosscut and the drifts, the overall advance on the program averaged 3.77 m (12.4 ft) per day over the entire 2,083.6 m (6,836 ft) of underground workings.

c) Costs - The contrast between the rate of advance in the adit to the Z Fault at 1,311.6 m (4,303 ft) and the rate of advance in the crosscut and drifts within (or near) the deposit is also reflected in the mining costs. These costs are summarized in Table U-I. As can be seen, the direct underground excavation costs are 73% higher in the approach to the deposit than within the deposit, and the surface support costs are 16% higher. Combining the two gives an overall cost comparison of 60% higher in the approach to the deposit.

Underground excavation costs include all ground support (steel sets and timbering, rock bolts, rock spiling and steel strapping). Within the adit approach this includes the exceptional case of the raise and bridging at 941 m (3,086 ft). Within the deposit crosscut and drifts, this includes all the steel mesh screening and rock bolting installed in the drill site excavations.

TABLE U-I

TROUT LAKE PROJECT
UNDERGROUND MINING COSTS

	ADIT (to Z Fault at 1,331.6m)		CROSSCUT AND DRIFTS- Including All Drill Sites (total 772.1m)		TOTAL (2,083.6m)	
	$ Can/m	$ Can/ft.	$ Can/m	$ Can/ft.	$ Can/m	$ Can/ft.
Underground Excavations	3,010.74	917.67	1,741.15	530.70	2,540.42	774.32
SUPPORT COSTS:						
Portal Facilities	197.38	60.16	197.38	60.16	197.38	60.16
Engineering	30.20	9.20	23.73	7.23	27.81	8.48
Camp	364.94	111.23	287.61	87.66	336.31	102.51
Surface & Gen.	9.61	2.93	9.61	2.93	9.61	2.93
SUB TOTAL	602.13	183.52	518.33	157.98	571.11	174.07
T O T A L	3,612.87	1,101.19	2,259.48	688.68	3,111.53	948.39

Support costs are broken down into four categories. Portal facilities do not include the construction costs of the bulk sampling dump bins or the core logging building. Engineering is principally surveying costs. Camp costs are the apportioned costs of the exploration camp and its operation. "Portal facilities" and "Surface and General" are distributed on the basis of the lengths of the respective underground excavation. "Engineering" and "Camp" are distributed on the basis of time over which the respective underground excavations took place.

Diamond Drilling

The diamond drilling contract was awarded to Cameron McCutcheon Drilling Ltd., a subsidiary of Canadian Mine Services Ltd. The contractor used three hydraulic drive, electric powered, "Super Drills", with power transmitted to the drills via cable from two 250 kw diesel generators at surface, operated in parallel. Water for drilling was initially supplied by the mining contractor and then from water made in some of the first underground drill holes.

a) Drill Performance - The drilling program began on November 22, 1980 with the collaring of a pilot hole on the line of the crosscut and was completed with the final hole on November 5, 1981. (Four short diamond drill holes (80-1 to 80-4) were drilled to investigate ground conditions during the driving of the adit, and do not form part of the actual underground program.) In all 22,119.4 m (72,570 ft) of drilling was completed in the underground program in 87 holes by three drills, phased in over the first six months of the program. A month-by-month production record for each drill, identified by the numbers 34, 35, and 36, is presented in Table U-II. Drill performance started at 18.63 m (61.1 ft) per machine day and improved through the course of the program to about 37.50 m (123.0 ft) by the end of the program. (The last five days of the program at the beginning of November 1981 were combined with the October 1981 figures to arrive at this final performance figure). The improvement reflects improved drilling conditions over those encountered early in the program when drill holes were collared in the drifts to the east of the Z Fault, less severe water problems as the crosscut and drifts drained the surrounding rock, and the drill contractor overcoming start-up problems with the equipment and becoming more familiar with ground conditions. The overall drill performance at the end of the contract was 31.20 m (102.36 ft) per machine day. The 72,674 ft (22,151.0 m) appearing in Table U-II, resulting in this performance figure, differs from the earlier stated total drilling because it includes footage drilled, but for a variety of reasons, not charged. This would include such things as having holes re-collared because of misalignment or repeated due to driller's mistake, and in one instance because drill support cribbing collapsed between shifts.

b) Costs - The costs of the underground drilling program are summarized in Table U-III. Drilling costs are direct charges from the contractor, including contract rates, diamond bit costs and any additive used during the program. Additional costs include an apportionment of the mobilization and demobilization costs of the project, camp costs and meals, power generation and fuels, surveying, and on-site management, including core logging and sampling. It does not include any drifting or drill site preparation, which are accounted for under mining costs.

Date
Nov. 1980
Dec. 1980
Jan. 1981
Feb. 1981
Mar. 1981
Apr. 1981
May 1981
June 1981
July 1981
Aug. 1981
Sept. 1981
Oct. 1981
Nov. 1981

TABLE U-III

DIAMOND DRILL HOLE COSTS

	Cost/m	Cost/ft	Total Cost
Drilling	$CN 143.49	$CN 43.73	$CN 3,172,745
Additional	39.91	12.17	882,432
Total	183.40	55.90	4,055,177

c) Drill Hole Distribution - A summary of the drill distribution is presented in Table U-IV. Table U-V gives an account of all the drilling done on the Trout Lake deposit to date. The underground drilling was carried out from 11 stations in the crosscut and four drifts, on 8 sections corresponding with geologic Sections No. 4, 5, 6, 7, 8, 9, 10 and 12. Though stations were excavated on Sections 3 and 11, no drilling was necessary on those sections.

The initial drilling in the program was the four pilot holes drilled in advance of the mining of the crosscut and Drifts No. 3 and 4. Two pilot holes were required for the crosscut because the first drill hole wandered and plunged below the crosscut grade. The purpose of the pilot holes was to provide direct comparisons between assay grades obtained from drill core and those obtained from the bulk samples. This is discussed in a separate chapter on the bulk sampling. The total length of pilot hole drilling is 503.5 m (1,652 ft) or 2.3% of the total length drilled.

The bulk of the drill program was the definition drilling of the B Zone as well as the less concentrated effort on the D Zone. This was done with a pattern of ring drilling from the stations in Drifts No. 3 and 4, supplemented by crosscutting inclined holes from Drifts No. 1 and 2. Most sections used a ring of 10 drill holes in the plane of the section, one in either direction of the horizontal, and holes in both directions

TABLE U-IV

DIAMOND DRILL HOLE DISTRIBUTION

Section	Drift No.	No. of Holes	Length (m) NQ	BQ	Total	Percent Total Drilling
4	3	6	574.3	482.2	1,056.5	4.8%
5	3	5	338.4	379.2	717.6	3.2%
6	1	5	656.3	1,501.4	2,157.7	9.8%
	3	10	102.7	1,608.5	1,711.2	7.7%
7	3	11	1,829.1	1,621.2	3,450.3	15.6%
8	Crosscut	2	447.9	894.1	1,342.0	6.1%
	2	4	811.1	832.1	1,643.2	7.4%
	4	8	637.0	1,139.7	1,776.7	8.0%
9	4	10	821.8	849.0	1,670.8	7.6%
10	2	5	1,216.2	1,175.3	2,391.5	10.8%
	4	10	366.7	1,511.1	1,877.8	8.5%
12	4	7	867.4	953.2	1,820.6	8.2%
Pilot		4	503.5	0.0	503.5	2.3%
TOTAL		87	9,172.4	12,947.0	22,119.4	100.0%

TABLE U-V

SUMMARY OF DIAMOND DRILLING

Year	Operator	Holes	No.	Actual Length (m)	Effective Length (m)
1970	Cascade Moly Mines	TL-1 to TL-6	6	992.4	938.8
1976	Newmont-Imperial	76-1 to 76-7	7	2,772.2	2,583.5
1977	Newmont-Imperial	77-1 to 77-3	3	1,712.4	1,573.1
1978	Newmont-Esso	78-1 to 78-5B	7	4,279.4	4,072.4
1979	Newmont-Esso	79-1 to 79-15	15	6,982.7	6,883.3
Total surface			38	16,739.0	16,051.1
1980-81	Newmont-Esso	80-5, 80-6 81-1 to 81-85	87	22,151.0	22,119.4
TOTAL			125	38,890.0	38,170.4

at ±35° and ±70°. Modifications to this scheme occurred where drill holes already existed and at either end of the deposit (Sections 4, 5 and 12) where previous drilling had indicated a weakening of the mineralizing system.

Besides the ring drilling, long inclined holes were drilled from Drifts No. 1 and 2 to crosscut the strongly vertically oriented B Zone and confirm the results of the ring drilling. These were drilled from Sections 6, 8 and 10 and consisted of three fanned holes on each section directed to the SW at dips of approximately +25 to 30°, -25 to 30°, and -50 to 60°. In addition, two holes were angled off either end of Drifts No. 1 and 2 to test adjacent sections at depth and one short horizontal hole was drilled to the SW on Section 8 to check mineralization near the adit. In total, 15,356.7 m (50,383 ft) of the underground program could be oonsidered definition drilling on the B and D Zones, or 69.4% of the total 22,119.4 m drilled.

Finally, several holes were drilled into untested areas to try to extend known mineralization and discover new zones. This includes not only holes designed specifically for this purpose, but also long extensions of the definition drill holes into new areas. Holes drilled exclusively for exploratory purposes include one drilled to the NW from Drift No. 1 and three drilled to the south and east from the end of Drift No. 4. All other exploratory drilling was accomplished by extending drill holes, which also aided in defining the B and D Zones. These were mostly holes drilled to the SW from horizontal to -82° from all four drifts. The longest drill hole was 81-8, extended for exploration purposes from Drift No. 1 on Section 6 to a final depth of 706.2 m (2,317 ft). Two drill holes drilled from the end of the crosscut to the NE under the workings helped define the NE boundary of the B Zone, and their extensions also explored new ground at depth on the east side of the Z Fault. In all, 15 of the 87 holes drilled in the underground program were extended past 400 m (1,300 ft) for the purposes of exploration. Combined with the holes drilled soley for exploratory purposes, total drill length devoted to exploration was 6,259.2 m (20,535 ft) or 28.3% of the drill program.

d) Core Size - The diamond drilling done underground at Trout Lake was a mix of NQ and BQ size core. Of the total 22,119.4 m, 9,172.4 m (30,093 ft) were drilled NQ, or 41.5%; and 12,947.0 m (42,477 ft) were drilled BQ or 58.5%. Holes were drilled NQ when it was known that they would be going through broken ground and a reduction in size would probably be necessary, when the objectives of the holes were at least in part exploratory, and when the definition holes were going to be sampled for assay by splitting. BQ drilling was used in the definition drilling when holes were going to be sampled whole for assaying.

e) Ground Conditions - Ground conditions presented few problems throughout the program. Initially, inclined holes drilled up from Drifts No. 1 and 2 encountered broken ground and high water pressures when they were drilled through the Z Fault. This lead to a slowed advance and some concern for the safety of the drillers, but techniques were soon employed to overcome the problem. A positive consequence of this experience was that holes from these two locations supplied the drill water requirements for the remainder of the program. Water under high pressure was encountered again near the end of the program on Section 12 at the end of Drift No. 4. Again, it was the water bearing Z Fault responsible for this pressure and the problem was overcome after some initial difficulties.

f) Core Recovery - Because of the generally good ground conditions, core recovery was good throughout the program. Recovery averaged 92.8% over the entire program with only three holes falling below an 85 - 95% recovery rate. Poorest recovery was in DDH 81-23 at 54.5% over 121.9 m, while the best recovery was in DDH 81-56 at 98.4% over 112.2 m. The longest hole, DDH 81-8, had a recovery of 94.5% over 706.2 m.

Drill Hole Surveying

Experience from surface drilling indicated that drill hole direction was difficult to control over long distances. It was decided that the underground drill holes should be surveyed to determine their location over their entire length, and the method selected was Atlas Copco's Fotobor technique. The Fotobor was used because magnetic pyrrhotite in the deposit made methods based on magnetic direction determinations suspect. Gyroscopic instruments were unsuitable because of the adverse environment underground, the delicacy of the equipment, the difficulty of surveying inclined BQ holes and the experience necessary in operating the equipment.

The Fotobor proved suitable to the requirements of the Trout Lake Project. The equipment was well designed and quite rugged. Operation of the equipment is simple and straightforward and the survey staff had no problem setting up an efficient routine which could be run by one surveyor supervising the drill crew. The only difficulty of significance was the necessity to run the survey in feet because of the drilling equipment, and calculate the survey in metres because of the design of the computer program necessary to convert the collected data to survey coordinates.

Fotobor surveys of the first few holes indicated that holes deflected only a few metres in the first 250 m (800 ft). It was decided that for drill holes shorter than this, no directional survey was necessary; it was assumed that for this length the drill holes maintained their original direction and that acid dip tests were all that were necessary to determine the vertical deflection of the hole along its length. In total, 28 drill holes were surveyed by Fotobor totalling 12,700 m (41,666 ft) or 57% of the length drilled.

A complete report of the Fotobor method of diamond drill hole surveying has been compiled by the project surveyor R. Wells, entitled "The Reflex-Fotobor Surveying of Diamond Drill Holes on the Trout Lake Molybdenite Project, Trout Lake, B.C."

Core Logging

Core logging at Trout Lake was carried out by a staff of three geologists assisted by two labourers-core splitters. Logging was carried out on forms modified to allow the semi-quantitative recording of data for later computer entry, along with descriptive sections on the geology and mineralization. Aspects entered semi-quantitatively included fault or fracture zones, the alteration minerals chlorite, sericite, biotite and silica, and the percent total sulphides. These supplemented a code for rock type, survey and sampling interval data, and assay results, all of which can be computerized. In addition to logging, all core was photographed on slides to provide a permanent visual record of the underground drilling. (See example at end of this report.)

Core Sampling

Sampling of the drill core was almost 100% over the entire length of the underground program. Only selected lengths in ground that was unquestionably barren by visual examination were not sampled for assay. These sections, totalling 462 m or 2.1% of the drilling, occurred in long stretches of massive granodiorite and silicified schist which had very low fracture vein intensities.

Sampling procedure was based on a standard 2 m (6.6 ft) sample interval in material visually estimated to be subeconomic (0.03% MoS_2) or better, and 3 m (9.8 ft) in very weak mineralized rock below this grade. (This compares with the standard 10 ft (3.05 m) sample used in all the surface drilling.) As had been the practice in all previous drilling under the joint venture, the standard sample interval was modified where abrupt changes in grade were apparent or to conform to a geologic boundary between distinctly different rock types or a major sturcture. The only major departure from this practice was in the sampling of the pilot holes, when the standard interval was shortened to 1.5 m (4.9 ft) to provide more detailed information. Altogether 9,938 samples were taken for assay from the underground drilling. From the surface holes, 4,951 samples were taken, for a total of 14,889 samples.

As mentioned earlier, some of the core was sampled whole and some was split for assay. This was done because the high density of the definition ring drilling made retention all split core redundant, and only alternate holes in the ring were split. Consequently, in order to maintain a reasonably constant volume of rock for each sample, holes to be split were drilled NQ and holes to be sampled whole were drilled BQ. This gave corresponding sample volumes of 890 cm^3 per metre of split NQ core and 1,046 cm^3 per metre of whole BQ core, a 15% difference in volume of sample material. In some instances, where it was desirable to retain a split core sample because the drill hole was exploring new areas, the core was split even though it was drilled BQ for technical reasons. In the total 21,657.4 m (71,054 ft) of core sampled, 16,461.9 m (54,009 ft) were split for assay and 5,195.5 m (17,045 ft) were sampled whole, or 74.4% and 23.5% respectively.

It was decided that for the underground drilling program, the drill core would be split using a rock saw. This would allow better control during the sampling procedure and provide a superior record for the drill core library. A Target "Handy Matic" 14 in. Circular Masonry Saw with a No. HM1433, 3 horse-power, 230 volt, 60 Hertz, 3 phase electric motor was used along with 14 in. diameter slotted diamond saw blades. A standard Longyear splitter was used occasionally when the supply of saw blades ran out before the next delivery arrived.

Core was split on a two shift per day, 5 day week basis while drilling was on a 24 hr a day, 7 day a week basis. Productivity varied from about 45 m (148 ft) per shift to 115 m (377 ft) per shift, depending on the skill of the splitter and the hardness of the rock. The overall average productivity was 62 m (203 ft) per shift. Productivity from each saw blade averaged about 185 m per blade over the length of the project. Combining the capital cost of the saw equipment and blades with the labour costs of splitting the core (the core splitter time is estimated to be divided into 60% splitting and 40% core handling, sample shipping and other duties) the total costs of splitting the core by saw are estimated at $2.22/m or $0.68/ft.

Computerization

At the start of the underground drilling program, computerization of the drill data had become a major objective. Survey and assay data from previous years had already been entered into a data bank at Newmont Exploration Limited's computer facility in Danbury, Connecticut. The underground drilling would more than double this data, and in addition geologic and alteration features were to be entered into computer data banks for later analysis. This required a slight modification of the drill log sheets and a concerted effort by all involved in core logging to standardize, in a numerical form, their recording of the various features observed during the logging procedure. Besides survey and assay data, the features recorded for later entry into a data bank included core recovery, fracture zones, rock type, alteration mineralogy (as represented by the relative abundance of chlorite, sericite, biotite and silica), fracture intensity, percent vein quartz (by volume), and percent total sulphides. To make the record complete, the logging from the previous years' surface drilling was to be converted to a oorresponding form and also entered into the data bank.

At this time, all the survey and assay data from the underground and surface drill holes have been entered into the Newmont computer facility, ready for analysis.

The situation regarding geologic and alteration information is not anywhere near as complete. Geology and alteration from DDH 81-1 to DDH 81-12 of the underground program have been entered into the computer, but have not been checked for accuracy. Geologic information from the rest of the underground holes has yet to be entered. As for the surface drilling, only the geological data from the 1979 program has been transcribed, entered and checked before final entry into the computer's data bank.

Virtually all other years' drilling has yet to even be transcribed to the format compatible with computerization. This has not been done because it is a slow and painstaking process requiring technically qualified personnel. Because the Trout Lake Project has been suspended, these labourious tasks have been deferred. The completion of this data transcription and entry into the computer would be essential before an effective, computer assisted interpretation of the geology of the Trout Lake Deposit could be undertaken. The purpose of such a study would be to improve our understanding of the deposit and indicate areas where additional reserves might be found.

BULK SAMPLING PROGRAM

General

A bulk sampling program was carried out on the Trout Lake Project to provide a comparison between the bulk sample assays and diamond drill hole sample assays. Bulk samples were obtained from the crosscut and Drift Nos. 3 and 4 within the deposit and their Mo content compared to that in ~~, compared to~~ pilot holes drilled in advance of these drifts. The bulk samples were also used for metallurgical testing at Newmont's research laboratory in Danbury, Connecticut (see chapter on Metallurgy).

Sampling Procedures

a) Bulk Samples - Advance in the adit reached the mineralized zone in November 1980 and continued through the crosscut and Drifts No. 3 and 4 to April 1981. During this time, individual rounds of approximately 100 tonnes were segregated in concrete bins at the portal dump. Each round was mixed and stored in a prepared stockpile area. Lot numbers assigned each round identified the round's location on the underground plan. In total, 189 bulk samples were segregated and stored for processing.

Processing of the bulk samples began on June 22, 1981 and continued on a 10 hour per day basis for 21 days to July 12. An eleven man operating crew consisted of three supervisory staff, two crusher operators, three workers in the sampling tower, one labourer to handle the shipping and storage drums, and two equipment operators. Nominal processing time per sample, including clean-up, was half an hour. In total, 227 samples were processed through the crushing plant and sampling tower; one sample from each of the 189 stored rounds, a second check sample from 17 rounds, and third and fourth check samples from 7 rounds.

Operating efficiency of the sampling plant was severely reduced by 50% down time on the sampling tower. Most of this (37%) was due to electrical problems, principally too small electric motors burning out under the load of sample material.

In the sampling procedure, a sample was cut from each pile using a Cat 966 front-end loader. The cut was taken a few inches above the base of the pile, crowding to the centre, removing one quarter of the round or approximately 25 tonnes. The cut was then reduced to minus 15 cm (6 in.) material in a Pioneer 28 x 36 crusher. This was further reduced to minus 1.6 cm (5/8 in.) in a Pioneer 45V gravel crusher and conveyed to a sampling tower.

The sampling tower was built by Minpro of Toronto, Ontario, for the Trout Lake Project from a design provided by Canadian Mine Services Ltd. at a cost of $65,000 delivered on site. An in-line sampler first cut about 1,360 kg (3,000 lb) of material, which was reduced to minus 6 mm (¼ in.) in a roll crusher. From this a Vezin type sampler cut 10%, which was then split in a second Vezin type sampler. One 68 kg (150 lb) split was collected in a 25 gallon drum and the second split was split again to two 34 kg (75 lb) samples in a 1.26 cm (½ in.) riffle. These final two 34 kg samples were weighed and bagged. One bag was placed in the 25 gallon drum which was sealed and marked for identification and stored on site. The second 34 kg bagged sample was marked for identification and placed in a 45 gallon drum for shipment. The final splits were 50% larger than planned due to the initial in-line cutter characteristics. Reject material was removed by a Wagner ST 2B scoop tram and/or the front-end loader. The entire bulk sampling procedure is summarized in the design flow sheet presented in Fig. B2.

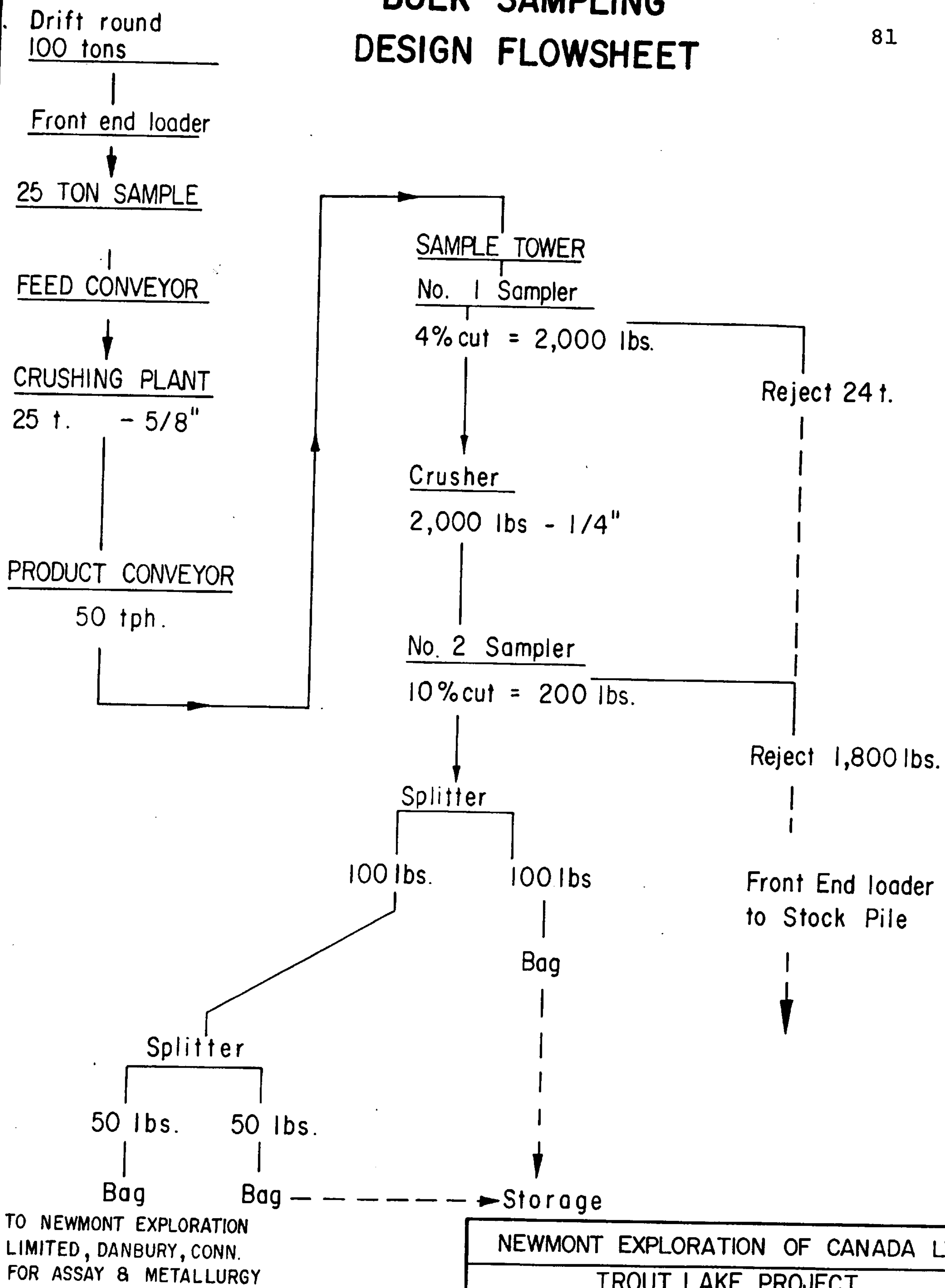
b) Diamond Drill Holes - Pilot diamond drill holes were drilled in advance of development through the length of the crosscut and mineralized portions of Drifts No. 3 and 4. A stabilizing core barrel assembly was used to maintain the drill holes on a straight line. Despite this, the holes tended to wander out of the area of the headings in 60 to 90 m (200 to 300 ft). In the drifts, this did not present a problem because both pilot holes in these cases were less than 80 m. However, the first crosscut pilot hole penetrated the sill of the crosscut after 89 m (292 ft), and a second hole had to be drilled to ensure that a comparison between the pilot hole and bulk sample was based on data from the same region.

The diamond drill core was split for assay using sample lengths of 1.5 to 2.0 m (4.9 to 6.6 ft) to provide detailed information on the mineralization. Sample lengths were also modified to accommodate distinct changes in the estimated grade of mineralization and/or rock type. The split core was identified with a sample number, bagged and sealed in a 12.5 gallon drum for shipment. The other half of the split core was retained for on-site storage.

The diamond drill core and crushed bulk samples were shipped to Newmont Exploration Ltd.'s research laboratory in Danbury, Conneticut, for assaying and metallurgical testing (see relevant chapters on these subjects).

Assaying results can be seen in Fig. B1. In this figure, it should be understood the "weighted average pilot hole grade

BULK SAMPLING DESIGN FLOWSHEET



NEWMONT EXPLORATION OF CANADA LTD.		
TROUT LAKE PROJECT BULK SAMPLING PROGRAM		
SCALE	LOCATION	DATE DEC. 1982
SURVEY BY	DRAWN BY	NO.

of drift round", which allows direct comparison between the grades of the bulk sample and pilot hole, is a calculated grade derived from the drill hole assays weighted by the length of each sample which falls within the bulk sample.

Analysis of Assay Results

a) Check Bulk Samples - As can be seen from the bulk sample plan (Fig. B1), there are significant differences in the assay values of some individual samples from various rounds. For instance, Round 2 has an initial sample assay value of 0.106% MoS_2 and a check sample assay of 0.185% MoS_2, a 75% difference. On the other hand, Round 55 has initial and check assays of 0.076 and 0.077, agreeing to within 2%. A detailed comparison done by F. T. Hancock (see Appendix C - Trout Lake Project, Bulk Sampling Program) indicates that, when considered statistically, the check sampling program supported the results of the bulk sampling program. The study pointed out that the average grade of the 24 second samples was 0.210% MoS_2 as compared to an average grade of 0.203% MoS_2 for the initial samples, or only 3% difference. It also appears from this check sampling program that the ordinary initial sample may slightly underestimate the actual grade of the round. This is suggested by the higher average grade for the second sample and by the higher averages again for the third and fourth samples for the 7 rounds in which these were taken.

b) Bulk Sample/Diamond Drill Hole Comparison - Of the 189 rounds excavated, 128 have a pilot diamond drill hole through them for comparison. Compared round by round, bulk sample grades vary from 85 to 137% of diamond drill hole grades. This highly variable comparison on an individual basis is resolved when the data is analyzed statistically and averages compared. In making this comparison it is noteworthy that the bulk sample to diamond drill hole sample size ratio is about 12,000 to 1. It is also noteworthy that a frequency plot for both bulk sample grades and diamond drill hole grades indicates positively skewed lognormal distributions. Because of this, a comparison based on a lognormal analysis of the data is presented as well as one based on an arithmetic analysis. The results are summarized in Table B-I taken from F. T. Hancock's report. Table B-I presents the analytical results for both the arithmetic and the lognormal analysis on assay data weighted for round length, i.e., on the assay grade times round length.

Table B-I shows an average grade for the bulk samples of 0.222% MoS_2 versus 0.205% MoS_2 for the diamond drill hole, or 8% higher, in the arithmetic analysis. In the lognormal analysis, the difference is slightly more than 1% (0.217 versus 0.214% MoS_2). If only the 77 pairs of samples from the crosscut are considered, representing the largest continuous sample with the greatest variation in individual grades, the bulk sample average grade is 12% higher in the arithmetic analysis, with average grades of 0.280% MoS_2 for the bulk samples and 0.251% MoS_2 for the diamond drill hole samples. The lognormal analysis also gives a higher average grade for these 77 bulk samples, 0.276% MoS_2 versus 0.254% MoS_2, or 9%.

The variance, a measure of the spread in the grades for individual samples, shows a somewhat anomalous pattern associated with the end of the crosscut beyond Drifts Nos. 3 and 4. In this area, the crosscut intersected two High Grade Dykes, but DDH 81-5 intersected only one. This resulted in more very

TABLE B-I

BULK & PILOT DDH GRADE COMPARISON

			Arithmetic Analysis				Lognormal Analysis			
	No. of Samples	Round Length Average (ft)	Bulk Grade Average ($\%MoS_2$)	DDH Grade Average ($\%MoS_2$)	Bulk Product Variance	DDH Product Variance	Bulk Grade Average ($\%MoS_2$)	DDH Grade Average ($\%MoS_2$)	Bulk Product Variance	DDH Product Variance
X-Cut 1 DDH #80-5	38	9.921	0.230	0.262	0.664	2.729	0.231	2.269	0.714	4.229
X-Cut 2 DDH #81-5	39	9.513	0.330	0.241	9.199	2.901	0.329	0.241	9.092	2.688
X-Cut 1+2	77	9.714	0.280	0.251	5.108	2.803	0.276	0.254	3.427	3.347
3 Dr. DDH #81-4	25	9.600	0.118	0.093	0.243	0.351	0.118	0.103	0.245	1.021
4 Dr. DDH #81-6	26	9.423	0.149	0.174	0.368	0.986	0.149	0.176	0.358	1.171
TOTAL	128	9.633	0.222	0.205	3.689	2.324	0.217	0.214	2.167	3.740

high grade bulk sample assays than diamond drill hole assays, and consequently, a higher variance. In fact, the variance for this section, identified in Table B-I as "Crosscut 2, DDH 81-5", is so much higher for the bulk sample results (9.199 vs 2.901), that it overpowers the variance figures of all the other areas in the arithmetic analysis. For the total number of sample pairs, this results in the reverse of what would normally be expected, the variance of the bulk samples (3.689) is greater than the variance of the diamond drill hole samples (2.324). However, in the lognormal analysis, because the analytical results appear to more closely reflect the actual distribution of mineralization, the variation between individual values is subdued. Consequently, though the variance relationship for the "Crosscut 2, DDH 81-5" area shows the same trend in the lognormal analysis as in the arithmetic analysis, its influence is reduced. The results of the lognormal analysis for the total number of sample pairs shows the more predictable lower variance for bulk samples compared to diamond drill hole samples (2.167 vs 3.740).

Conclusions

The conclusions from the sampling results and statistical analysis are that individual or small groups of diamond drill hole sample grades do not provide a reliable estimate of bulk sample grades and by extension, the surrounding volume of rock. They do, however, when taken in large enough groups (e.g. approximately 100 or more) give a reasonably close estimate of the bulk sample grades. The diamond drill hole grades may, in fact, give a conservatively low estimate of the surrounding rock grade. A comparison of a straight arithmetic analysis to the lognormal analysis gives a closer estimate of grades for the latter, being about 1% lower as opposed to 8% lower.

With respect to grade estimations for reserve estimates, this means that the larger, better understood mineralized zones (such as those outlined by the 0.10 and 0.20% MoS_2 contours) defined by several diamond drill holes involving hundreds of core samples, are likely to be well (and possibly conservatively) estimated from diamond drilling. Conversely, smaller zones or those defined by single drill holes with only a limited number of core samples probably cannot be estimated accurately. This is consistent with general experience and the impressions gained from six years exploration work on the Trout Lake Project.

ASSAYING

Laboratories Used

Throughout the exploration program on Trout Lake the analytical services of Chemex Labs Ltd. of North Vancouver, B.C. were used for the assaying of the core samples. The selection of Chemex Labs Ltd. was based on the extensive history of satisfactory service to Newmont, a sound reputation in the industry and a familiarity with the problems associated with sampling and assaying for molybdenite. Chemex was used through the entire program to provide continuity of techniques and quality of assaying from hole to hole, year to year.

Chemex's usual procedure was to crush the core samples to ¼ in. and then split them in a Jones riffler down to about 250 gm. However, following the 1978 program the procedure was modified to a finer crushing (to 1/8 in.) for which Chemex was paid extra. The finer crush was designed to produce a more homogeneous sample before splitting, thereby reducing sampling errors. For the 1981 underground drilling, a three stage crushing procedure was employed. The core was run through a primary jaw crusher to -1 in. and then through a secondary jaw crusher to -¼ in. It was finally reduced to -1/8 in. in a tertiary cone crusher.

After Chemex takes a 250 gm split of -1/8 in. material out of the sample, it is pulverized by a "puck and ring" device to -100 mesh and homogenized by rolling 100 times. From this material, 2 gm are digested in a hot perchloric nitric acid mixture for 2 hours, diluted to a specific volume with demineralized water and allowed to settle. The solution is buffered with aluminum chloride and the determination made on a "Techtron" AAS (atomic absorption spectrophotometer) against prepared standards.

For the pilot holes and bulk samples from the underground workings, assaying was done at the Newmont Exploration Limited research laboratory in Danbury, Connecticut. This was because all the metallurgical work was to be done in this laboratory and consistency and continuity in procedures were desired. Newmont's laboratory procedure differs from Chemex's in three areas. Newmont crushes to 10 mesh (1.65 mm) before splitting and pulverizing to -100 mesh (0.15 mm) and takes 1 gm instead of 2 gm of sample for digestion after homogenization. The potentially most important difference is in the digestion procedure. Newmont digests the sample in a mixture of nitric, perchloric, hydrochloric and hydrofluoric acids and evaporates the mixture to a moist residue, which is then dissolved in dilute hydrochloric acid with aluminum added to suppress interferences. This is a more complete digestion process and liberates any molybdenum that may possibly be entrapped in the silica matrix. The sample is then analyzed in an atomic absorption spectrophotometer against prepared standards, as is done at Chemex. These two methods constitute the standard procedures used for assaying samples taken from Trout Lake.

Check Assaying Program

Through the course of the surface and underground programs, a checking procedure was implemented to determine the reproducibility of assays and to identify any systematic errors resulting from sample preparation or analytical techniques. About 1% of the 4,491 surface drill hole samples and 9,938 underground dill hole samples were selected to cover a grade range of 0.05 to 4%

MoS_2. The checking was carried out largely at Chemex's own laboratory and Newmont's laboratory in Danbury. Limited additional checking was conducted by Loring Laboratories of Calgary, and General Testing of Vancouver. All laboratory results and figures from the check assaying program are on file for review in Newmont's Vancouver office.

a) Sample Preparation Checks - Sample preparation was identified as an important factor affecting the reliability of assays. Checking was done by taking another 250 gm split from the bag of crushed rejects, then pulverizing and assaying it. Results are summarized below in Table A-I.

TABLE A-I

SAMPLE PREPARATION CHECKS

ON RESPLITS FROM CRUSHED REJECTS

Program	Number of Samples	% Deviation From Original 0-5%	6-10%	11-20%	> 20%
Surface Drilling 1976-79	51	27	12	26	35
Underground Drilling 1981	100	47	22	22	9
Combined Drilling 1976-81	151	40	19	23	18

There is a definite improvement in the results from the underground program over those of the surface program. In the surface drilling only 39% of the resplits agreed to within 10% and a distressingly high 35% disagreed by more than 20%. This compares to 69% of the resplits in the underground drilling agreeing to within 10% and only 9% varying by greater than 20%. It is possible that this substantial improvement, and resulting increased confidence in the assaying through the underground drilling, is due to the finer crushing instituted in 1979 and

1981. Sample preparation on the coarser crushed material was demonstrated to be a problem at Newmont's Danbury lab. Pulps from the original and the rejected splits were analyzed by XRD and XRF and compared. Discrepancies in MoS_2 assays were reflected in mineralogic discrepancies between corresponding pulps, confirming a sampling error. The modification to a finer crush in the sample preparation was designed to eliminate this error. Although an improvement in reproducibility of assays on resplits from the surface drilling done in 1979 is not apparent, the sample size (18 samples) may be too small to be significant. The improvement in the finer crushed samples from the larger underground drilling is certainly evident.

The combined drilling figures, 59% of the samples agreeing to within 10% and 18% varying by more than 20%, reflects greater discrepancies than could be wished. However, the improving trend from the surface to the underground comparisons and the larger more significant sample from the underground drilling provide for a more encouraging conclusion. The finer crush does appear to improve the reproducibility; and the larger the sample number, the less significant the individual discrepancies between original and resplit assays.

b) Assay Checks - The checking program also investigated the reliability of assaying procedures by re-assaying original sample pulps on the same samples as those selected for sample preparation checks. The results are summorized in the following Table A-II.

TABLE A-II

ASSAY CHECKS ON ORIGINAL PULPS

Program	Number of Samples	% Deviation From Original 0-5%	6-10%	11-20%	20%
Surface Drilling 1976-79	51	59	23	12	6
Underground Drilling 1981	18	61	28	11	0
Combined Drilling 1976-81	69	59	25	12	4

As would be expected, there is substantially better agreement on re-assayed pulps than reject resplits. The assay checks on pulps agree to within 10% in 84% of the samples overall, and disagree by more than 20% in only 4% of the samples. As well, there is no great contrast between the samples from the surface drilling and those from the underground drilling. The smaller number of samples involved is even significant since the much better agreement is based on a much smaller sample (69 vs 151). A larger sample could quite probably lead even more than 84% of the assay pairs agreeing to within normal assay limits of ±10%. Only 18 pulps were re-assayed from the underground drilling because the project was shelved before the remaining 83 could be done. They are in storage at Danbury. Overall, it appears that the analytical laboratory procedures produce reliable results.

Pulps and Rejects

All sample pulps have been returned from Chemex Labs Ltd. All sample pulps from the surface drilling (1976 - 1979) have been sent to Newmont Exploration's lab in Danbury as part of a continuing alteration study, and are held in storage there. The sample pulps from the 1980 - 1981 underground program are stored at Newmont's Similkameen Mine at Princeton, B.C., except for the four pilot holes 80-5, 81-4, 81-5 and 81-6 which are stored at Danbury with the surface drilling pulps.

All reject material, except that sent to Danbury for metallurgical purposes, has been discarded after being held until it was deemed that no further use would be served by its retention.

METALLURGY

Scope of Test Work

Limited investigations on five composites of drill core were carried out in 1979 and 1980. These represented the "A" and "B" zones of the Trout Lake orebody. (Gorken 1980)

In the period 1981 - 1982, a more extensive bench-scale investigation was carried out on a composite of all ore grade core from a horizontal hole drilled along the center line of the adit on the 960 Level and on bulk samples taken from material removed in driving the adit. Composites of the major rock types encountered in the adit bulk samples were tested individually and a master composite made up on a weighted basis for testing. (Gorken, 1981; Nabbs, 1982)

Metallurgical Results

A summary of the metallurgical results obtained in closed circuit testing of the drill core composites tested in 1979 and 1980 is provided in Table M-I and the results on the horizontal core composite and adit bulk samples are summarized in Table M-II.

Testing of the five composites of core from holes drilled from surface indicated a simple flowsheet incorporating primary grinding to -65 mesh, rougher and scavenger flotation of the molybdenite with frother only; regrinding of the rougher concentrate and multiple cleaning stages would recover about 90% of the molybdenite in a concentrate assaying 90 - 92% MoS_2.

TABLE M-I

SUMMARY OF METALLURGICAL RESULTS 1979 - 1980

Composite No.	Zone	Hole No.	Interval	Sodium Cyanide	Assay Head % MoS_2	Final Concentrate %MoS_2	% Cu	% Fe	MoS_2 % Recovery
1	B	78-5	61' from 1,075' to 1,150'	No	0.366	96.91	0.028	0.22	86.9
2	B	78-5A	97' from 1,750' to 1,860'	No	0.274	93.55	0.33	1.02	91.0
				Yes	0.274	94.87	0.01	1.04	90.6
3	B	79-1	103' from 1,477' to 1,580'	No	0.150	92.15	0.30	0.79	81.8
				Yes	0.150	92.14	0.01	0.49	82.4
4	A	79-1	142' from 498' to 640'	No	0.286	89.3	0.46	0.87	91.6
				Yes	0.286	91.9	0.02	0.48	89.9
5	A	79-4	140' from 230' to 370'	No	0.178	91.3	0.23	0.70	90.3

N.B. Drill core composites 6, 7 and 8 are in storage at Danbury for future testing.

TABLE M-II

SUMMARY OF RESULTS - ADIT SAMPLES - 960 LEVEL

Sample Description	Assay Head %MoS_2	Final Concentrate %MoS_2	Final Concentrate % Cu	Final Concentrate % Fe	MoS_2 % Recov.
Master Composite of Drill Core (80-5)	0.37	85.46	0.46	3.76	90.8
Master Composite of Drill Core (80-5)	0.37	90.89	0.44	2.22	83.3
Master Composite of Drill Core (80-5)	0.37	91.62	0.35	2.2	64.9
Granodiorite Quartz Diorite (bulk sampling)	0.175	93.90	0.45	1.7	89.7
Quartz Veining/Stockwork + Granodiorite Quartz Diorite (bulk sampling)	0.438	91.35	0.69	2.0	94.0
Silicified Schist (bulk sampling)	0.19	81.49	0.59	3.8	92.2
Silicified Schist (bulk sampling)	0.19	85.29	0.59	3.2	90.2
Silicified Schist (bulk sampling)	0.19	87.68	0.54	3.2	84.0
Silicified Schist (bulk sampling)	0.19	91.91	0.45	1.5	67.3
Silicified Schist + Granodiorite Quartz Diorite (bulk sampling)	0.45	85.89	0.39	2.8	88.8
Silicified Schist + Granodiorite Quartz Diorite (bulk sampling)	0.45	89.35	0.41	2.8	80.5
Master Composite* (bulk sampling)	0.23	79.16	0.59	4.7	89.0
Master Composite* (bulk sampling)	0.23	83.92	0.52	2.85	88.3

	% Wt	
* Master Composite =	74.3	Silicified Schist
	13.9	Granodiorite Quartz Diorite
	10.4	Silicified Schist + Granodiorite Quartz Diorite
	1.4	Quartz Veining/Stockwork

The final molybdenite concentrate contained up to 0.5% copper, which could be detrimental in the selling of this product. A small addition of sodium cyanide, not more than 0.01 lbs/ton of original feed, added to the final cleaner stages, effectively reduced the copper content to less than 0.05%. The cyanide addition is equivalent to 5.0 ppm of original feed. Chemical reactions with the components of the molybdenite concentrate and with gangue constituents of the ore when the cleaner tailings are recycled, plus hydrolysis of sodium cyanide are expected to reduce the cyanide level in the tailings to below the limits set forth in "Pollution Control Objectives for the Mining, Smelting and Related Industries of British Columbia, 1979".

In testing the Master Composite of the core from the horizontal Hole 80-5, a strong grade versus recovery relationship was detected, with a recovery of 90.8% at a concentrate grade of 85.46% MoS_2, 83.3% recovery at 90.89%, and 64.9% recovery at 91.62% concentrate grade.

Testing of the individual rock type composites of the adit bulk samples identified the Silicified Schist as the major cause of this problem. Satisfactory metallurgy was obtained on the other two major rock types. Unfortunately, this represents almost 80% of the ore on this horizon. Microscopic examination of concentrates identified the major diluent as non-opaque gangue with fine coatings of molybdenite. Examination of ore samples detected appreciable amounts of such material in the Silicified Schist, but little in the other rock types. While such middling particles can be produced in grinding, the major portion in the Silicified Schist are of geologic origin.

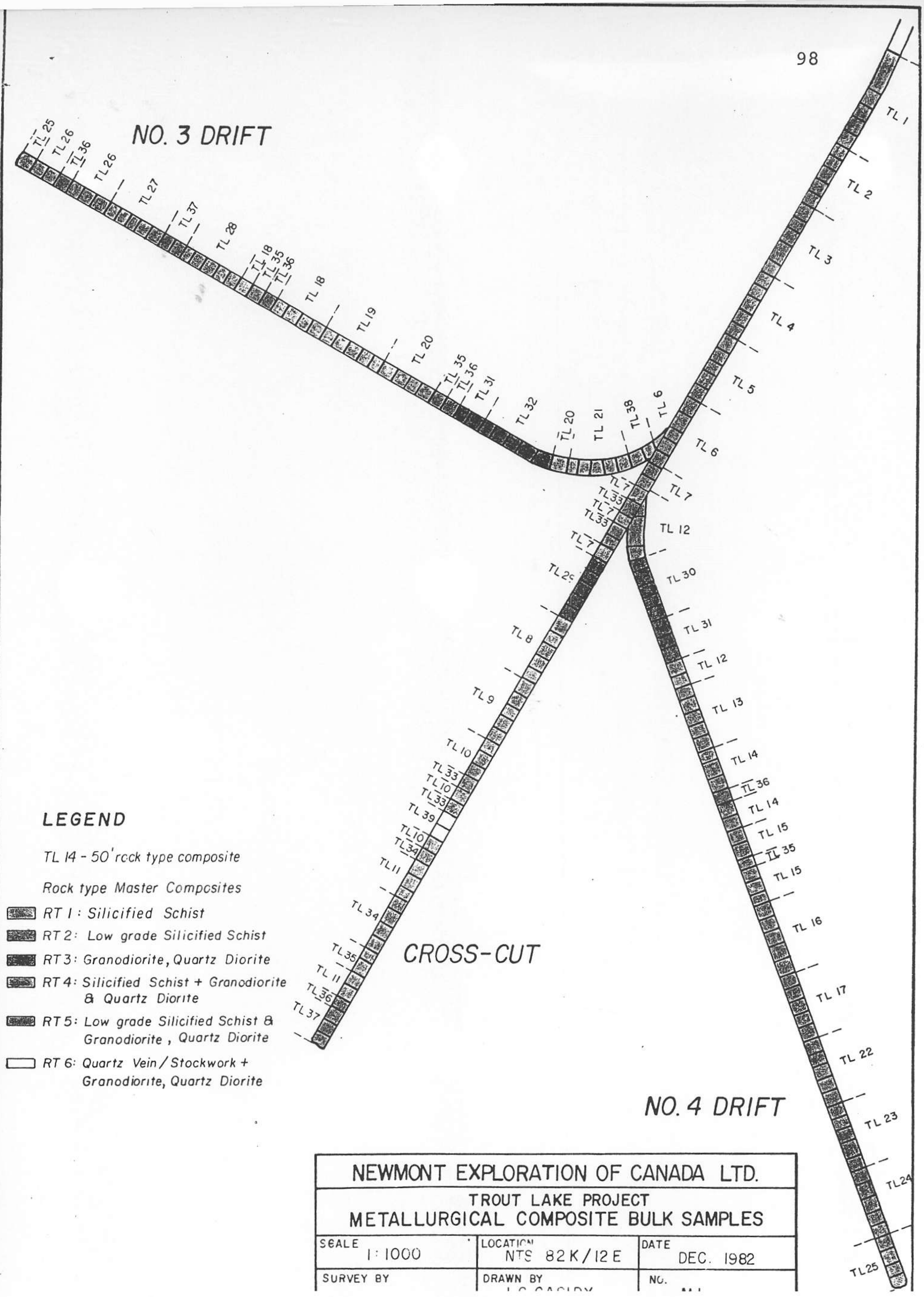
On the Silicified Schist composite sample molybdenite recovery decreased at a relatively constant rate with increasing concentrate grade:

Recovery, MoS_2 %	Concentrate, MoS_2 %
92.2	81.49
90.2	85.29
84.0	87.68
67.3	91.91

Testing of a Master Composite of the adit bulk samples, made up on a weighted basis of the Silicified Schist, the Granodiorite Quartz Diorite, and Quartz Veining/Stockwork ore grade intersections, gave similar results as obtained on the Silicified Schist alone. The improved metallurgy obtainable on the other rock types was masked by the predominance (about 80%) of the Silicified Schist in this composite sample. The rounds of which the Master composites are composed are shown in Fig. Ml.

Addtional Test Work Required

The metallurgical results obtained on the Silicified Schist are disappointing, particularly as this rock type represents about 80% of the ore grade adit samples. However, these samples represent only one horizon, the 960 Level. The strong grade/recovery relationship was not noticeable in the testing of the core holes drilled from surface. It is important to determine whether all Silicified Schist has similar characteristics and how much of the total reserves are contained within this rock


NO. 3 DRIFT
TL 25
TL 26
TL 36
TL 26
TL 27
TL 37
TL 28
TL 18
TL 35
TL 36
TL 18
TL 19
TL 20
TL 35
TL 36
TL 31
TL 32
TL 20
TL 21
TL 38
TL 6
TL 1
TL 2
TL 3
TL 4
TL 5
TL 6
TL 7
TL 33
TL 7
TL 33
TL 7
TL 7
TL 12
TL 29
TL 30
TL 31
TL 8
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TL 10
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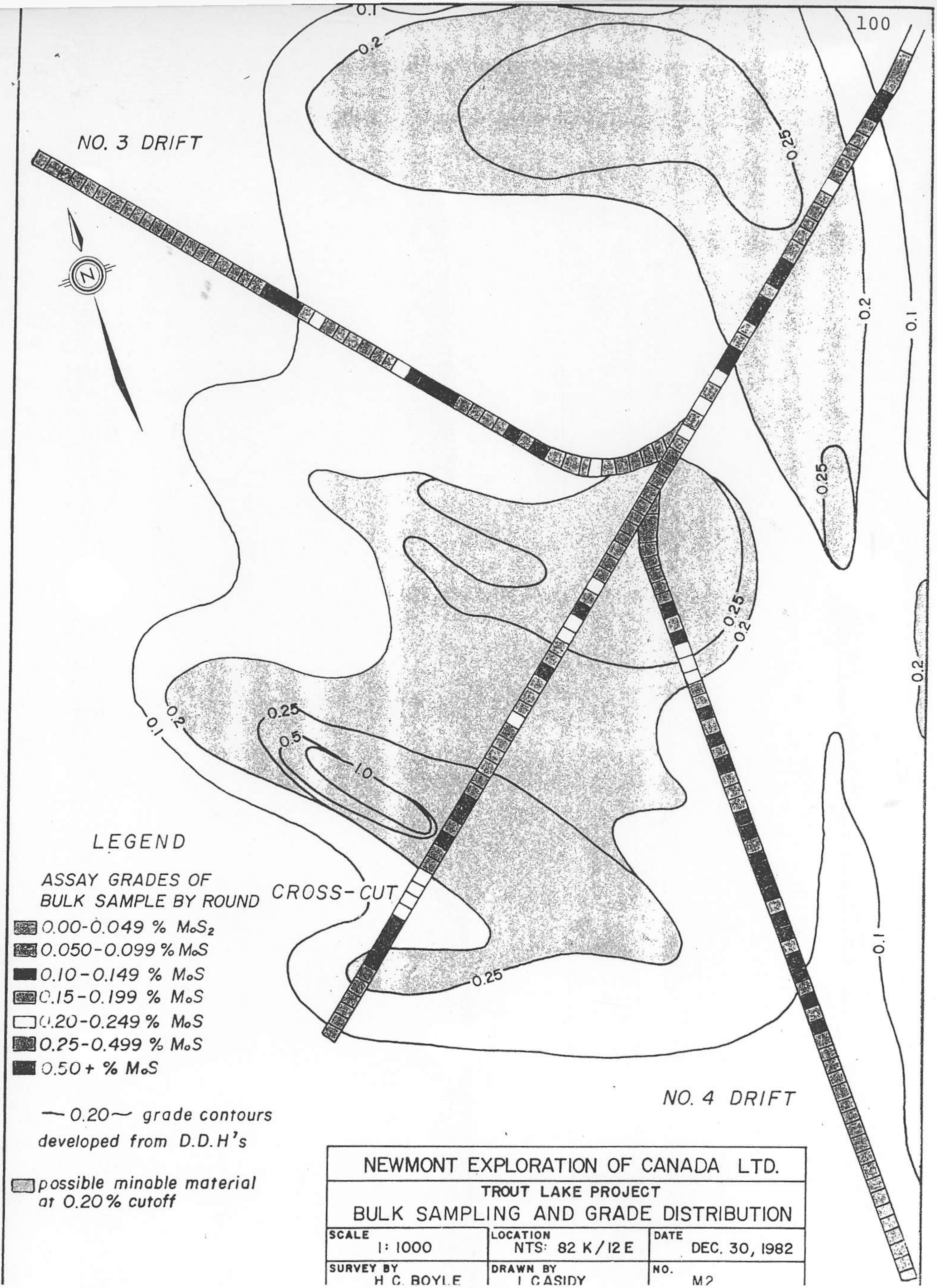
TL 33
TL 39
TL 10
TL 34
TL 11
TL 14
TL 36
TL 14
TL 15
TL 35
TL 15
TL 34
TL 16
TL 35
TL 11
TL 36
TL 37
CROSS-CUT
TL 17
TL 22
TL 23
TL 24
TL 25
NO. 4 DRIFT
LEGEND
TL 14 - 50' rock type composite
Rock type Master Composites
RT 1 : Silicified Schist
RT 2: Low grade Silicified Schist
RT 3: Granodiorite, Quartz Diorite
RT 4: Silicified Schist + Granodiorite & Quartz Diorite
RT 5: Low grade Silicified Schist & Granodiorite , Quartz Diorite
RT 6: Quartz Vein / Stockwork + Granodiorite, Quartz Diorite
NEWMONT EXPLORATION OF CANADA LTD.
TROUT LAKE PROJECT
METALLURGICAL COMPOSITE BULK SAMPLES
SCALE 1: 1000
LOCATION NTS 82 K / 12 E
DATE DEC. 1982
SURVEY BY
DRAWN BY
NO.

type. Fig. M2 illustrates the distribution of grades in the bulk samples as compared to the distribution of grades as developed from diamond drilling.

Extensive drilling has been carried out from the adit, which should provide suitable samples for this testing. A number of such samples have been received at Danbury, but testing was halted on instructions from New York. Drill core logging could be used to determine what percentage of the total reserves occur in this refractory rock type.



NO. 3 DRIFT
N
0.1
0.2
0.25
0.2
0.1
0.25
0.25
0.2
0.2
0.2
0.1
0.25
0.5
1.0
0.1
0.25
CROSS-CUT
NO. 4 DRIFT
LEGEND
ASSAY GRADES OF
BULK SAMPLE BY ROUND
0.00-0.049 % MoS2
0.050-0.099 % MoS
0.10-0.149 % MoS
0.15-0.199 % MoS
0.20-0.249 % MoS
0.25-0.499 % MoS
0.50+ % MoS
— 0.20 — grade contours
developed from D.D.H's
possible minable material
at 0.20 % cutoff
NEWMONT EXPLORATION OF CANADA LTD.
TROUT LAKE PROJECT
BULK SAMPLING AND GRADE DISTRIBUTION
SCALE 1: 1000
LOCATION NTS: 82 K/12 E
DATE DEC. 30, 1982
SURVEY BY H C. BOYLE
DRAWN BY L CASIDY
NO. M2

Plant Design Data

Plant design data were developed from test work carried out on composites prepared from the bulk sampling program.

Feed	
Specific Gravity	2.73
Bulk Density at -½ in.	121 lb/cu ft
Grind	
Nominal 65 mesh	
98% passing 65 Tyler mesh	
65% passing 200 Tyler mesh	
Grinding Power Requirements	
Estimated net kwh/ton	8.5
With 25% Safety Factor	10.6 net kwh/ton
Installed hp @ 90% efficiency (per 1,000 tpd of mill feed)	660 hp
Estimated steel ball consumption (based on 0.15 lbs/ton per kwh/ton)	1.3 lbs/ton
Estimated liner consumption (10% of steel ball consumption)	0.13 lbs/ton
Primary Cyclone	
Ball Mill discharge	70% solids
Cyclone Feed	56% solids
Cyclone Overflow	38% solids
Cycline Underflow	65% solids
Circulating Load	350%
Regrind Circuit	
Mill Feed	50% solids
Estimated net kwh/ton of circuit feed	4.5 net kwh/ton
Installed hp @ 90% efficiency (per 1,000 tpd of primary mill feed)	15 hp
Regrind Cyclone	
Cyclone Feed	40% solids
Cyclone Overflow (90% -325 mesh)	25% solids
Cyclone Underflow	50% solids
Circulating Load	350%

Flotation Cell Volume Requirements
(per 1,000 tpd of Mill Feed)

Operation	Retention Time Minutes	Required Volume Cu Ft
Rougher Flotation	10	530
Scavenger Flotation	10	520
1st Cleaner	16	20
2nd Cleaner	8	5
3rd Cleaner	5	3
4th Cleaner	4	2
5th Cleaner	3	2

Tailings Thickening

Feed	38% solids
Specific Gravity of Solids	2.73
pH	8.2 to 8.4
Pulp Temperature	Ambient
Flocculant Addition	0.02 lbs/ton
% Solids Underflow	60
Area Requirements	5 sq ft/ton/24 hrs
Area Required (per 1,000 tpd of mill feed)	5,000

Reagents

Addition Point	lbs/ton of New Mill Feed		
	Pine Oil	Dowfroth 250	NaCN
Rougher Flotation	0.027	0.022	
Scavenger Flotation	0.041	0.036	
1st Cleaner		0.007	
2nd Cleaner		0.007	
3rd Cleaner		0.007	
4th Cleaner		0.003	0.005
5th Cleaner		0.002	0.005
TOTAL	0.068	0.084	0.010
lbs per 24 hours per 1,000 tpd mill feed	68	84	10

MINING STUDY

Case 1: 3,000 Tonnes Per Day

a) General - The exploration adit at Trout Lake was driven with a cross-section 12 ft wide by 15 ft high until the ore zone was reached. It was then reduced to 10 ft x 12 ft in the ore zone and in the diamond drill crosscuts. The larger section was chosen so the adit could serve as a production haulage level. The elevation of the adit, 960 m (3,150 ft) was selected to conform with possible concentrator and tailings storage sites, as well as exploration and topographic considerations.

A preliminary study of a mining method for the Trout Lake mineral deposit was made in some detail by F. T. Hancock (Report dated June 29, 1982). This study assumed a treatment rate of 3,000 tonnes per day, and used recent actual and estimated operating and capital costs adjusted to 1982 Canadian dollars.

The mining method considered was open stoping with delayed cemented fill, using adit and ramp above the 960 m Level and shaft and secondary ramp below the 960 m Level. An adit at the 1,250 m level would be required for development of the upper ore zones and for ventilation. Large diameter down-the-hole drills would be used for production drilling with conventional equipment used for extraction and secondary haulage. Main line haulage on the 960 m Level would be by an electric trolley rail system.

Although considerable difficulty was encountered with poor ground conditions and excessive water in the adit from the portal to the main fault, the rock in the ore zones was quite competent. However, it is considered that cemented fill would be required in the primary stope panels after mining to permit extraction of the secondary panels. The total amount of sands available from cycloned tailing would be required for tailing dam building. Because of local scarcity of other suitable material, provision was made in the costs for the use of crushed surface rock for fill.

Mining was based on the reserves calculated to a 0.20% MoS_2 cutoff grade, assuming 15% dilution at 0.14% MoS_2. The estimated geological reserves at this cutoff grade are 11,736,000 tonnes at 0.362 MoS_2. After fitting the proposed stoping panels to the mineralized outlines and allowing for sill and crown pillars (some of which unavoidably occur in high grade areas), the estimated recoverable ore is reduced to 8,189,000 tonnes at 0.328% MoS_2. Some of the material in the thick sill pillar around the 950 m level could probably be recovered later in the operation, depending on ground stability.

If 50% could be recovered, this might add about 1,500,000 tons to the mineable reserves. At a milling rate of 3,000 tonnes per day, the production life would be 7.5 years with perhaps an additional 1.4 years from pillar recovery.

b) Costs - Mining costs per tonne milled are estimated to be:

	Above 960 m Level	Below 960 m Level
Equipment	1.14	1.14
Development	3.87	9.03
Mining	7.43	8.82
TOTAL	$12.44	$18.99

Mining costs include all salaries, wages and fringes (30%) up to and including the mine superintendent. The costs attributed to equipment and development would be somewhat reduced if recovery of the sill pillar proved successful.

Case 2: 1,500 Tonnes Per Day

Because of the short mine life which would result from mining the estimated reserves at 3,000 tonnes per day, a lower treatment rate of 1,500 tonnes per day was considered. A preliminary study was made by W. G. Martin (Report dated August 25, 1982).

This study considered actual costs at Noranda's Boss Mountain molybdenum mine, which has a milling rate of 1,500 tons per day from an underground operation, with mining costs adjusted to Hancock's figures.

Martin concluded that the lower mining rate was not advantageous.

Economic Considerations

Using the Boss Mountain experience, Hancock's estimates for mining costs and pro-rated early order of magnitude estimates by Newmont for capital costs, Martin calculated the price of molybdenum required to cover operating costs and return of capital. He concluded that a price of approximately $15.00 Can. per pound of Mo would be required for both the 3,000 and 1,500 tonnes per day milling rates.

At presently projected prices for molybdenum the Trout Lake property is not economically attractive. However, the low tonnage of ore estimated at a 0.20% MoS_2 cutoff grade, together with the relatively poor extraction expected from the shape of the deposit and the proposed mining system, results in high per ton costs for development and capital expenditures. Reserves were calculated at a 0.10% MoS_2 cutoff grade (48,738,000 tonnes at 0.193% MoS_2) but while the tonnage was much greater, the grade is considered too low for a viable mining operation.

The envelope of material surrounding the 0.20% outlines, used in dilution calculations, averaged 0.14% MoS_2. This suggests that there could be other cutoff grades, perhaps 0.15% or 0.18% which might result in a more favourable combination of tonnage and grade and permit a viable operation at a price considerably below $15.00 per pound. A range of cutoff grades should be investigated, preferably with the aid of a computer program when this is available.

TAILINGS STORAGE

Klohn Leonoff Consulting Engineers were engaged to carry out a preliminary study of possible tailings storage facilities for the Trout Lake property. Their findings and recommendations are given in their report "Feasibility Report on a Tailings Storage Facility for Trout Lake Mine", July 29, 1981.

Their engineers visited the property and, after an examination of the area, identified two possible sites in the vicinity of the mine. These were designated "Site A" and "Site B". Preliminary foundation investigations and initial dam designs were carried out for both sites.

Site A

Site A, considered to be the preferred site, is located immediately downslope from the adit portal and would be fairly close to the proposed treatment plant. The storage area would be created by constructing dams at each end of the elongated depression between a ridge and the SW valley slope.

The underlying material and bedrock appear suitable for dam foundations, although a possible fault, inferred from air photos, would have to be checked. Two creeks flow into the area from the SW and these would need to be diverted. Because strict control of seepage would be required in view of the important trout fishery downstream, investigation of the permeability of the site and provision for intercepting and returning seepage would be necessary. Sources of suitable material for both starter dam and on-going dam building would have to be explored.

Site A has an estimated storage capacity of 26 million cubic metres or approximately 47 million tonnes. Klohn Leonoff assumed the dams would be constructed of borrowed or quarried material and that 40% of the tailings would be removed as cycloned sands for mine backfill. Under this assumption the storage area could accommodate slimes equivalent to 54 million tonnes of ore. Hancock, in his mining report, assumed that tailings sands would be used for dam building and not for backfill. The cost advantages and effect on storage capacity of the use of sands for dams vs backfill would have to be further evaluated.

Site A is considered to be the "preferred" site. Its main advantages are its closeness to the concentrator, good foundations for dams, avoidance of relocation of the highway and less environmental impact than Site B. Its main disadvantage is its limited storage capacity.

Site B

Site B is located about 2 km NW of the adit portal in a flat swampy area between the drainages of Beaton and Wilkie creeks. Provincial Highway 31 traverses the site on the north side of the valley. This highway would have to be relocated further up the hillside to accommodate a tailings storage facility. Because of the rock excavation involved, such a relocation would be a major undertaking.

This site has a potential storage capacity for more than 100 million tons and has a higher volume of tailings storage per unit volume of dam than Site A. However, it has the disadvantages of being much less ecologically acceptable, in full public view, with rather poor underlying material to support the dams.

It is also considerably further from the concentrator and would require the expensive highway relocation mentioned previously.

The site should probably be considered only if the total tonnage planned for treatment is larger than the capacity of Site A, or it could possibly be supplemental to Site A on a smaller scale.

Land Requirements

Most of the land required for potential tailings storage at Site A is covered by mineral claims owned by the Company, but some is owned by others. An agreement was recently signed with Oakey Holdings Ltd. to permit using the surface of the Oakey claims. If the project goes ahead it would be necessary to purchase Lot 772 from Cancel Company and a small portion of Lot 771 from its owner.

Site B is completely covered by located mineral claims held by the Company but it is also on part of Lot 770, which is owned by Cancel.

POWER

Preliminary discussions were held in 1979 and 1980 with British Columbia Hydro and Power Authority regarding the supply of the electric power to the Trout Lake property. Hydro advised that power could be supplied from a point about two miles from Nakusp. The line would be approximately 46 miles in length and would be built to supply 10 to 20 megawatts at 138 kv. The cost was roughly estimated to be $8,000,000 of which $2,000,000 would be refundable dependent on power consumption.

Hydro was notified when the Trout Lake operations were terminated in 1981. However, an estimated amount of 20 Mw of power was tentatively reserved for Trout Lake in Hydro's long range power consumption forecasts.

WATER

Process water requirements for the Trout Lake property, assuming that tailings would be thickened to 50% solids, are estimated to be 1,000 U.S. gpm for a 6,000 tpd treatment rate or 500 U.S. gpm for a 500 tpd rate.

? typo error likely - 1500?

TNM 2.00.4

This would be substantially reduced by reclaiming water from the tailings impoundment pond once this was operational. Considering the low evaporation rate at Trout Lake and that seepage would likely have to be caught and returned to the pond, water recovery could reach 80%. With the tailings system in full operation, water requirements could be approximately 100 or 200 U.S. gpm.

Alternative sources of water supply for the project were studied by Klohn Leonoff Consultants Ltd. and their findings were given in a draft report dated January 11, 1980. They considered the capital and operating costs and the environmental impact of the following sources:

1. pumping from Trout Lake,
2. Wilkie Creek by gravity,
3. pumping from Wilkie Creek or wells beside Wilkie Creek.

They concluded that there was adequate water in the immediate vicinity, but that sources other than those considered were inadequate. For reasons of costs and environmental considerations they recommended alternate (3.), pumping from Wilkie Creek or preferably from wells in the gravels beside the creek. They also recommended further investigation of Wilkie Creek, particularly in winter.

Further work was carried out during 1980 and 1981. This is reported in the Klohn Leonoff report of July 30, 1982, entitled "Hydrologic and Climatologic Data Collection". Besides snowpack, rainfall and other climatologic data, the report gives streamflows for Wilkie and Humphries Creeks and flow from the adit.

During driving of the adit, strong flows of water, often in excess of 2,000 Imperial gpm, were encountered. This flow decreased slowly after completion of the underground work, but did not go below 1,000 Imperial gpm at any time up to cessation of operations. If this flow remained constant at approximately this rate, it could supply the estimated process water requirements or at least substantially reduce the amount pumped from Wilkie Creek.

Application was made August 10, 1980 to the Comptroller of Water Rights, Department of Environment, British Columbia, for a water licence on Wilkie Creek, including payment of $5,000.00. This application is still in process. Part of this payment would apply to the first year's fees when the installation was completed and the water actually used.

A program of water quality sampling was instituted to establish background data well before production started. Samples from a number of locations on the creeks in the vicinity of the property were taken four times per year between October 1978 and May 1982. Samples of the waterflow from the adit were also taken regularly.

Analyses were carried out by B.C. Research and later by Beak Consultants. Samples were taken by the mine staff or by consultants. Results are on file and copies were sent to the government authorities.

HOUSING

The housing requirements for the Trout Lake property would of course depend of the scale of operations finally decided on. The total number of employees required for a 3,000 tpd operation is estimated to be about 300, with a community of about 700.

Trout Lake is located about 84 km from Nakusp and Revelstoke, the two nearest towns. Both are presently considered too far from the mine for commuting, especially under winter conditions. Revelstoke is also separated by a ferry, which does not operate 24 hours. The present settlement at Trout Lake is too small and has too few facilities to be of assistance.

Unicon Project Consultants were engaged to make a preliminary appraisal of the possible townsite requirements and advise on a site. They made a visit to the property and submitted a short report dated December 10, 1979. After examining sites at Beaton and Galena Bay, as well as considering Nakusp and Revelstoke, they recommended that a townsite and related facilities be established on a site to the east and north of the existing settlement.

The proposed site is located on Crown Land and would require an area of between 27 and 43 ha, depending on the size of the mine workforce. Mineral claims were staked by Newmont to cover this land. The possibility of a townsite at this location was verbally discussed with the Minister of Lands and his officials. While it was not possible to ensure obtaining this land until plans for production were more definite, the Minister felt there should be no problem.

ENVIRONMENTAL STUDIES

Conclusions and Recommendations

The following conclusions and recommendations are quoted directly from the report of Beak Consultants Ltd.:

"The development of a molybdenum mine at the Newmont Trout Lake property can likely be accomplished without serious detrimental effects to the existing environment of the area. Construction and operation of the mine and concentrator will undoubtedly result in changes, but providing mitigative measures are incorporated during all phases of the development, environmental impacts should be kept to a minimum.

"The most valuable fisheries and wildlife habitats in the Trout Lake area are found in the Wilkie Creek lowlands. The marshes, wetlands and streams in the valley bottom support diverse communities of flora and fauna. They provide breeding areas for a diversity of bird species, summer range for ungulates and essential spawning area for sport fish resources.

"For these reasons it is strongly recommended that development in the area avoid disturbances to the Wilkie Creek lowlands. If there are plans for extension of the Trout Lake townsite to accommodate personnel and families, expansion should not include fill and construction near Wilkie Creek and its marshes.

"The mine and concentrator should not cause serious air quality impacts in the area, as a roaster is not planned. All facilities should be located to take advantage of the topography at the site to ensure portection of Wilkie Creek. Mitigative measures that are suggested should be followed to maintain water and air quality. If these environmental components are maintained near their existing conditions, it will ensure fish and wildlife resources are protected. Construction at the site will cause some disruption of vegetation, but if kept to a minimum, this is not expected to cause significant impacts.

"It is recommended that an ongoing water quality program be instituted during both construction and operation of the mine. Regular sampling of mine water and effluent will allow the identification of potential problems and rectification before there is a serious degradation of fish and wildlife habitats."

General

An environmental assessment of the Trout Lake property was conducted during the latter part of 1979 and through the 1980 and 1981 seasons. While some further work would be required in reference to the specific impacts related to the detailed development plan for the project, when this is decided, the reports provide a substantial data base for a Stage 1 report as required in the procedures for obtaining approval from the Ministry of Energy Mines and Petroleum Resources.

In 1979, Environcon Limited were retained to perform an environmental overview study for the proposed development of the property. The purpose of this study was to identify the main potential concerns which could result from the development of an underground mine. These were water quality, fish and wildlife. The fish resources, particularly rainbow trout, are of paramount importance. As no roaster is planned, ambient air quality would not be a concern. Detailed findings by Environcon are set out in their report "Environmental Overview", January 15, 1980.

In March 1980, after a review of a number of proposals from consulting firms, Beak Consultants Limited were retained to conduct environmental and socio-economic studies sufficient to establish a data base for a Stage 1 report. Their field crews carried out surveys and studies during 1980 and 1981. Detailed results and conclusions are contained in two reports: "The Trout Lake Property, Preliminary Stage 1 Environmental Assessment, August 1981" (Reference K4515) and "Environmental Studies in the Trout Lake Area, 1981; March 1982" (Reference K4614). Included in the first Beak report is a detailed socio-economic study for the project prepared by Horsman Associates.

Air Quality/Noise

Development of a molybdenum mine and mill complex at Trout Lake is unlikely to have significant impact on air quality, particularly as no roasting plant is contemplated. Dust emmissions from the mine and mill would be controlled and noise would be local and not excessive, so no problems are expected.

Water Quality

Development of a mine and concentrator at Trout Lake is possible with little impact on surface water quality providing that waste treatment facilities are properly engineered to avoid potential hazard to the acquatic resources of the valley. The principal potential impacts associated with the development could be seepage from tailings ponds, run-off from plant site or roads, in-plant spills and sanitary waste from the plant and townsite. In normal conditions, recirculation of water from the tailings pond for process water, together with seepage collection and return, if necessary, will eliminate surface water discharges. Collection and treatment facilities would have to be provided to handle excess run-off and storm flows.

Proper engineering design of the tailings dams would of course be mandatory to avoid the potential hazard of a dam failure discharging tailings into Wilkie Creek. Because of the fine sediments associated with some of the soils in the area, proper engineering practices and precautions would be required during the construction phase to protect water quality.

As noted in the section on water, a program of sampling and analysis of surface waters on a monthly and quarterly basis was started in October 1978 and continued through May 1982. No appreciable difference was apparent in the content of heavy metals in water draining the adit compared to stations upstream or downstream of the confluence of this water with Wilkie Creek.

Aquatic Ecology

The streams in the north end of the Trout Lake valley contain populations of Dolly Varden char, kokanee and rainbow trout, which are all important to the sport fishing in the area. Of these streams, Wilkie Creek contains the most important fish habitat. Most of the previous aquatic studies conducted in the area concentrated on the rainbow trout that utilize the outlet of Trout Lake at Gerrard for spawning. Here, fish management and spawning facilities have been installed for the widely known trophy status "Gerrard stock" rainbow trout. The Trout Lake headwaters have received only limited study until those commissioned by Newmont.

The aquatics field program in 1980 consisted of two general components: (1) in-stream surveys to monitor the relative and seasonal abundance of fish species in the Trout Lake headwaters and (2) spawning surveys to monitor the numbers, timing and distribution of spawning fish in these waters. These studies were continued and amplified in 1981 to make the data base more complete. During the 1981 season, samples of fish taken from Wilkie Creek were retained for analysis of heavy metals content in muscle tissue. Results showned that fish taken from Wilkie Inlet D, the system receiving water draining from the adit, did not contain metal concentrations in excess of those contained by fish from Wilkie Creek.

Development of the Trout Lake property can be accomplished with minimum impact to the existing water and aquatic habitat quality in the area providing that proper mitigative measures are incorporated in the project design. The potential aquatic environment impacts resulting from mine construction and operation, and recommended mitigative measures are summarized as follows:

a) Suspended Solids - The greatest environmental concern identified to this time is an increase in the amount of silt and sediment entering the local streams. All efforts should be made to isolate tailings areas and waste rock dumps from water courses. Vegetation should be retained as much as possible. Run-off ditches, collection traps and settling ponds should be provided if required.

b) Streambed Disturbance - Disturbances to streambeds could cause increases in silt load and alteration of the streambed, resulting in damage to aquatic life. Borrow material should be obtained from sites away from streambeds. The movement of heavy equipment in stream channels should be avoided. The natural drainage pattern should be maintained as far as possible.

c) Toxic Substances - Toxic substances in water courses can kill or permanently damage fish and other aquatic life. The treatment of mine water, runoff and sewage may be required. Impermeable dykes around tailings areas and waste dumps could be required depending on the degree of toxicity of leachates, seepage or processing chemicals. Oil and chemical storage should be located away from Wilkie Creek feeders, with protective berms and spill contingency plans. A continuous program for monitoring effluent and mine water quality would be required.

d) Water Requirements - Reduction in stream flows from extraction of large volumes of water could be detrimental to aquatic life. However, recycling of water from the tailing system will minimize fresh water make-up requirements, and the portal flow could likely supply a substantial portion of the make-up water, if it continues at its present rate. Water withdrawal from Wilkie Creek will therefore likely be minimal, or perhaps not required, and no problem is anticipated.

Wildlife

The Trout Lake area supports a number of wildlife species. These include woodland caribou, mule and whitetail deer, mountain goat, grizzly and black bear, wolf, coyote and a number of furbearers. Fortunately, there are no endangered species in the area.

In 1980 aerial surveys were conducted in summer, fall and winter. Direct observations were recorded by field personnel throughout the seasons and information from trapline records and from mine staff and local residents was also recorded. The work was continued in 1981 with efforts concentrated on woodland caribou, including two aerial flights. It was concluded that very few caribou are present in the area and that the Lardeau range is not used to any extent by caribou.

Although wildlife resources of the Trout Lake area are not of regional significance, it is assumed that there is a necessity to preserve existing wildlife populations. Because the size of the proposed mine, its ancillary services and the increase in population are not yet known, concerns are addressed only in general terms. These would be principally the alieniation of critical habitat by mine buildings and tailings ponds, noise disturbance and pressure from an increase in human population.

Birds

A survey of birds in the Trout Lake area was conducted through a literature search and field surveys. Lists of the terrestrial and aquatic bird species are included in the report. Waterfowl breeding potential is estimated to be generally low, although same broods were noted on the beaver ponds in the Wilkie Creek lowlands. No bird species identified to date at Trout Lake is on the endangered species list.

The main environmental risks are alteration of water quality and reduction of flow because of water extraction. Mitigative measures outlined previously would eliminate these risks. Careful planning and an educational program would also be required to minimize the effect of the increased number of people in the area.

Soils and Vegetation

The general area around Trout Lake, the West Kootenays, was subject to extensive glaciation. This glacial history is reflected in the composition of the surficial deposits in the immediate project area. These are primarily glaciofluvial deposits in the lowlands and morainal deposits (till) on the lower to middle slopes. Soil resources within the Trout Lake area were recently described and mapped by the British Columbia soil survey and the results in the report are based on this survey. None of the immediate mine area is considered suitable for agriculture.

The predominant soil subgroup in the lowland Wilkie Creek area is the Orthic Humic Gleysol, which is poorly drained with frequent flooding a characteristic. The lower to middle slopes are orthic humoferric podzols of the Kuskanax, Stubbs and Cataract soil associations. The seepage phase of the Kuskanax occurring on the slope below the mine site is imperfectly drained.

Concerns relate to suitability or limitations of the soils for various storage facilities and construction related activities. A table of the suitability of the various soil types is included in the report. It is assumed that no construction activities will occur in the lowland soils. The limitations of the soils for septic tank absorption fields range from slight to severe and these areas will require careful locating. Similarly the sedimentation characteristics of the soils will have to be carefully considered to prevent damage to water courses.

Most of the area in the vicinity of the mine has been logged recently and is presently in a state of regeneration. The vegetation occurring at the mid to lower elevations of the mine area is part of the Interior Western Hemlock wet belt. Very little information on the immediate area existed prior to the present study so a fairly detailed sampling program was conducted. Red cedar and western hemlock are the predominent species with a few black cottonwood and white birch.

Socio-Economic Study

A study of the social and economic aspects of the Trout Lake property was made by Horsman Associates for Beak Consultants Limited and is incorporated in the Beak 1980 report. Although no specific survey of community opinion was carried out, no direct opposition to the mine was revealed during interviews. The majority of the local residents are dependant on resource development, and while concern with protection of the environment, were also interested in population growth and improving services in the area. The mine would be a benefit to the region economically and socially. It is Horsman's opinion that development of the mine would not be opposed by any environmental group.

Some potential employees could be recruited in the area, but the pool is quite small and probably limited in trade skills. Horsman recommends that permanent housing should be in Nakusp rather than at Trout Lake, contrary to recommendations of Unicon Consultants (see "Housing") and the opinion of Newmont personnel.

The study was done in much greater detail than required for the Provincial Guidelines and is designed for client internal information rather than for direct submission to government. If a decision is made to proceed with the project it would be essential that the report be critically reviewed and edited. In particular the recommendation regarding employee housing in Nakusp, with the work force commuting to Trout Lake, especially in winter, is questionable. Newmont's extensive and successful experience with employee housing and recruitment, such as at Princeton, should not be disregarded.

HYDROLOGY AND CLIMATE

To provide key information required for submission to the Provincial Government and in the design of water supply and tailings storage facilities, Klohn Leonoff Limited were commissioned to establish a streamflow monitoring program and a climatological station at the mine site. Data were collected from the spring of 1980 through November 1981 with the assistance of Newmont staff.

All data collected, including information on temperature, precipitation, snowpack, streamflow and evaporation, are presented in the report entitled "Hydrologic and Climatological Data Collection" by Klohn Leonoff, File No. PB2850-0201, July 30, 1982. Wind speed and direction were recorded but the period was considered too short for detailed analysis of the results. The wind data are on file with Newmont.

January 31, 1983

Vancouver, B.C.

H. C. Boyle

J. H. Parliament

PHOTOGRAPHS

Photo No.	Description
1	Looking south to Project Site in clearing. Trout Mountain in cloud at left. Location of deposit at surface is circled.
2	Project Site close up. Adit portal, covered track, settling pond, shop and dumping bins. Pad for storage of bulk sample rounds in centre foreground, bunkhouse, and kitchen left middle-ground, and core storage left background.
3	Jarvis Clark Jumbo fitted with Tamrock C-40 machines at the adit face drilling a blast hole.
4	Cameron McCutcheon Drilling electric-hydraulic "Superdrill" set up to drill a +35° hole.
5	Water issuing from relief holes in adit face in country rocks at 335 m, ¼ of the distance into the deposit.
6	Dumping bins for segregating muck rounds for the bulk sampling program.
7	Individual muck rounds segregated and stored for bulk sampling on the pad constructed from adit muck.
8	Bulk Sampling Plant: Primary crusher in right background, secondary crusher and screening plant at left, with conveyor to sampling tower in centre.

Photo No.	Description
9	Fotobor Surveying a down hole. Surveyor, on right, is assisted in lowering Fotobor instrument into the drill hole by driller on left.
10	Drill core from DDH 81-63 from photographic inventory of drill core. Core is mostly of High Grade Dyke with only a little of the Silicified Schist in the upper part of the hole at the top of the picture. The entire length (from 85.5 to 114.9 m) runs 1.12% MoS_2 with 12.2 m of the High Grade Dyke (from 99.0 - 111.2 m) assaying 2.28% MoS_2.
11	Spawning Kokanee trout in Wilkie Creek in the Trout Lake valley on east side of property.
12	Hand specimen of the well veined and mineralized Silicified Schist. Note the blue grey molybdenite occurring as rich aggregates and as narrow trains at the core of quartz veins.
13	High Grade Dyke: NW wall of the crosscut at 1,503 m showing the High Grade Dyke. Note vertical ribbony quartz veins. A parallel banded texture in the sulphides in the High Grade Dyke is also present along with the vertical contact with the Silicified Schist, showing to the left of the photograph.
14	Intense Quartz Stockwork in Silicified Schist typical of many locations within the core of the deposit.

Photo No.	Description
15	Inclusion of Silicified Schist in Granodiorite in the SW wall of the No. 3 Drift at 40 m from junction. Note that some veins stop at the intrusive walls while other later veins cut through the intrusive, the schist and the earlier veins. The small black X's indicate veins which were measured as part of the vein orientation survey.
16	Quartz Diorite (dark) cutting Granodiorite (light) from the east wall in the No. 4 Drift at 26 m. Note the quartz diorite also cuts the large quartz veins and is cut by the smaller quartz vein.

APPENDICES

APPENDIX I

TROUT LAKE PROJECT

by: K. F. Dahlke February 12, 1982

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TROUT LAKE PROJECT

by: K. F. Dahlke

INTRODUCTION

Newmont Mines Limited and Esso Minerals Canada Ltd., as joint venture partners, undertook a comprehensive underground exploration and bulk sampling program at their Trout Lake molybdenum prospect. The property is located approximately 4 km. west of Trout Lake, a small village in south eastern British Columbia within the Selkirk mountain range.

The village of Trout Lake has a population of approximately 50 people, but had very little to offer in support of the program. It has a small one classroom school, a gas station with corner store, a fishing resort and an old hotel, which was not catering to the public. Trout Lake has telephone service, but has no power or t.v. signal. The nearest communities are Revelstoke, 80 km. to the north by road and a 20 minute ferry ride across the Arrow Lake, and Nakusp, 80 km. to the south by road. The access roads to Revelstoke and Nakusp are paved except for the last 12 km. before entering Trout Lake, which is gravel. Access from Trout Lake to the property is by logging road. The main access roads were well maintained and kept free of snow during the winter months. The nearest rail road connection is at Revelstoke.

SCOPE OF THE PROJECT

1. Site Preparation

   (a) Clear and level camp and portal sites
   (b) Set up a 40 man camp and kitchen facilities
   (c) Excavate a dump site
   (d) Build reinforced dump bins
   (e) Erect a maintenance shop and plant buildings
   (f) Lay outside track to dump and shop
   (g) Cover outside track and dump by snow shed

2. Underground Excavations

   (a) Drive a 12 ft. wide by 15 ft. high adit
   (b) Drive four cross outs 10 ft. wide by 12 ft. high
   (c) Excavate diamond drill stations
   (d) Store rounds from mineralized zone for bulk sampling.

3. Bulk Sample Mineralized Rock

   (a) Set up a sampling tower and crushing plant
   (b) Sample mineralized drift rounds

4. Diamond Drilling

   (a) Pilot Hole Drilling
   (b) Deep Exploration Drilling
   (c) Definition Drilling

5. Conduct Environmental Studies

   (a) Fish and wildlife
   (b) Water quality
   (c) Meteorological

PROJECT PREPARATIONS

The portal and camp sites, located at the 3,150 ft. elevation, were selected in early July 1979. The deciding factors in site selection were:
(a) The overall length of the adit
(b) Room for mill, plant and permanent camp sites
(c) Room for tailings

Clearing and levelling of the sites commenced on July 15, 1979. The contract to supply housing and catering for the employees was awarded to Cal-Van Canus Camp Services Ltd. Cal-Van Crews commenced setting up a mobile type trailer camp and kitchen facilities to house 40 people on August 17, 1979 and completed the same on August 28, 1979. A septic field for the camp, consisting of two 2,000 gal. fiberglass septic tanks, and 2,000 ft. of 3 inch perforated plastic pipe, layed on 8 inches of drain rock in 3 ft. deep trenches was also installed during this time.

Camp water was obtained by gravity from two small creeks, (Eckerman and Eckhart) under temporary water rights licence, renewable at six month intervals. Water was collected by 2 inch plastic pipes, and fed into a 6,000 gal. holding tank set up approximately 70 ft. above camp elevation, and from there fed by gravity through a 1½ inch plastic pipe to camp. Both creeks, however, dried up during the winter months, at which time water was obtained from the adit by pumping to the holding tank.

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Camp power was supplied by two 30 kw. KAO Diesel powered generators, rented through Cal-Van Canus.

The underground excavation and bulk sampling program was awarded to Canadian Mine Services Ltd. on August 7, 1979. Crews commenced with the installation of the surface track, dump, concrete bins, snow shed and plant buildings on Sept. 8, 1979. A 40 ft. wide by 100 ft. long ATCO fold-a-way building set up on concrete foundation was selected for the maintenance shop. This building proved to be quite adequate for the entire project. The power plants and air compressors were housed in a 20 ft. wide by 60 ft. long wooden structure. All surface facilities with the exception of the snow shed were completed on November 10, 1979. The snow shed, however, was not completed until January 25, 1980.

Equipment supplied and used by the Mining Contractor

2 - 250 K.W. CAT Generator
1 - 150 K.W. CAT Generator
3 - 600 C.F.M. Gardner Denver Compressors
1 - 75 H.P. Joy Air Fan
2 - 15 Ton Battery loci
20 - 8 ton side dump cars
1 - Car Dumper
1 - 3 Boom Jarvis Clark Jumbo (fitted with Tamrock C-40 machines)
1 - 40 Eimco mucking machine
1 - Scissor lift service car
1 - flat car
2 - 30 H.P. flygt pumps
2 - 15 H.P. drill water pumps
1 - 400 amp. welder
1 - D.7 CAT
1 - 966 CAT loader
1 - 12-E CAT grader
1 - Ambulance

Equipment supplied and owned by the Joint Venture Partners

1 - Sampling Tower
1 - Laser Instrument (leased)
1 - Masonry saw
1 - 3 H.P. water pump
1 - Lynx 2000T snow mobile
1 - Weather Station Equipment
1 - Water Flow Meter

1 - 450 K.V.A. Transformer (substation)
1 - 500 K.V.A. Dry Transformer
1 - 75 K.V.A. Transformer
1 - High Voltage Transformer
1 - 5 K.V. Load Breaker Panel
Switch Gear and Cable

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UNDERGROUND EXCAVATIONS

All underground and related support work with the exception of engineering and geology was contracted out to Canadian Mine Services. The underground excavations consisted of a 12 ft. wide by 15 ft. high adit, to a distance of 4,157 ft. and then reduced to 10 ft. x 12 ft. high in the mineralized zone and extended a further 930'. Four cross-cuts, 10 ft. wide x 12 ft. high were driven to facilitate diamond drill station cut outs. Work on underground excavation commenced on September 26, 1979, and was completed April 26, 1981. Footage excavated was Adit 5,087 ft., cross-cuts 1,489 ft., diamond drill stations slash equivalent 260 ft. (cu ft. 28,749.73).

All underground work was done on a three shift per day, and seven days per week basis. Crews consisted of one supervisor, three miners, and one mechanic per shift.

Engineering, a Newmont Mines Limited responsibility, was provided by one surveyor and one helper. Line and grade were monitored by laser in the adit, and grade chains in all cross-cuts.

The scheduled advance of 25 ft. per day never materialized due to the adverse conditions encountered over the first 4,300 ft. in the adit. Badly fractured ground and inflow of water resulted in stuck drill steel, and caved blast holes, causing considerable time loss in the loading and blasting operations. Water relief holes had to be drilled in many instances to permit the loading of the blast holes against high water pressure. Working conditions were poor, to say the least, due to the extreme wet conditions with water pouring constantly from the back and walls of the adit. During the driving of the adit, the following is an indication of performance.

Best 2 week period	323'	= 21.53' per day
Best month	561'	= 18.10' per day
Poorest 2 week period	22'	= 1.47' per day
Poorest month	138'	= 4.76' per day
Total adit & cross cuts	6,576'	= 11.99' per day
From start of contract mining		= 12.67' per day
Considered average for normal ground		= 16' per day

Once the mineralized zone was reached and the "Z" fault was passed, performance improved to an average advance of 15.79' per day. The water flow from the adit averaged 2,000 plus I. G.P.M. during the greater part of the project, and peaked at 6,000 G.P.M. The major water inflow areas are fault zones located at the 2,254 - 3,086 - 3,350 - 3,446 - 3,518 and 3,567 ft. mark. 3½ Inches by 50 ft. long drain holes were drilled ahead of the face to help alleviating the water problem. The bulk of the water, however, dried up gradually with the advance of the face.

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The water flow from the adit was at 1,125 I. G.P.M. at the end of the project, with an estimated 800 G.P.M. coming from the fault zones, and the remainder from various diamond drill holes.

Major ground support consisting of 6 inch steel arch sets, heavy timber cribbing, and rock locked spiling, consisting of old drill steel, or 1½ inch x 12 ft. long re-bar installed prior to the basting of a round, was required through all major water bearing fault zones. Twenty six days were required to cross the fault zone at the 3,086 ft. mark. An inflow of approximately 1,000 G.P.M. of water from the face washed out approximately 600 tons of fault material, resulting in a cavity 15 ft. wide by 40 ft. high and 35 ft. long. Fifteen steel sets, plus 150 only, 16 ft. long 8" x 8" timbers were installed crossing this zone.

Steel and bit cost for the project was 31.3¢ per foot advance. This exceptional high cost reflects the poor ground encountered in the adit prior to the mineralized zone. The ground through the mineralized zone was however mostly competent and dry, requiring minimum ground support. The contractor chose Fagersta Drill steel, and Hard Metals drill bits for the project, after conducting extensive testing of other brands.

Explosives used were Forcite 75%, Xactex 75% and long delay electric blasting caps. Forcite 75% was used for its water resisting characteristics and Xactex 75% in back and side holes for better ground control. An average of 37# of powder was used for every foot of advance in the 12' x 15' heading which is equal to $53.50 per foot as opposed to the bid price of $22.55 per foot. In the 10' x 12' headings, 25# of powder was used per foot of advance at a cost of $42.50 per foot as opposed to the bid price of $18.02 per foot.

Services installed were 60lb. rail, 6" x 8" x 6'6" treated ties, 6" victaulic air pipe, 2" victaulic water pipe, and 42" nylon ventilation ducting. All services were left in place at the end of the project.

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GROUND SUPPORT

Ground support work was carried out as required, material used was 6 inch steel sets, 8' rock bolts, 12" wide by 3'6" steel straps, wire mesh and 1" rebar spiling.

Steel sets in conjunction with heavy timber cribbing and spiling were installed at all major fault zones. Rock bolts and straps were installed as required and the wire mesh was mainly used in the diamond drill stations.

Steel set locations are:

18 sets at the portal
6 sets at 1,154 ft.
12 sets at 2,083 ft.
7 sets at 2,254 ft.
5 sets at 2,562 ft.
4 sets at 2,850 ft.
4 sets at 3,009 ft.
15 sets at 3,086 ft.
6 sets at 3,350 ft.
3 sets at 3,461 ft.
3 sets at 3,518 ft.
6 sets at 3,567 ft.
2 sets at 4,448 ft.

The Z-fault was intersected at the 4,303 ft. mark. It was relatively dry, and supported by spiling and four square sets, consisting of 10' x 10" posts and 8" I-beam caps. Square sets were used in preference to steel arched sets in order to provide more room for ventilation ducting and other services.

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BULK SAMPLING

Bulk Sampling commenced on June 22, 1981, and was completed on July 12, 1981.

Equipment used was one jaw crusher, one roller crusher, one sample tower, one 966 loader and one Wagner S.T.-2. All equipment, with the exception of the sampling tower, was rented. The sampling tower is owned by the Joint Venture partners. It was designed by Canadian Mine Services Ltd., and constructed by Minpro of Toronto, Ontario.

A total of 227 samples, each consisting of approximately 25 tons were processed from previously stockpiled drift rounds. Considerable time was lost on the sample tower due to under powered electric motors which burned out, and poorly designed splitter chutes which constantly plugged.

Attached (page 10) is a flow sheet for the program. All reference samples are stored in 25 gal. drums, identified by drift round number under the snow shed at the mine site.

The sampling crew consisted of one superintendent, one crusher foreman, two equipment operators, one crusher operator four labourers and one engineer.

DIAMOND DRILLING

Cameron McCutcheon Drilling Ltd., a subsidiary of Canadian Mine Services Ltd., were awarded the Diamond Drill Contract. The Contractor utilized three "superdrills," each powered by a 100 H.P. electric motor for the program. Power for the three drills was provided by two 250 K.W. Diesel generators operated in parallel. Power was generated at 460 volt, then stepped up to 4,160 volts and stepped down again to 480 volts at the drills. Drill water was first obtained from the Mining Contractors system, and later from diamond drill holes drilled on section #6 in the #1 Drift West, and section #10 in the #2 Drift South.

Drilling commenced on November 22, 1980, and was completed on November 5, 1981. Start-up date for the three machines was #1 - November 22, 1980, 1980, #2 - January 14, 1981, and #3 - May 8, 1981. A total of 72,570' were drilled comprising of approximately 60% B.Q. core size and 40% N.Q. core size. Hole size was mainly determined by the length of the hole, i.e. all long holes were collared N.Q. Then later reduced to B.Q., also the majority of holes sampled by splitting the drill core were drilled N.Q. The diamond drill bit performance was B.Q. = 113 ft. per bit, or $3.53 ft., and N.Q. = 133 ft. per bit and $4.22 ft.

The majority of B.Q. drilling was achieved with a huddy orange bit (27.270'). The majority of N.Q. drilling was achieved using a huddy yellow bit (11,694') and a huddy orange bit (8,838').

Drilling was carried out in the #1 Dr. W., section 6, #2 Dr. S., sections 8 and 10, #3 Dr. W. sections 4, 5, 6 and 7, #4 Dr. S., sections 8, 9, 10 and 12 and at the adit face. Four pilot holes were drilled for comparison of core samples against drift rounds bulk samples, two in the adit, at the 4,339 ft. and 4,716 ft. mark respectively, and one each in the #3 Dr. W. and #4 Dr. W.

Average performance over the drilling program was 44.85' per drill shift or 102.36' per machine day. The best month produced 55.56' per drill shift or 112.73' per machine day. Diamond drill performance increased every month over the span of the project beginning at 61.11' per machine day through to 102.36' per machine day.

The bulk of the core was split by diamond saw. Core not required for assaying is stored on site.

Down the hole surveying was done by acid dip test on short holes and fotobar instrument on plus 800 ft. holes.

High water pressure was encountered in the following diamond drill holes:

80 - 6	1 - Dr. W.	Section 6
81 - 1	1 - Dr. W.	" 6
81 - 2	2 - Dr. S.	" 10
81 - 3	1 - Dr. W.	" 6
81 - 7	2 - Dr. S.	" 10
81 - 8	1 - Dr. W.	" 6
81 - 9	1 - Dr. W.	" 6
81 - 10	2 - Dr. S.	" 10
81 - 12	2 - Dr. S.	" 8
81 - 13	2 - Dr. S.	" 10
81 - 72	4 - Dr. S.	" 12
81 - 73	4 - Dr. S.	" 12

...../9

ENVIRONMENTAL

Envirocon Ltd. of Vancouver was engaged in late August, 1979, to perform an environmental overview study. Envirocon identified some areas of concern, and recommended a program for a preliminary environmental study in preparation for the Stage I Report required under provincial regulations. The main areas of concern were fish and wildlife, water quality and meteorology.

Klohn Leonoff Consultants Ltd. of Riohmond were engaged to perform a preliminary study on water supply alternatives, and issued a draft report on January 11, 1980.

The contract for the environmental impact study was let to Beak Consultants Ltd. of Richmond. Beak began with the field work in April 1980, and continued with it throughout the project.

Water quality sampling began in October 1978, on a quarterly basis, and is still continuing. Sample stations were increased from the original 8 to 13 in 1980. Water flowing from the adit was sampled on a monthly basis during operations due to some concern of possible contamination.

A water flow recorder was installed in Wilkie Creek at the Oliver Road Bridge. A small meteorological station consisting of a rain gauge, evaporating pan, thermograph and wind speed direction meter were installed at the project site. All meteorological equipment was removed and stored at the completion of the project.

****

February 12, 1982.

KFD/as

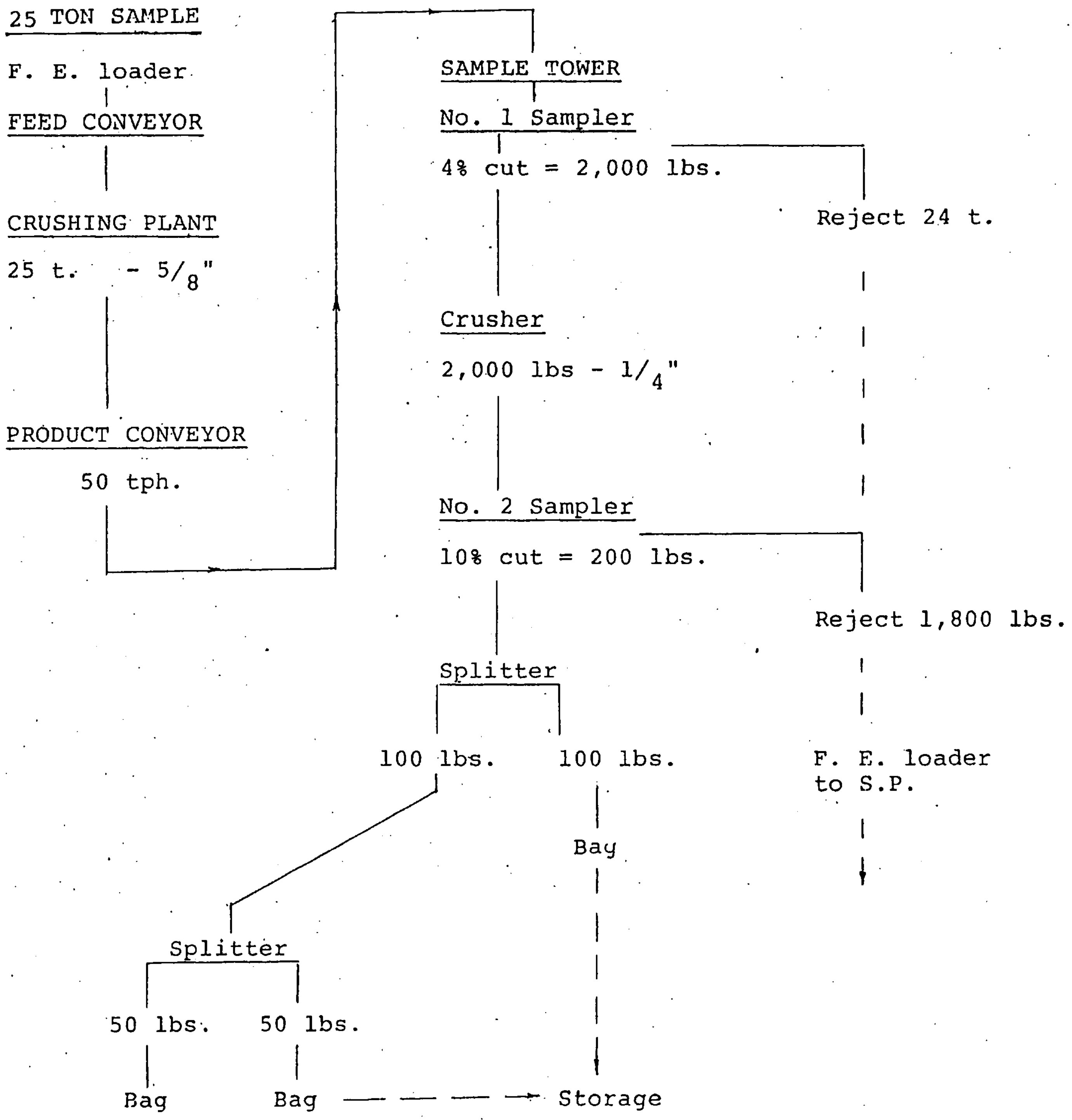
ebr.1981
F.T.H.

NEWMONT MINES LTD.

TROUT LAKE PROJECT

BULK SAMPLING PROGRAM

FLOWSHEET



APPENDIX II

To: J. Harvey Parliament February 24, 1982

From: F. T. Hancock

Subject: Trout Lake Project, Bulk Sampling Program

The underground program of development, diamond drilling, and bulk sampling at Trout Lake was started in July 1979 and completed in November 1981. An adit was driven on the 960 m. (3,150 ft.) level to the mineralized zone, where a cross-cut and four lateral drifts provided stations for diamond drilling. Pilot diamond drill holes were completed ahead of development in the cross-cut and the two drifts within the deposit to provide core samples for comparison with bulk samples. Material excavated from these headings was segregated and stored by round for future bulk sampling and metallurgical testing.

A bulk sampling plant was erected on-site in mid 1981. About one quarter of each round (25 tons) was reduced to sample size, following the procedures described in Appendix I. The headings, pilot DDH's, bulk and core sample grades are shown on the 960 level Bulk Sample Plan.

Comparison of Bulk Sample and Pilot DDH Core Grades.

For comparison, bulk and DDH grade data is grouped by drill hole, heading and combined total. "Arithmetic data" is based on actual grades, while "Ln data" is based on natural logarithms of grades, a lognormal distribution of grades being apparent.

Unweighted grade data is summarized in Table 1, weighted grade data is practically the same and is summarized in Table 2.

The combined total of bulk grades is 8% higher than DDH grades in arithmetic average and about 1% higher in average derived from Ln transposition.

128 combined samples, from Table 2		Bulk	DDH
Arithmetic average grade	% Mo. S_2	0.222	0.205
Ln average grade		0.217	0.214

The largest continuous group of grade data is in the cross-cut where there is a wider spread between averages, the bulk grade is 12% higher than DDH grades in arithmetic average and 9% higher in Ln average.

77 cross-cut samples, from Table 2	Bulk	DDH
Arithmetic average grade, % Mo. S_2	0.280	0.251
Ln average grade	0.276	0.254

In the smaller groups of grade data, bulk grades vary from about 85 to 137 percent of DDH grades in arithmetic and Ln averages.

Compared round by round, grades vary from coincident to a wide spread, particularly in higher grade dikes.
The Bulk : DDH sample size ratio is about 12,000 : 1.

Correlation between bulk and DDH data varies widely between groups and is relatively poor; log transposed data produces better correlation for the combined cross-cut and total groups. Correlation is termed better as the coefficient value approaches unity.

Bulk variances are expectedly lower than DDH variances with the exception of the cross-cut part 2, where higher bulk grades are not reflected in DDH 81-5. The resulting higher bulk variance carries into the arithmetic combined total. Variances derived from log transposed data are smoothed, bulk being lower than DDH in the combined total group.

SUMMARY

Compared in small groups, DDH grades do not provide a reliable estimate of bulk grades, they may be higher or lower. For the combined total of samples, DDH grades are 8% lower than bulk grades in arithmetic average, providing a conservative estimator. The simple lognormal model used gives better correlation and a close estimation of bulk grades. It is inferred that grade estimation from DDH samples requires a large number of samples, together with a model of their grade distribution, and that the spacial distribution of grades within the average may not be accurately predicted.

Appendices

I. Sampling Procedures
II. Data Processing, Bulk check samples, Grade distribution.

Drawings

Headings, pilot diamond drill holes and grades are shown on the 960 level Bulk Sample Program Plan.

F. T. Hancock,
Project Engineer.

TROUT LAKE PROJECT

Table 1.

BULK & PILOT DDH GRADE COMPARISON

UNWEIGHTED GRADES		X-Cut 1	X-Cut 2	X-Cut 1+2	3 Dr.	4 Dr.	Total
	DDH	#80-5	#81-5		#81-4	#81-6	
No. of Samples		38	39	77	25	26	128
ARITHMETIC DATA							
Bulk grade average	% MoS_2	0.233	0.324	0.279	0.118	0.149	0.221
DDH grade average	% MoS_2	0.265	0.240	0.252	0.091	0.173	0.203
Correlation coefficient		0.571	0.395	0.361	0.358	0.747	0.466
Bulk Variance		0.007	0.086	0.048	0.003	0.004	0.036
DDH Variance		0.029	0.032	0.031	0.003	0.011	0.025
Ln DATA							
Bulk grade average	% MoS_2	0.234	0.324	0.276	0.118	0.149	0.216
DDH grade average	% MoS_2	0.272	0.240	0.255	0.101	0.175	0.213
Correlation coefficient		0.564	0.651	0.569	0.485	0.713	0.654
Bulk variance		0.008	0.091	0.035	0.003	0.004	0.022
DDH variance		0.046	0.029	0.036	0.010	0.012	0.039

TROUT LAKE PROJECT

BULK & PILOT DDH GRADE COMPARISON Table 2.

GRADE X LENGTH PRODUCTS		X-Cut 1	X-Cut 2	X-Cut 1+2	3 Dr.	4 Dr.	Total
	DDH	#80-5	#81-5		#81-4	#81-6	
No. of Samples		38	39	77	25	26	128
Round length average (ft)		9.921	9.513	9.714	9.600	9.423	9.633
ARITHMETIC DATA							
Bulk grade average	% MoS_2	0.230	0.330	0.280	0.118	0.149	0.222
DDH grade average	% MoS_2	0.262	0.241	0.251	0.093	0.174	0.205
Product correlation coeff.		0.565	0.420	0.367	0.406	0.749	0.468
Bulk product variance		0.664	9.199	5.108	0.243	0.368	3.689
DDH product variance		2.729	2.901	2.803	0.351	0.986	2.324
Ln DATA							
Bulk grade average	% MoS_2	0.231	0.329	0.276	0.118	0.149	0.217
DDH grade average	% MoS_2	0.269	0.241	0.254	0.103	0.176	0.214
Product correlation coeff.		0.545	0.668	0.573	0.541	0.728	0.665
Bulk product variance		0.714	9.092	3.427	0.245	0.358	2.167
DDH product variance		4.229	2.688	3.347	1.021	1.171	3.740

Appendix 1.

Sampling Procedures

Bulk Samples

Bulk sampling was planned for the late spring of 1980 to avoid problems associated with snow and freezing temperatures. Fractured ground and large inflows of water delayed the advance of the adit which entered the mineralized zone in November 1980. Material was then stockpiled for processing during the summer of 1981.

Rounds were segregated in concrete bins at the portal dump. Each round was mixed and stored in a prepared stockpile area, a lot number identified the round's location on the underground plan.

Bulk samples were cut from each pile using a Cat 966 Front end loader, cuts were taken a few inches above the base of the pile crowding to the centre, removing one quarter or approximately 25 tons.

Each sample was reduced to minus 6 inch in a Pioneer 28x36 crusher to remove oversize material, then reduced to minus 5/8 inch in a Pioneer 45V gravel crusher and conveyed to the sampling tower.

In the tower an in-line sampler cut about 3,000 pounds which was reduced to minus ¼ inch in a roll crusher. A Vezin type sampler cut 10 percent, which was then split in a second Vezin type sampler. One 150 pound split was collected in a 25 gallon drum, the other half split again in a ½ inch riffle. The final splits were weighed and bagged, one split being placed in the 25 gallon drum, the other placed in a 45 gallon drum for shipment. The 25 gallon drums were sealed for on-site storage. The final splits were 50 percent larger than planned due to the in-line cutter characteristics. The design flow sheet is attached. Figure 1.

Rejects were removed to storage by a Wagner ST 2B scoop-tram, assisted by the Front end loader.

OPERATING SUMMARY

Start	22 June 1981
Finish	12 July 1981

No. of rounds sampled	189
No. of rounds check sampled	24
No. of check samples	38
Total no. of samples	227

Nominal weight of sample from round	25	tons
1st cut	3,000	pounds
2nd cut	300	
1st split	150	
2nd split	75	

Size of 1st cut	100% minus 5/8 inch
	75% minus 1/4 inch
Size of 2nd cut	90% minus 1/4 inch

Average running time per sample, including clean-up	27 minutes

Total time (10 hours per day)		209.5 hours
Operating time		103.5
Tower downtime	electrical	77
	mechanical	29
	total	106

Tower availability	49.4%
Crushing plant availability	99 %

Labour:	Supervision and general	3
	Crusher	2
	Tower	3
	Drum Handling	1
	Mobile Equipment	2
	Total	11

Febr. 1981

F.T.H.

NEWMONT MINES LTD.

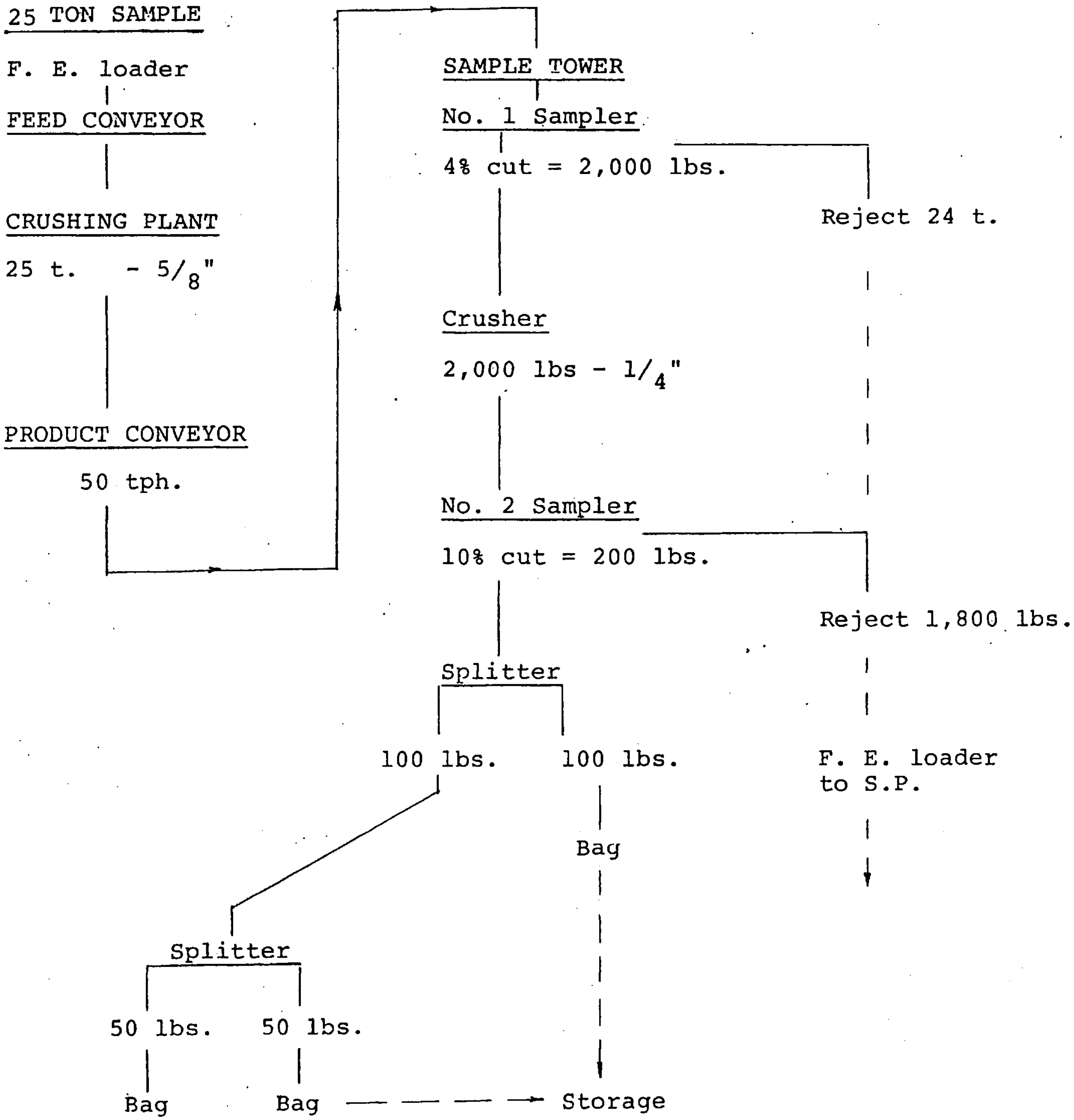
TROUT LAKE PROJECT

BULK SAMPLING PROGRAM

DESIGN

FLOWSHEET

Figure 1.



CORE SAMPLES

Pilot diamond drill holes size NQ (48 mm., 1.875 inch core diameter) were drilled in advance of development through the length of the cross-cut and predictably mineralized portions of two lateral drifts, using a stabilizing core barrel assembly. The pilot holes stayed within the perimeter of the headings for a distance of 60 to 90 m. (200 to 300 ft).

DDH core was sampled by splitting, sample lengths being 1.5 to 2 m. (4.9 or 6.6 ft.) or as determined by marked ohanges in estimated grade. Samples were bagged and sealed in 12½ gallon pails for shipment. The other half of the core was retained in the box for on site storage.

ASSAYING

Final sample preparation and assaying of bulk and core samples was carried out by Newmont Exploration Limited at Danbury, Connecticut.

NEWMONT MINES LIMITED
SIMILKAMEEN DIVISION

August 25, 1982.

TO J. H. Parliament

FROM W. G. Martin

SUBJECT Trout Lake Property

As discussed with you earlier, I have taken a preliminary look at the Trout Lake project.

Without proper sections and plans of the mineralized zones, and without reserve calculations at various cut-off grades, only a preliminary estimate of cash flows can be made.

Using R. S. Mattson's capital expenditure estimates for a 3000 ton per day and a 6000 ton per day milling rate, I have estimated that the capital expenditure for a 1500 ton per day mill would cost $67,500,000.

I obtained present costs from Boss Mountain which is milling at 1500 tons per day. By using these costs as a basis, and adjusting the mining costs as outlined by Trevor Hancock in his report of June 29, 1982, I have arrived at an estimated operating cost of $29.85 per ton milled. I also adjusted R. S. Mattson's costs which were related to a 3000 ton per day mill, to reflect the new mining costs and town site costs which were not included.

I calculated the price of Mo required for both a 3000 ton per day mill and a 1500 ton per day mill assuming that the capital would be returned in 3 years and 5 years respectively. In both cases, the price of Mo would have to be approximately $15.00 per pound.

Obviously there is insufficient data available at the present time to properly assess this property. However, it is my personal opinion, based on the data that is presently available, that unless the price of Mo increases to a $15.00 per pound range, at present day costs this property would not be a viable operation.

WG Martin

W. G. Martin
Vice President and General Manager

WGM/dg
attach.

August 25, 1982.

BOSS MOUNTAIN DATA

U/G Mine started in 1963

Head Grade: .30% Mo (0.50% MoS_2) at start of operations.

Present mill tonnage: 1500 - 1700 TPD @ .19% Mo (0.32% MoS_2) in mid 1982.

Costs - July, 1982:	$/Ton Milled
Administration & General	$ 4.25
Plant - surface	2.40
Camp & Townsite	1.90
U/G Mining	6.75
U/G exploration	.20
Haulage & hoisting	2.00
Milling	5.30
	$ 22.80

Total people - 180 (head office not included)

Sales costs not included - approximately .10 to .15/lb. Mo

Mining method - long hole stoping

TROUT LAKE

(Mining at 1,500 TPD)

(Using Boss Mountain data & assuming Hancock's mining costs)

1. **Ore Reserves:** 8,189,000 tonnes @ .328 MoS_2

2. **Life Expectancy:** 15 years

3. **Capital Expenditure:** $67,500,000

4. **Operating Costs:**

	1500 tpcd - $/ton milled
Mining	$15.90
Milling	5.30
Plant & Surf. Mtce.	2.40
General & Administration	4.25
Handling & Freight	.10
Camp & Townsite	1.90
	$29.85/ton milled

5. **Milling:** 547,500 tons/year @ .328% MoS_2

   Therefore: .328 MoS_2 x 20 x .92 Rec. x .60 @ 3.15

   = 3.62 lbs. Mo/ton milled. $ 6,243,190

6. **Capital Write Off:** $\frac{67,500,000}{8,189,000}$ = $8.24/ton milled

7. **Calculate Price of Mo Required:**

   To return capital in 5 years - $\frac{\$67,500,000}{5}$ = $13,500,000/yr

   or $\frac{13,500,000}{547,500}$ = $24.66/ton milled.

   Income = operating costs + return of capital
   ∴ 3.62x = 29.85 + 24.66
   = 54.51
   x = 15.06/lb.

	$/ton milled
Income 3.62 x 15.06/lb.	$ 54.52
Operating costs	29.85
Operating Profit	$ 24.67
Capital Write-off	8.24
Net Profit (before taxes)	$ 16.43

Cash flow/yr 547,500 x 24.67 =	$ 13.51/yr.
Payback capital	5 years
Profit 547,500 x 16.43 x 15	134.93
Taxes (Estimated) 37%	49.92
Profit after taxes	85.01

August 25, 1982.

TROUT LAKE

(Mining at 3,000 TPD)

1. Ore Reserves: 8,189,000 tonnes @ .328 MoS_2 (Hancock's Report - June 29, 1982)

2. Life Expectancy: 7.5 years

3. Capital Expenditure:

   $90,000,000

4. Operating Costs:

	3000 tpcd - $/ton milled
Mining	$ 15.90
Milling	3.50
Plant & Surf. Mtce.	1.62
General & Administration	2.81
Conc. Handling & Freight	0.10
Camp & Townsite	1.90
	$ 25.83/ton milled

5. Milling: 1,095,000 tons/year @ .328% MoS_2

   Therefore: .328 MoS_2 x 20 x .92 Rec. x .60

   = 3.62 lbs. Mo/ton milled.

6. Capital Write Off: $\frac{\$90,000,000}{8,189,000}$ = $10.99/ton milled

7. Calculate Price of Mo Required:

   To return capital in 3 years - $\frac{\$90,000,000}{3}$ = $30,000,000/yr

   or $\frac{30,000,000}{1,095,000}$ = $27.40/ton milled.

   Income = operating costs + return of capital

   $\therefore$ 3.62x = 25.83 + 27.40

   = 53.23

   x = $14.70/lb.

	$/ton Milled
Income 3.62 x 14.70	$ 53.21
Operating Costs	25.83
Operating Profit	$ 27.38
Capital Write-off	10.99
Net Profit	$ 16.39
Cash flow/yr 1,095,000 x $27.38	$ 29.98
Payback capital	3.0 years
Profit 1,095,000 x$16.39 x 7.5	134.6
Taxes (Estimated) 37%	49.8
Profit after taxes	84.8

TROUT LAKE PROJECT

CAPITAL EXPENDITURES

SUMMARY

	Estimated Cost 1500 TPD	R.S. Mattson Costs Case 1 3000 TPD	Case 2 6000 TPD
U/G Exploration Program	$ 4,505,000	$ 4,505,000	$ 4,505,000
Mine Pre-Prod. Dev. & Equip.	14,600,000	18,860,000	24,400,000
Plant & Surface Facilities	27,700,000	37,238,000	50,039,000
Land Aquisition	1,000,000	1,000,000	1,000,000
Power Supply	4,500,000	6,000,000	8,000,000
Water Supply	700,000	1,000,000	1,500,000
Townsite & Camp	9,300,000	14,000,000	21,000,000
Inventory	2,300,000	3,000,000	4,000,000
Working Capital (3 months)	2,840,000	4,388,000	6,778,000
TOTAL $ CANADIAN	$ 67,500,000	$ 89,991,000	$ 121,222,000

site. We would recommend Area 2 to be the most suitable site for the following reasons:

General Topography	-	Area 1, narrow and elongated.
	-	Area 2, more compact.
Soils	-	Area 1, clays.
	-	Area 2, gravels.
Climate	-	Area 1, early afternoon sun cut-off (poor in winter).
	-	Area 2, generally better sun angles.
Sensitivities	-	Area 1, immediately above Wilkie Creek, known to be controlled fish spawning habitat.
	-	Area 2, no known sensitivities.
General	-	Area 2, is will serviced by the existing highway, has better view of the lake valley and is physically more closely aligned with the existing settlement.

We expect that with your development, the potential for the area in mining, logging and tourism and a community already established, that our solution of expanding the existing community into Area 2 to accommodate the required labour force for your proposed mine will ultimately be accepted by Government and that the Company should proceed accordingly. The implications of the recently tabled Procedures for Obtaining Approvals for Metal Mine Development, however, should be recognized. From our experience, considerable time and monies will be expended, on studies and reports, to satisfy Government on the recommended solution. It could be that since Crown Lands are involved that Government itself will conduct these studies concurrent with your investigations, and if the solution is accepted, will plan and develop the land subdivision which would leave the Company vulnerable, not only to direct costs, but indirect costs from lack of control of schedules.

We trust that the foregoing summary is satisfactory. We would

UNECON

be pleased to discuss it further with you and to be part of any ongoing studies that may be required.

Yours truly,

UNECON PROJECT CONSULTANTS

W.B. Scott

W.B.Scott

WBS/ek

cc: Mr. H. Parliament

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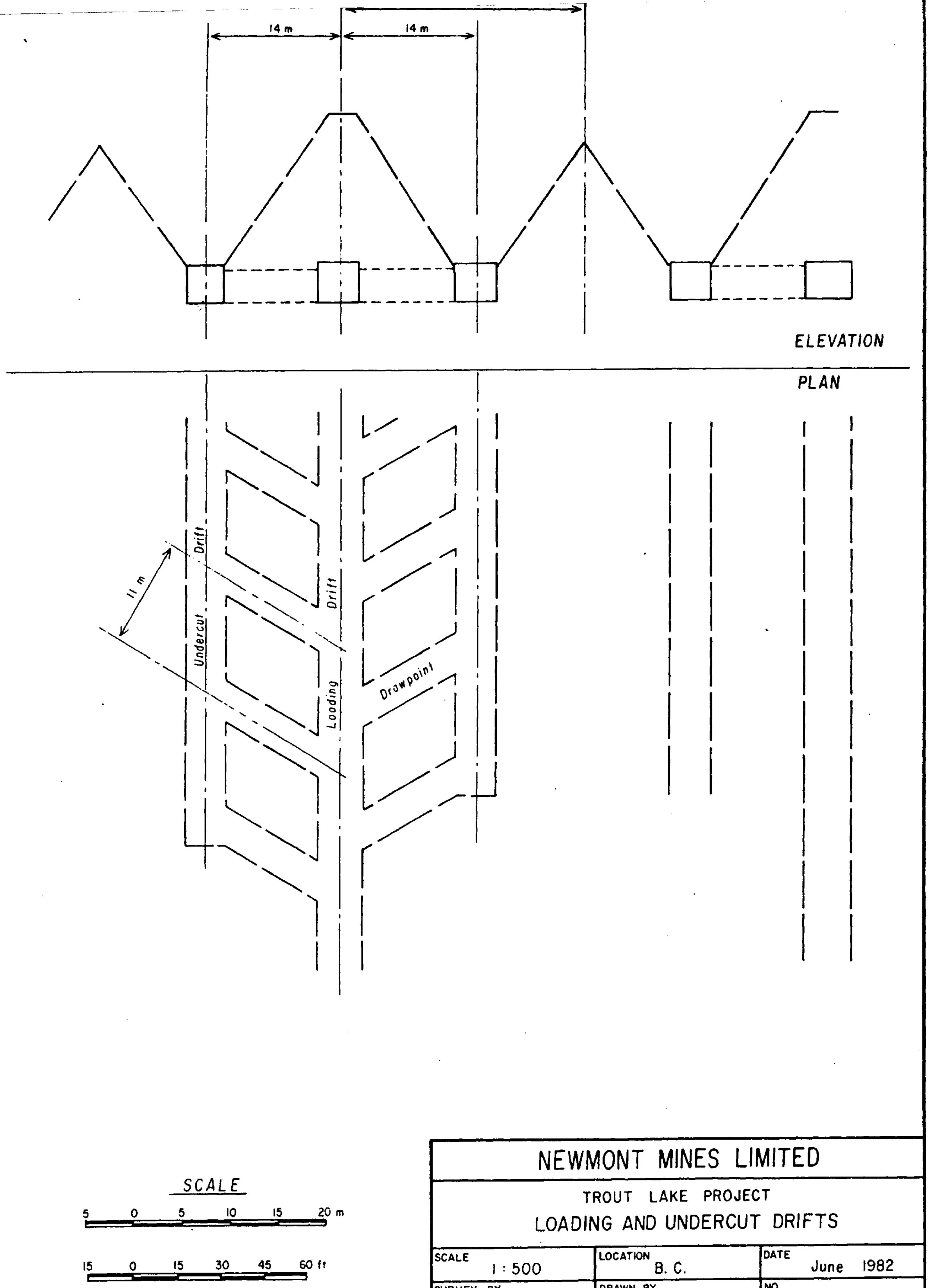
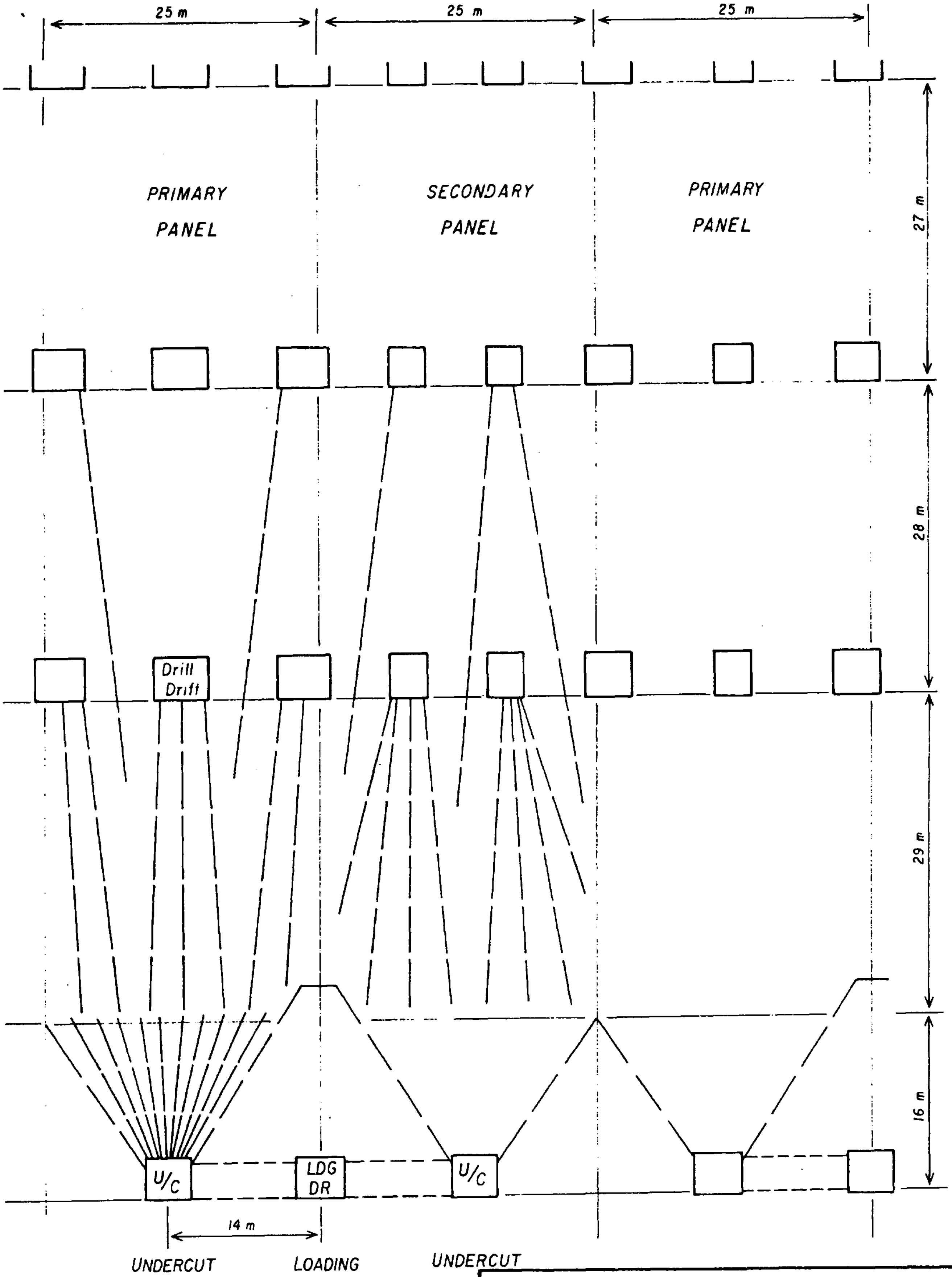
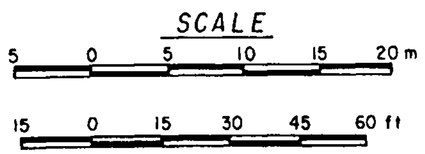
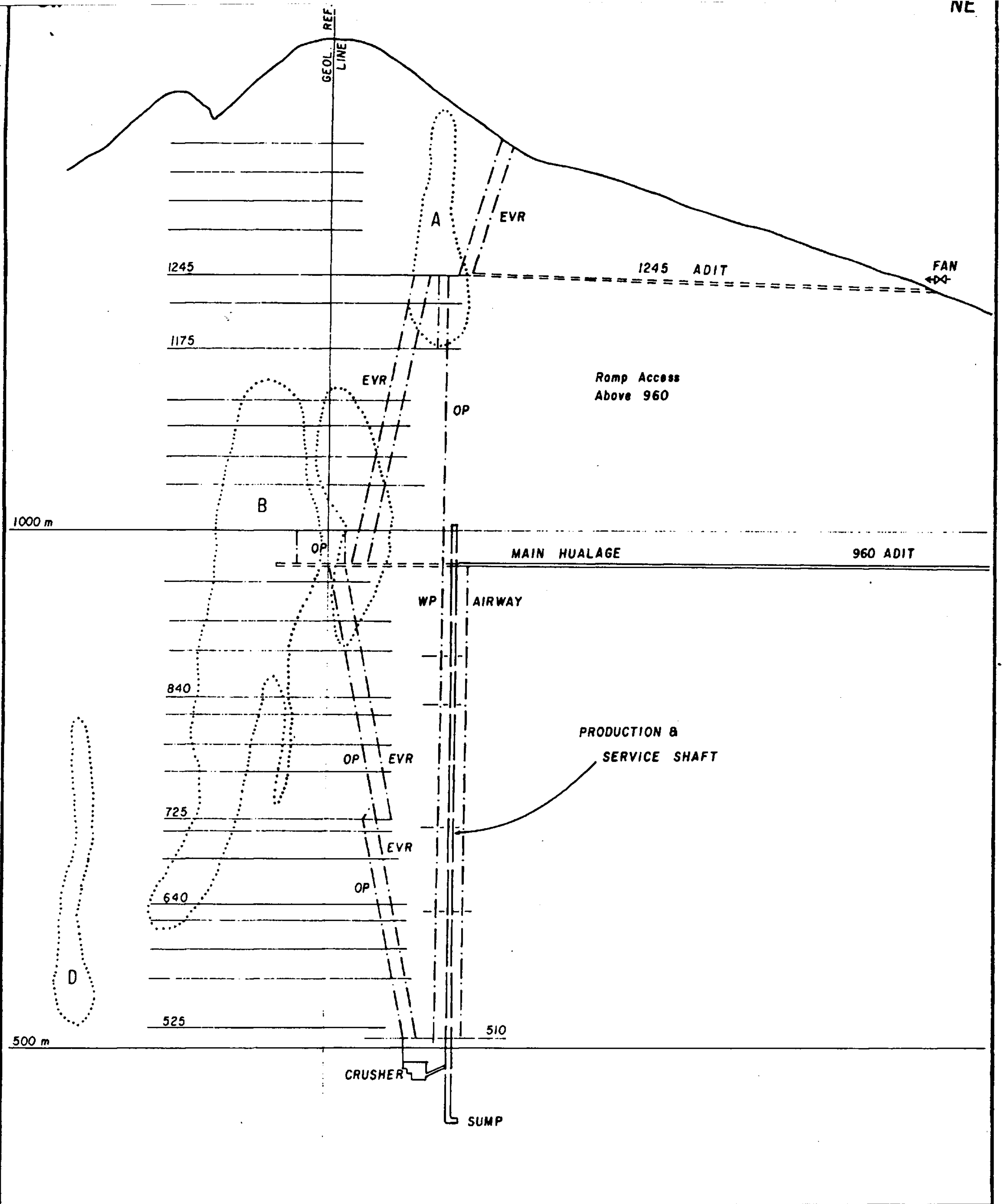


NEWMONT MINES LIMITED		
TROUT LAKE PROJECT LOADING AND UNDERCUT DRIFTS		
SCALE 1 : 500	LOCATION B. C.	DATE June 1982
SURVEY BY T. Hancock	DRAWN BY J. Goh	NO. Fig. 4





NEWMONT MINES LIMITED		
TROUT LAKE PROJECT		
STANDARD STOPE PANELS-X SECTION		
SCALE 1 : 500	LOCATION B. C.	DATE June 1982
SURVEY BY T. Hancock	DRAWN BY J. Goh	NO. Fig. 5



LEGEND

EVR	Exhaust vent raise
OP	Ore pass
WP	Waste pass

SCALE

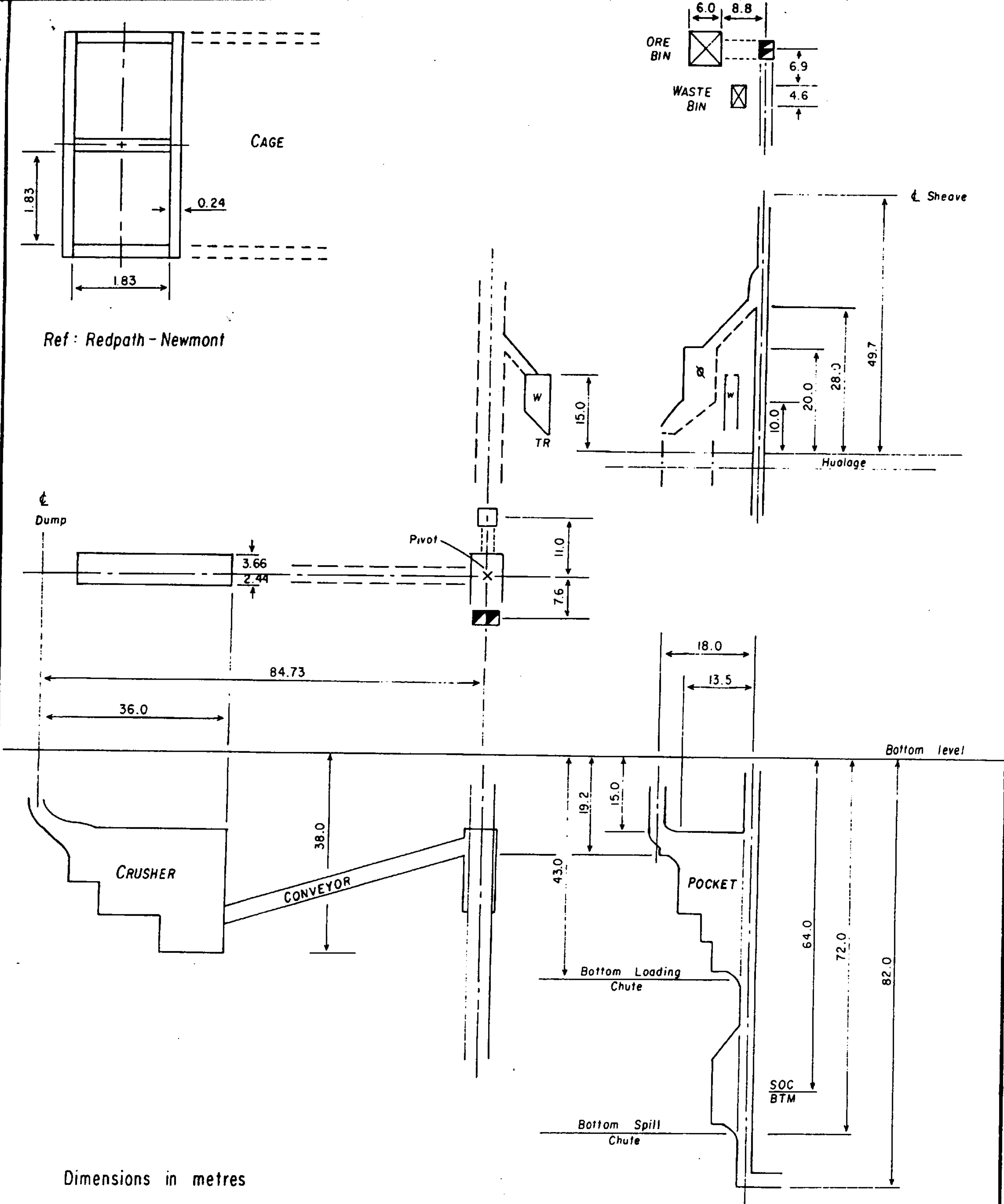
50 0 100 200 m

100 0 200 400 600 ft

NEWMONT MINES LIMITED

TROUT LAKE PROJECT
CONCEPTUAL SECTION

SCALE	LOCATION	DATE
1: 5000	B. C.	June 1982
SURVEY BY	DRAWN BY	NO.
T. Hancock	J. Goh	Fig. 6



SCALE

10 0 10 20 30 40 m

20 0 40 80 120 ft

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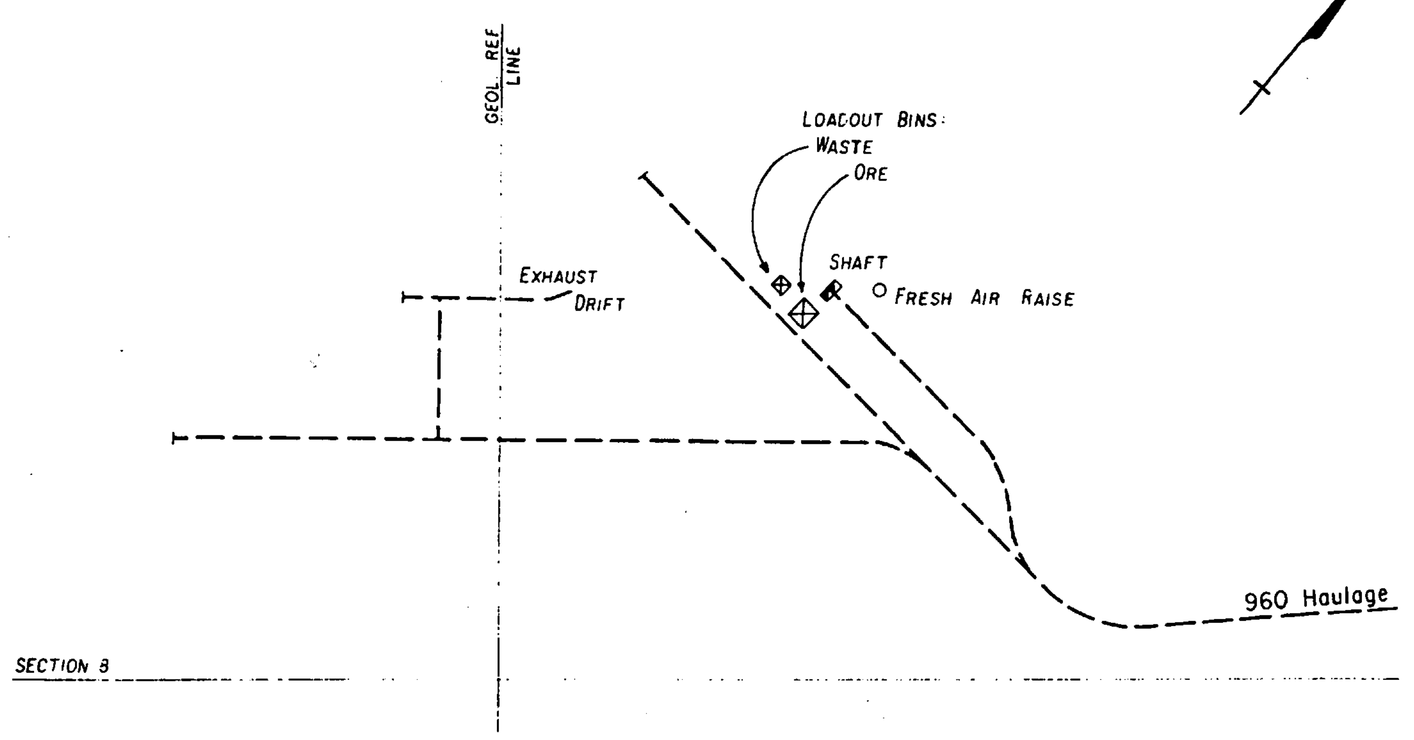
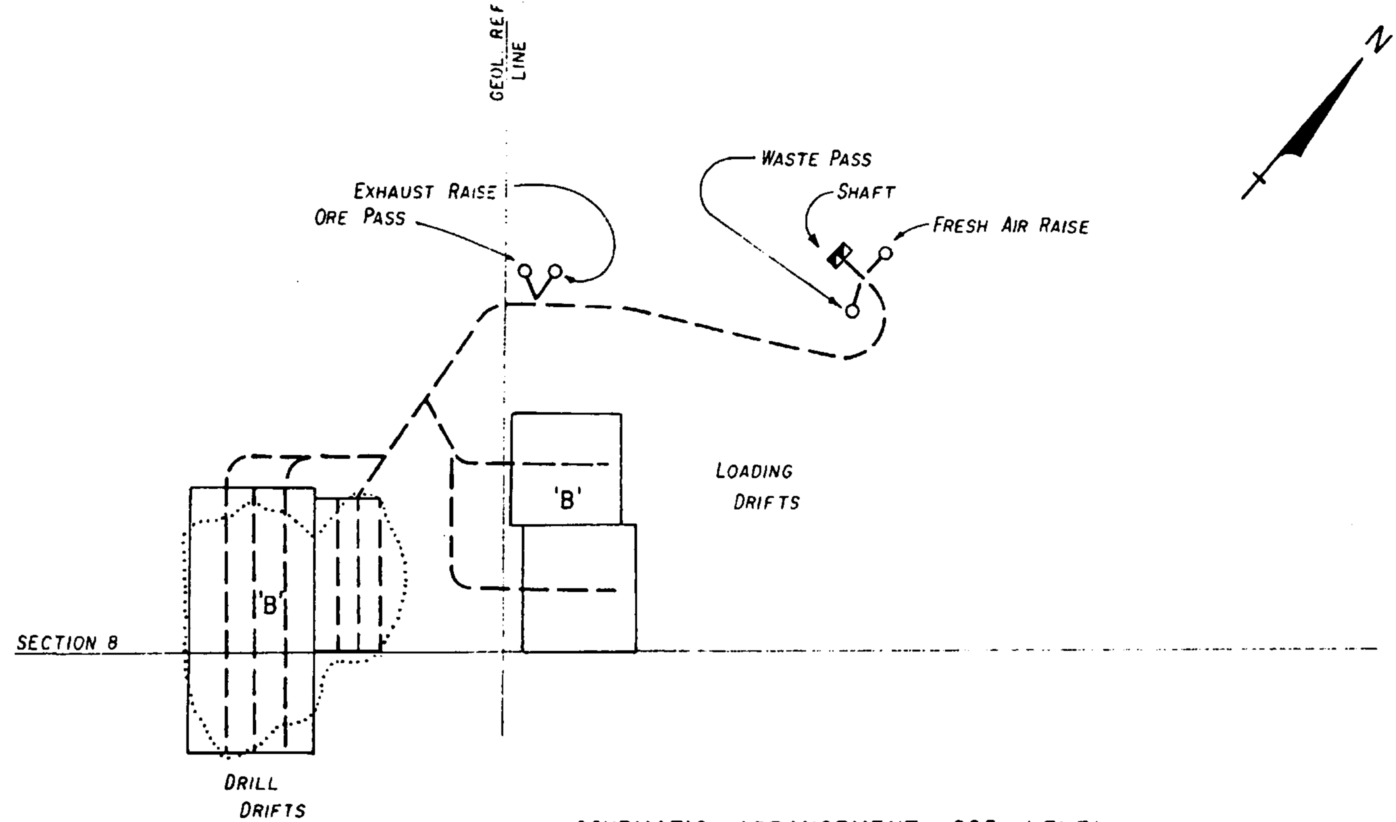
TROUT LAKE PROJECT

SHAFT-ROCK HANDLING

SCALE 1:1000	LOCATION B. C.	DATE June 1982
SURVEY BY T. Hancock	DRAWN BY J. Goh	NO. Fig. 7

SCHEMATIC ARRANGEMENT 960 LEVEL





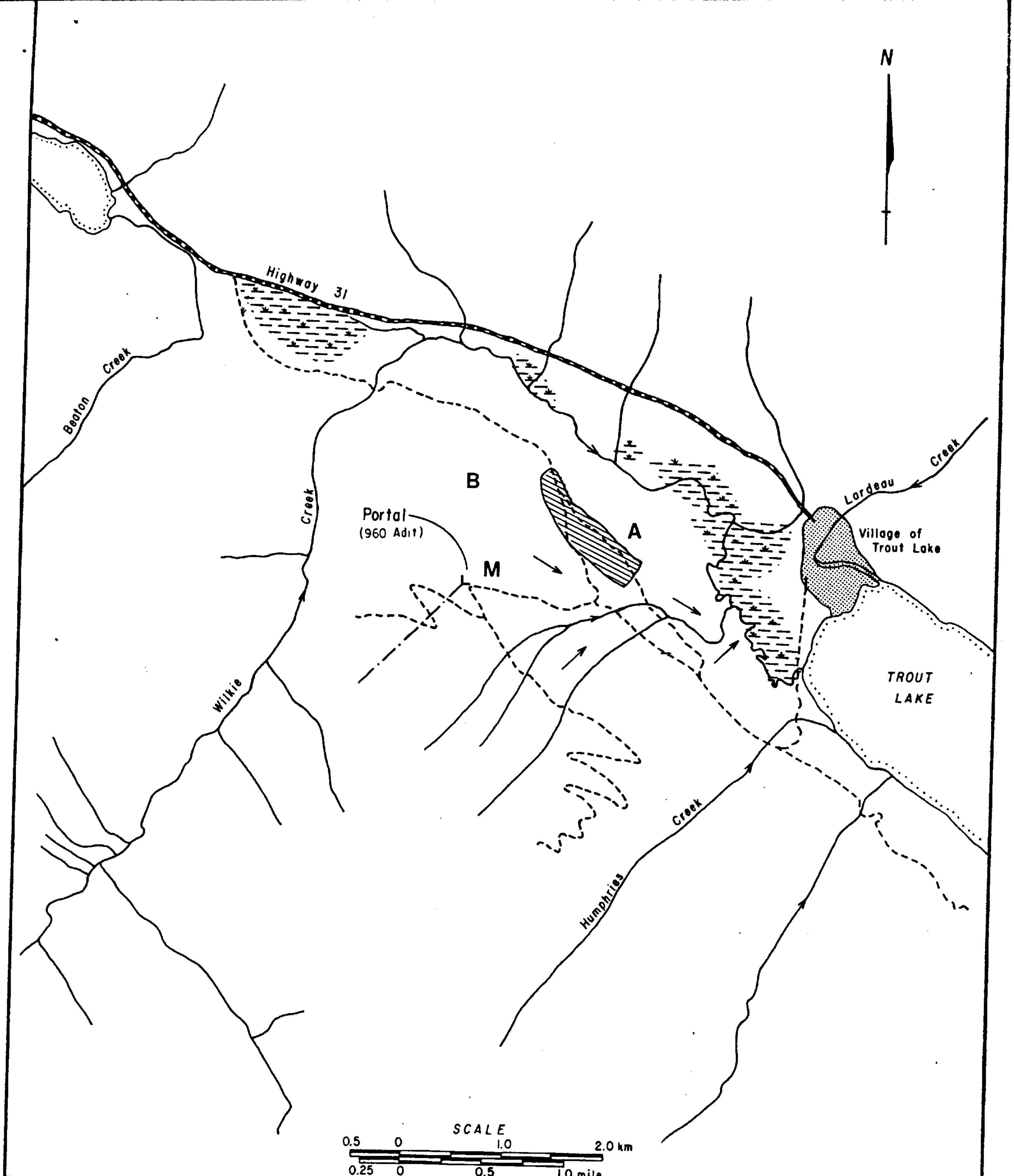
SCHEMATIC ARRANGEMENT 885 LEVEL

SCALE

20 0 40 60 120 m

50 0 100 200 300 ft

NEWMONT MINES LIMITED		
TROUT LAKE PROJECT SUB-960 LEVEL MINING-SCHEMATICS		
SCALE 1:3000	LOCATION B. C.	DATE June 1982
SURVEY BY T. Hancock	DRAWN BY J. Goh	NO. Fig. 8



LEGEND

A — Proposed tailings site A
B — Proposed tailings site B
M — Proposed mill site
→ — Direction of surface runoff

NEWMONT MINES LIMITED

TROUT LAKE PROJECT

LOCATION OF PROPOSED FACILITIES

SCALE 1: 50 000	LOCATION B. C.	DATE June 1982
SURVEY BY T. Hancock	DRAWN BY J. Goh	NO. Fig. 9