W.A. NO. ....

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NAME Geol Rpt

SUBJECT ....

GOTCHA

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PROPERTY FILE

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A REPORT OF
DETAILED GEOLOGICAL MAPPING PROGRAM
WITH PROPOSED DRILL PROGRAM

GOTCHA TUNGSTEN PROPERTY

KAMLOOPS MINING DIVISION

by

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I. INTRODUCTION

This report summarizes the results of a seven day mapping project and is accompanied by a 1 inch representing 10 feet geological map of part of the Gotcha Tungsten property. It is the purpose of this report to (1) discuss briefly the general geology of the property; (2) outline geologically probable tonnages of scheelite mineralization based on data acquired during this project and from previous work on the property; and (3) present the details of a drill program that will prove or disprove the presence of these tonnages. The phrase geologically probable is used informally and the degree of certainty implied should become evident as one reads this report.

II. OWNERSHIP

The Gotcha claim group is owned by United Mineral Services Ltd.

III. LOCATION, ACCESS & TOPOGRAPHY

The Gotcha claims are located 20 miles northeast of Clearwater, British Columbia and are within the Kamloops Mining Division. More specifically, the claims are 2 miles up Maxwell Creek from its junction with Raft River, on the west bank of the creek, at an elevation of about 3,750 ft. The claim group is covered by the Raft River map 82M/13E of the 1:50,000 topographic series and by the northwest corner of the Geological Survey of Canada map 48.1963 Adams Lake 82M/W.

A well maintained logging road along the west bank of Raft River and

Maxwell Creek provides access to the property. This road adjoins the Yellowhead Highway (Route 5) 4 miles east of Clearwater.

The claims are located on the thickly vegetated west side of the valley which in the area of the claims is sloping at 20° to 45° towards Maxwell Creek.

IV. REGIONAL GEOLOGY

The claim group lies within the Omineca Crystalline Belt which is the high grade metamorphic core zone of the Eastern Cordilleran Fold Belt. Rocks in the zone have generally been metamorphosed to upper amphibolite facies and have experienced multiple phases of intense penetrative deformation. The deformation and metamorphism were probably completed by late Jurassic to early Cretaceous times but the stratigraphic age of some of the metasediments is at least 1500 m.yr. Large volumes of "granitic" rock were intruded into the belt during and after the protracted metamorphism and deformation.

The property covers an area of contact between the metasediments and a post metamorphic stock. The metasediments may correlate with the Lower Paleozoic,quartzite,limestone assemblage of the Kootenay Arc to the southwest, though such a correlation is extremely tenuous. The stock may be late Cretaceous or early Tertiary in age based on a single muscovite pottassium-argon age of 64 m.yr. from a phase of the stock.

V. GENERAL GEOLOGY OF THE PROPERTY

a) Previous Work

The property was first found in July 1972 by Union Carbide as a result of a regional stream sampling program. Silts in Maxwell Creek at its confluence with Raft River have anomously high scheelite content; the anomaly can be traced back up the creek to scheelite bearing boulders in the creek below the present pit. Union Carbide worked on the property from 1972 to 1974 and called it the Boulder Group. In the summer of 1972 preliminary trenching exposed mineralized skarn and geological mapping revealed the presence of two skarn bands. Diamond drilling of holes 1, 2, 3, 4, 5, 6, & 7 (total length 1,769.3 ft.) was completed in the winter of 1972/1973. The next summer further mapping and trenching took place. Also a soil sampling grid (7000 ft. x 4000 ft., sample spacing 200 ft.) was established over the property and surrounding ground. Soil samples were collected and panned for scheelite but no major anomalies other than those previously identified in the vicinity of the pit were found. In the winter of 1973/1974 diamond drill holes 8, 9, and 10 (not on accompanying map but located 350 ft. west of DDH 5) were drilled (total length 1,436 ft.).

The property was restaked by United Mineral Services Ltd. in 1977 who extended some of the trenches and exposed more mineralization. N.C.A. Mineral Corp. optioned the property in 1977 and in January 1978 drilled 18 percussion holes (total length 950 ft.) before the option was terminated. In 1978 a pit was opened up and about 1500 tons of about 1.5% scheelite ore mined. Figures 1 and 2 are panorama views of the pit area.

FIGURE 1

PANORAMA OF PIT AREA

FIGURE 2

PANORAMA OF PIT AREA

b) Map Techniques

The accompanying map represents the major output from 7 field days (July 7, 8, 9, 11, 12, 13, 14) spent on the property. Ground control was obtained by surveying in a number of pegged stations. Stations 1, 3, 7, 8, 9, 10, 11, 13, 14, 15, 16, 17 and 18 form a closed loop and were sited in using a brunton compass fixed on a tripod for azimuth, chain or alidade plus staff for distance and alidade for changes in elevation. These stations were back sited to minimize errors. Closure errors shown on the map at station 14 are about 5 ft. laterally and 2 ft. vertically. Additional stations were located in the same way but were not back sited. Outcrops were located by chaining off from the stations. Field mapping was completed using a scale of 1" to 10 ft. and a compass declination of 025°. The elevation of station 1, located 30 ft. southeast of the pit, was arbitarily set at 3,732 ft. The map covers an area of 600 ft. by 350 ft.

c) Rock Types

Most of the outcrop in the area mapped occurs in the pit or in the walls of the trenches. In the rest of the area bedrock is covered by up to 50 feet of till. Outcrops consist of metasediments, skarns and quartz monzonites. No attempt has been made to establish a lithologic succession so that rock types rather than rock units are illustrated on the map. Five rock types have been identified; (1) medium grained biotite-quartz monzonite; (2) medium to coarse grained alaskite or muscovite granite; (3) biotite schist; (4) biotite quartzite; (5) skarn or calc-silicate.

Medium grained, biotite-quartz monzonite - Quartz monzonite outcrops extensively on the access road to the pit as an orange to brown weathering moderately well fractured rock containing no macroscopic fabric. In detail the quartz monzonite is a medium grained, equigranular, biotite (5 to 15%) quartz monzonite. The biotite does not outline a foliation and the rock generally has a very uniform appearance; veins of pegmatite or zenoliths of metasediments are rare. Contacts of quartz monzonite with the metasediments generally appear to be conformable.

Alaskite - Outcrops of alaskite occur in the pit and adjacent to the pit. In the pit fresh boulders are massive and chalky white. The grain size of the alaskite varies from medium to coarse, and the texture is generally equigranular but in places graphitic. Quartz makes up about 60% of the rock, most of the rest is composed of equal proportions of plagioclase and K-feldspar, no mafic minerals are present and muscovite (sericite?) is present in amounts up to 5%. There is no fabric to the rock except for occasional quartz stringers and pegmatite veins. The upper contact of alaskite with skarn is locally discordant and indented by skarn, but over a distance of 200 feet seems to be approximately concordant. There is no distinct border phase, though the upper part of the alaskite may be mixed with a considerable amount of skarn. The lower contact of alaskite with skarn appears to be fairly distinct and concordant.

The alaskite does not extend northwest of the pit and outcrops southeast of the pit along strike are pegmatitic and contain a considerable amount of metasedimentary material. Rocks equivalent to the alaskite are probably intersected in diamond drill holes 2, 3 and 5 where they are generally described as muscovite or leucocratic,quartz monzonite. A second distinct mass of alaskite may be intersected in diamond drill hole 6 which intersects a substantial amount of muscovite bearing intrusive.

Muscovite separated from the alaskite has a K-Ar model age of 64 m.yr. +/- 2 m.yr.

Schist - Outcrops of brown weathering, medium banded, well foliated schist account for about 20% of the exposed rock. The schist is medium grained and contains 40% quartz, 20% feldspar and 20% biotite. More exotic minerals are absent and the only major variation from this sample mineralogy is the appearance of large disoriented flakes of sericite in the schist near contacts with the alaskite or quartz monzonite. The schist grades to biotite quartzite with increase in quartz content.

Biotite Quartzite - This rock type forms massive to medium banded, brown weathering outcrops that account for about 10% of the exposed rock. In appearance the quartzite is similar to the schist but banding is coarser and foliation is absent.

Skarn or Calc-silicate - calc-silicate rocks and skarns derived from them, make up about 30% of the exposed rock. Outcrops of calc-silicate have a distinctive grey,pitted surface if the rock contains significant amounts of calcite or are grey-green and coarse banded if the amount of silica is high. Calc-silicate adjacent to alaskite or quartz monzonite is converted to skarn of which there are 3 major types; types (1) and (3) are scheelite bearing.

1) Quartz-garnet (grossularite?)-idocrase skarn
2) Wollastonite-garnet-calcite skarn
3) Diopside-quartz skarn.

Quartz-garnet-idocrase skarn forms massive, rough surfaced, brown outcrops with indistinct layering. In hand specimen it is coarse to very coarse grained containing from 10 to 50% idocrase, 10 to 50% garnet and 10 to 50% quartz. Garnet occurs as clusters of euhedral, medium grained crystals or as coarse grained, subhedral crystals. Often there appears to be two generations of garnets with the subhedral crystals belonging to the earlier generation. Idocrase forms coarse grained, sub-euhedral crystals. Quartz forms a coarse grained matrix to these two mafic minerals. Figure 3 is a close-up photo of this rock type.

Wollastonite-garnet-calcite skarn forms chalky white, rough surfaced outcrops. Garnet which makes up 5 to 20% of the rock occurs as medium grained, equigranular crystals clustered together in 1 to 5 cm. diameter masses. Wollastonite forms radiating masses growing outward from the garnet masses. Calcite occurs as medium to coarse grained masses often outlining the indistinct layering.

Diopside-quartz skarn forms massive to medium banded, grey to greenish outcrops. The fine grained nature of the skarn precludes a detailed description of its mineralogy but it certainly contains a high proportion of diopside and probably other minerals such as actinolite and epidote. Fresh samples are dark green to black, medium to fine grained and fine to medium banded. Figure 4 is a close-up photo of this rock type.

The calc-silicate - skarn rock type is coloured on the map as a single rock type, however, varieties of skarn are indicated by a superimposed letter code which is explained on the map. In addition, in the vicinity of the pit, the skarn derived from the calc-silicate forms 5 major bands which are numbered on the map. Three of these bands contain economic quantities of scheelite mineralization. The five bands are described in more detail under the heading of mineralization.

FIGURE 3

QUARTZ-GARNET-IDOCRASE SKARN

FIGURE 4

DIOPSIDE-QUARTZ SKARN

d. Structural Geology

Contacts between the various rock types and the layering in them trend northeast and dip northwest with no great change in orientation. Figure 5 is a stereonet plot of layering measurements. Bands of the various rock types vary greatly in present thickness and converge and diverge with each other though no major fold closures are clearly evident. Layering and foliation in the schist are parallel and a fine crinkle lineation can be identified on the foliation. This lineation, which generally plunges shallowly northeast is parallel to the hinges of mesoscopic, rootless, isoclinal folds which are occassionally observed in schist (Figure 6). The lineation and isoclinal folds constitute the only evidence for an early phase of isoclinal folding. Large scale isoclinal folds may be present in the area but have not been positively identified.

South of the pit near station 23 the layering is deflected round an open fold whose hinge plunges 230°/40°. This structure post dates the isoclinal folding and may represent evidence for a later phase of pervasive deformation that could pre or post date the alaskite or be caused by its intrusion though it most likely predates intrusion of the alaskite.

Intrusive rocks are well jointed and 2 conjugate sets are evident (020°/30°W and 105°/60°S) which intersect about a line plunging steeply southwest. These are probably shear joints. Fracture zones in the metasediments generally seem to parallel layering. One fracture zone identified in the quartz monzonite is oriented 010°/60°W. Fracture or alteration zones were encountered in most diamond drill holes though generally core recovery was good. No fault with an accompaying offset could be identified in the pit

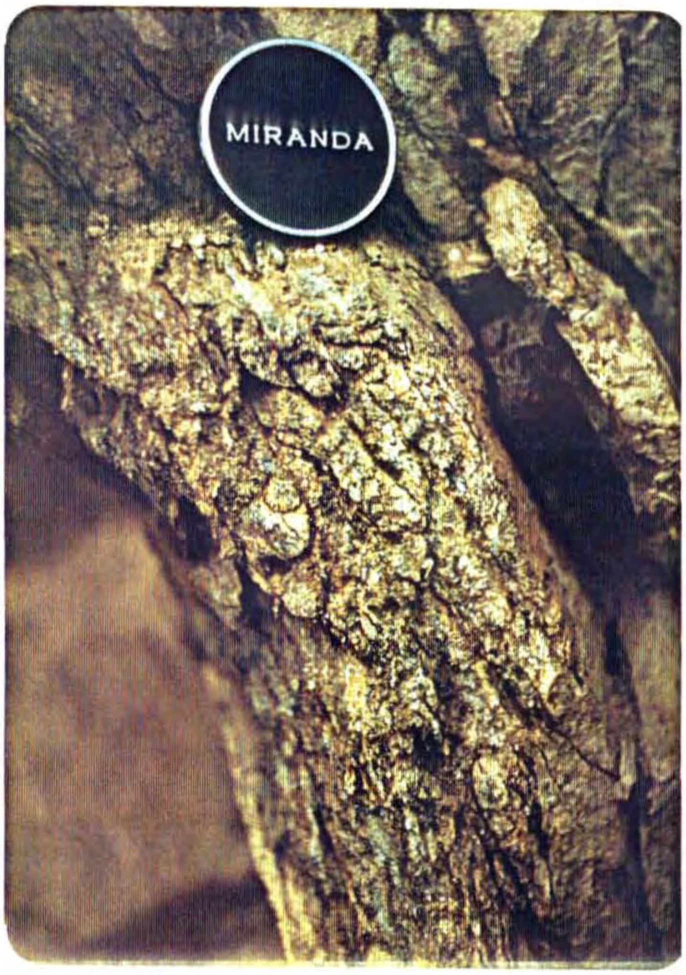
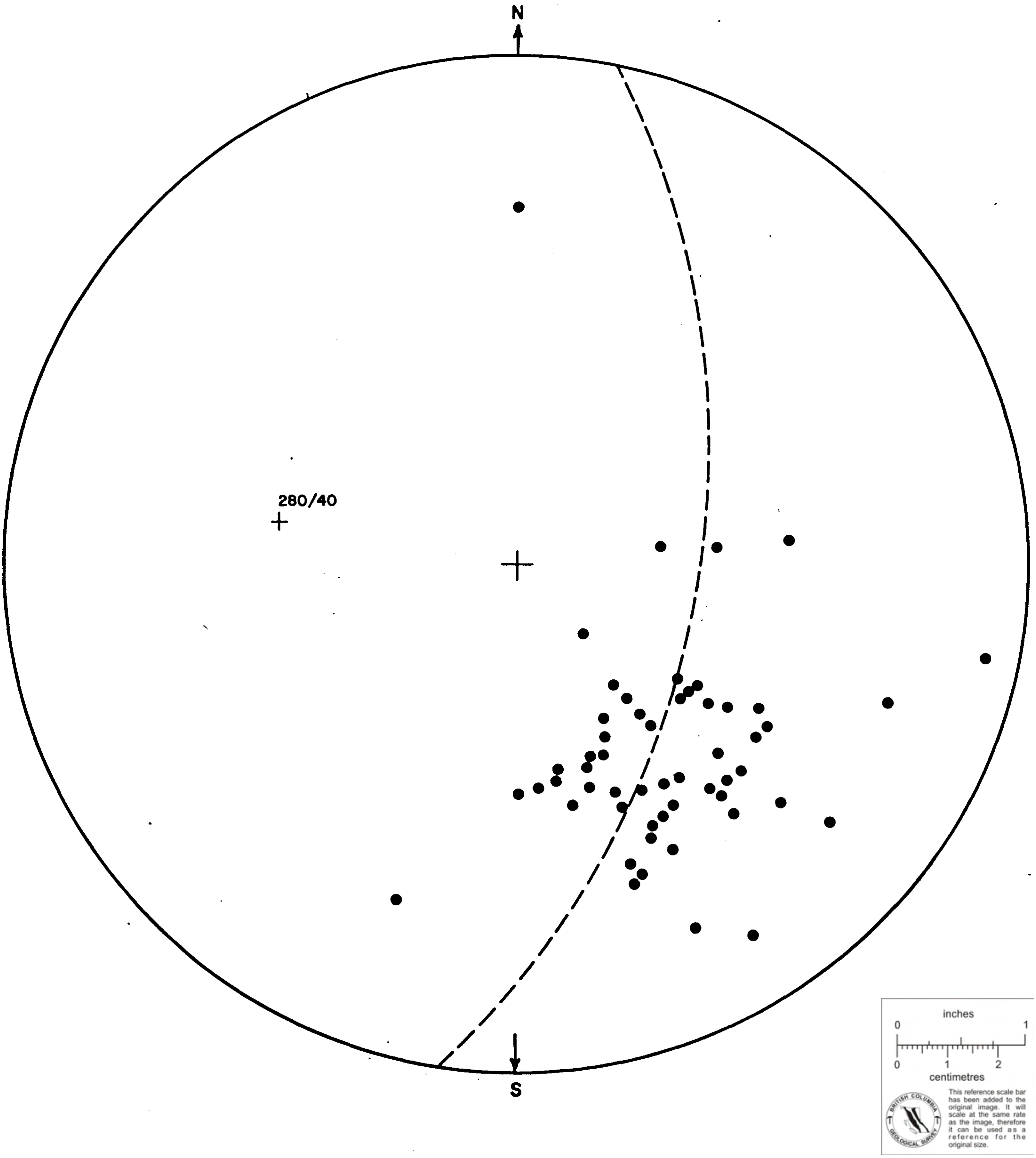


FIGURE 6

POSSIBLE SMALL SCALE FOLDS IN SCHIST

FIGURE 5

STEREOGRAPHIC PLOT OF POLES TO LAYERING



or surrounding area.

e. Metamorphism and Skarn Formation

The rock types have been affected by regional metamorphism of upper amphibolite grade that probably finished by late Jurassic time; by contact metamorphism, caused by intrusion of the quartz monzonite, and possibly at about the same time by metasomatism originating from the alaskite. The contact metamorphism and metasomatism probably occured 64 m.yr. ago, long after the deformation and regional metamorphism had ceased.

It is difficult to distinguish between the effects of contact metamorphism and metasomatism but in the pit area a sequence of metasomatic changes can tentatively be identified. The quartz-garnet-idocrase skarn seems to be the end result of metasomatism of a calcium rich, calc-silicate. The first stage of metasomatism involved introduction of iron and crystallization of garnet probably grossularite. The second stage saw introduction of silica and crystallization of wollastonite. This was followed by continued introduction of silica and crystallization of idocrase and garnet possibly at the expense of wollastonite. The final stage was represented by silica flooding and introduction of tungsten which crystallized as scheelite (powellite has not been observed).

The diopside-quartz skarn apparently has not experienced the same degree of metasomatism though there are vuggy zones in which patches of coarse calcite and idocrase crystals occur. Scheelite seems to be concentrated in these zones but also occurs as coarse grained poikiolitic crystals enclosing diopside, scattered through the rock.

The presence of wollastonite (confirmed by X-ray diffraction work) indicates that the skarn formed under conditions of high temperature and low pressure. Under these conditions the muscovite in the alaskite (if it is of igneous origin) was probably metastable at the time of crystallization.

VI. STRUCTURAL INTERPRETATION AND ORE CONTROLS

The primary control of the location of mineralization is the intrusive contact of the quartz monzonite with the metasediments. Previous mapping and drilling indicates that this contact trends northerly with metasediments to the west, but in the area of the pit there is an embayment of metasediments projecting northeasterly into the intrusive mass. The embayment probably acted as an energy and chemical trap. The alaskite (granitic composition) which is probably a late phase of the quartz monzonite, crops out in the embayment and probably acted as a channel way, carrying iron, silica and tungsten plus other elements from the nearly solid intrusion into the calc-silicate metasediments. The exposed scheelite mineralization is restricted to metasomatised calc-silicate outcropping in the embayment.

The northern, west-trending intrusive wall of the embayment may be a fault or discordant intrusive contact. One interpretation based on data in diamond drill hole 2 and near diamond drill hole 1 is that it is a late fault oriented 050°/55°SE. This postulated fault does not parallel any joint set but has the correct orientation to be a tension fracture related to the 2 joint sets identified. The five skarn bands identified in the pit area are assumed to be cut off by the fault. This model provides a northern limit to the mineralization in the pit area and helps in the estimation of ore tonnages.

Folding may complicate the simple model proposed. One possibility is that Škarn Bands 1 and 3 are respectively the upper and lower limbs of an overturned,nearly isoclinal synform. In support of this suggestion it can be seen from the map that the two bands converge to the north-east and that the open fold in Band 3, near station 23, is the correct sense for its position on the lower limb of a synform. The open fold plunges 280°/40° which would presumably be the same as the plunge of the synform. A plunge of 280°/40° would project the hinge of the synform into the fault below a point about 20 feet north of the pit.

Alternatively Band 3 and 4 may represent a fold, in which case this would most likely be a northeasterly plunging antiform related to the earlier isoclinal folding. The plunge of this structure would be the same as the crinkle lineations in the area and therefore its hinge would almost parallel the topography into Maxwell Creek. In the absence of adequate diamond drill hole information in the pit area neither of these fold hypotheses can be checked, but they do not greatly influence the fault plus skarn slab model used in predicting ore tonnages.

VII. SCHEELITE MINERALIZATION

a) General Distribution and Petrogenesis

Scheelite occurs in parts of Skarn Bands 1, 3 and 4 and to a lesser extent in Skarn Band 2. Within these bands it was found in the quartz-garnet-idocrase skarn and the diopside-quartz skarn. Scheelite was not found in skarns containing wollastonite or skarns containing more than 10% calcite. In the quartz-garnet-idocrase skarn scheelite percentage is higher if the quartz content is between 10 and 60 percent.

In the quartz-garnet-idocrase skarn scheelite occurs as anhedral rounded crystals 2mm to 2cm in diameter evenly distributed throughout or defining indistinct layering. The percentage of scheelite in individual samples can range up to 10%. Scheelite crystals contact all major skarn minerals but are not seen to enclose any other mineral. Despite this it appears that scheelite was one of the last minerals to crystallize in the skarn.

Scheelite occurs as large poikiolitic crystals in the diopside-quartz-skarn in part concentrated in vuggy zones. It appears to have crystallized after the diopside.

b) Chip Sampling and Lamping

The pit area was lamped on two nights using an ultraviolet lamp. Extensive scheelite mineralization was observed in Bands 3 and 4 and in boulders in the pit. Lamping of samples during the day revealed good scheelite mineralization in outcrops on the slope northwest of the pit. Eight chip sample lines (which are located on the map) were established across the skarn bands. Samples were collected along these lines and the present thickness of the band was measured. The lines were lamped at night and their grade estimated. The results of this sampling are tabulated in Table 1. The grade estimates from lamping should be reasonably accurate. Any results from the chip samples may not be representative because of the difficulty of obtaining equal amounts of material from each foot of sample line.

c) Skarn Band 1

General Petrology - The Band includes a number of skarn types. Generally the upper part of the Band is coarse grained,wollastonite-garnet-calcite skarn which grades downwards into quartz-garnet-idocrase skarn which sometimes contains scheelite.

Structure of Band 1 - The alteration (fracture?) zone cropping out 10 feet south of station 45 is assumed to be the base of the Band. This zone is recognized in diamond drill hole 2 but not in diamond drill hole 5 or 1. The orientation of the zone seems to be 050°/55°. Band 1 is cut by an intrusive contact or fault near diamond drill hole 1 and by a fault in diamond drill hole 2 at 89 ft. This fault, previously described, limits the downdip extent of Band 1.

Lithologic sections

Near station 38 base of Band not exposed

Present thickness (exposed) = 20ft.

Top-coarse grained, garnet calcite skarn - Bottom

Near diamond drill hole 1 present thickness = 25 ft.

Top - coarse grained wollastonite-garnet skarn / coarse grained, quartz-garnet-idocrase skarn/ banded,diopside-quartz skarn - Bottom

TABLE 1

SAMPLE LINE WO_3 GRADE ESTIMATES BY NIGHT ULTRA VIOLET LAMPING

Line No.	True Unit Thickness	Lamping Results
1	14	0-6 (T), 6-8 (3%), 8-10 (2%) (10-14 T)
2	19	0-15 (T)
3	11	0-12 (T)
4	10	0-2 (5%), 2-4 (1%), 4-6 (1.5%), 6-8 (2%), 8-10 (2%)
5	6	0-6 (T)
6	6	0-2 (1.5%), 2-4 (2.5%), 4-5 (1%)
7	8	0-3 (T), 3-4 (3%), 4-5 (5%), 5-7 (T)
8	18	0-2 (T), 2-3 (1%), 3-19 (T)
9	76' N-S line through pit	38, 2' Ft sections average = 1%

N.B. T equals trace WO_3

In diamond drill hole 2 present thickness = 40ft

Top - coarse grained,garnet wollastonite skarn / banded, diopside skarn / coarse grained,garnet-diopside skarn - Bottom

North end of the pit base partially digested by alaskite,present thickness remaining 10 ft.

Top coarse grained, calcite wollastonite skarn /coarse grained, quartz-garnet idocrase skarn / alaskite - Bottom.

Mineralized sections

1) 2 ft in diamond drill hole 2 at 1% scheelite
2) approximately 10 feet at 1% in outcrop south of diamond drill hole 1.

Tonnage possibilities Band 1

Band 1 is cut by the fault to the north and is not mineralized in diamond drill 5. The surface extent of probable mineralization can be approximated by the triangle A C D A is near diamond drill hole 1. D is along strike to the southwest from A and C is near the mineralized portion in diamond drill hole 2. A mineralized thickness of 8 ft. is assumed to exist at A and 2 ft. at C and D. These thicknesses should be conservative as is the extent of triangle A C D. The volume of the prism AA' CC' DD' (Figure 7) is 30,400 cu.ft. which is equivalent to 2763 tons (11 cu.ft. = 1 ton). A grade of 1% is assumed for this tonnage.

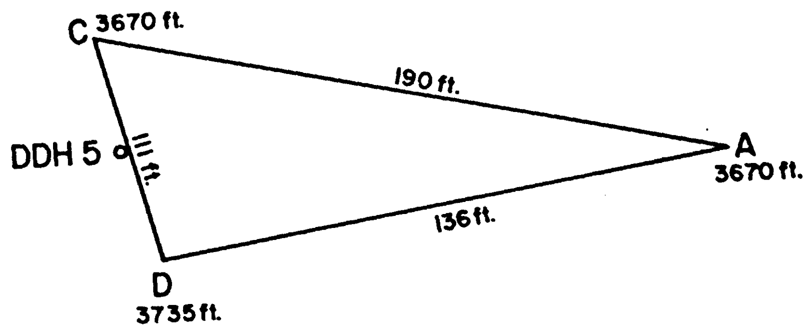
d) Skarn Band 2

Band 2 outcrops near station 28 where it is a coarse grained, quartz-garnet-idocrase skarn. To the south it is covered by overburden and may in fact merge with Band 1. Near station 28 the top of the Band is in

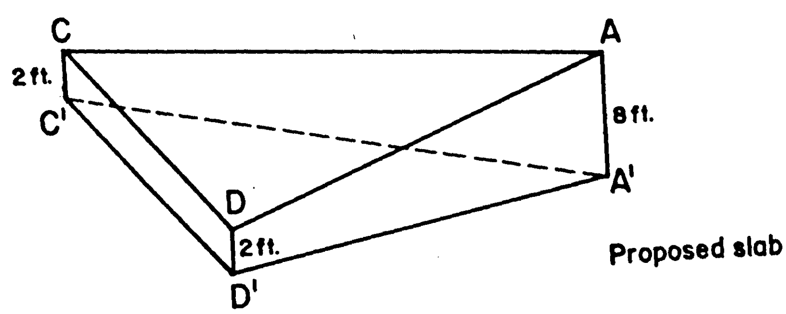
FIGURE 7

MINERALIZED PORTION OF BAND I

(NOT TO SCALE)



Thickness at A = 8 ft. at C = 2 ft.



Volume of slab = 30,400 cu.ft.

Tonnage = 2,763 tons

Assumed grade = 1%

conformable contact with alaskite and the bottom of the Band is fractured and possibly sheared but appears to be conformably underlain by alaskite.

There is some scheelite mineralization in the center part of chip sample line 8 which traverses the band near station 28 but it is not of economic thickness.

e) Skarn Band 3

Petrology - Much of the mineralized part of Band 3 is diopside-quartz-garnet-idocrase skarn and banded, diopside skarn. No wollastonite was found in Band 3 and the band grades rapidly into banded calc-silicate southwest of the pit. In the pit area the type of skarn within Band 3 changes abruptly from outcrop to outcrop.

Structure - In the northeast Band 3 is separated from Band 1 by about 20 ft. of schist while towards the southwest, separation of the two bands increases and alaskite and Band 3 intervene. The strike of Band 3 is fairly constant except for the open fold southwest of the pit. The dip of the band is variable but for the purpose of tonnage calculations it is assumed to be 55°. The base of the band is generally schist or quartzite and in the pit area the top is alaskite. A gossany zone which is mapped near station 25 and in diamond drill hole 5 as a pyritic zone may be a useful marker for identifying Band 3.

Lithologic Sections

South end of pit and trench present thickness at least 50 ft.

Top - coarse grained, quartz-garnet-idocrase skarn / fractured gossany zone/ quartz rich skarn or quarztite / coarse grained, quartz-garnet skarm / fine banded to massive, diopside-quartz skarn - Bottom.

Diamond drill hole 3 present thickness about 30 ft.

Top - fine grained, banded, diopside-quartz skarn / medium to coarse grained, garnet-diopside-quartz skarn / fine grained, massive, diopside skarn with pyrite / coarse grained, massive, garnet-diopside-quartz skarn - Bottom.

Diamond drill hole 5 about 15 ft. present thickness.

Top - coarse grained, quartz-garnet-diopside-idocrase skarn / banded, diopside skarn, gossany in part - Bottom.

Northeast of pit and down slope present thickness about 30 ft.

Top - coarse grained, garnet-quartz-idocrase skarn / schist / banded, diopside skarn, banded skarn? (inaccessible) - Bottom.

Mineralized sections

1) Percussion drill hole 1 intersected about 20 ft. present thickness at 1.34%
2) Percussion drill hole 5 intersected about 10 ft. present thickness at .5%
3) Percussion drill hole 3 intersected about 20 ft. present thickness 1.56%
4) Chip sample line 4 lamping results suggest about 10 ft. present thickness at 2.3%
5) Diamond drill hole 3 intersected the equivalent of 20 ft. present thickness at 1.56%
6) Northeast of pit and down slope is difficult to evaluate because of terrain but 10 ft. present thickness at 1.5% should be conservative.
7) Diamond drill hole 5 minor scheelite over less than 1 ft.

Also percussion drill hole 9 checked down dip extension for 70 ft. at

3.77% and percussion drill hole 7 checked down dip extension for 50 feet at .65%

Tonnage possibilities

Mineralization does not seem to extend west southwest of a line through station 21 to mineralization in diamond drill hole 5. Mineralization is cut off by the topography to the northeast and by the fault to the northwest. The block of potentially mineralized ground is outlined on the map by triangle EFG and Figure 8. Based on exposed mineralization a thickness of 10 feet is assigned to F,a thickness of 0 feet to G and a thickness of 15 feet to E. A thickness of less than 2 feet is not considered mineable so that only the tonnage of EIHFF'E'I'H' (Figure 8) is considered. This block contains 16,134 tons. A grade of 1.5% is assigned to this tonnage.

f) Skarn Band 4

Band 4 is composed of fine grained, siliceous skarn, fine grained, diopside-quartz skarn and a central gossany zone. The Band is separated from Band 3 by schist and a schist plus pegmatite mixture. It may merge with Band 3 to the northeast but for the purpose of ore tonnage calculations it is assumed to dip at 55° and to be 25 ft stratigraphically below Band 3. The base of the Band is marked by schist or quartzite.

Chip sample lines 5 and 6 traverse the Band and lamping results indicate about 5 ft. present thickness at about 1.5%. The band was intersected by diamond drill hole 3 but no mineralization was encountered.

The mineralization extends along strike for 20 feet between chip sample lines 7 and 6 but stops before chip sample line 5. For the purpose of tonnage estimation a mineralized strike length of 60 ft. is assumed and the slab is limited down dip by diamond drill hole 3. The resulting volume is illustrated in Figure 9, and provides 779 tons with an assumed grade of 1.5%.

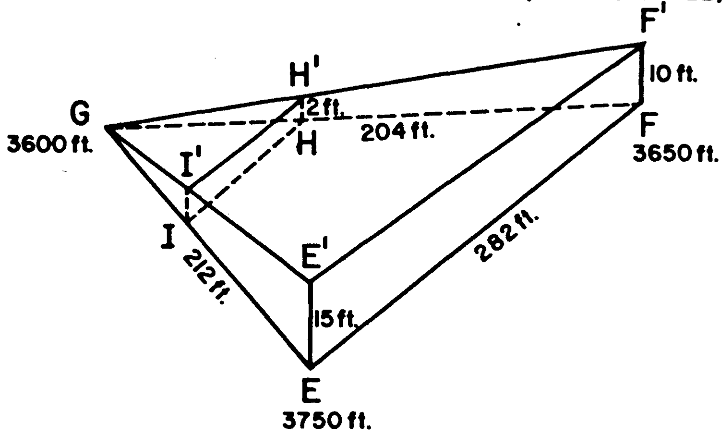
Skarn Band 5

Skarn Band 5 is intersected in diamond drill hole 2 above Band 1. It does not outcrop and is not mineralized in diamond drill hole 2. This band forms a small wedge against the fault but because of its location near other mineralized skarn bands it has been delineated.

FIGURE 8

MINERALIZED PORTION OF BAND 3

(NOT TO SCALE)



Volume of E E' G F F' = 178,600 cu.ft.

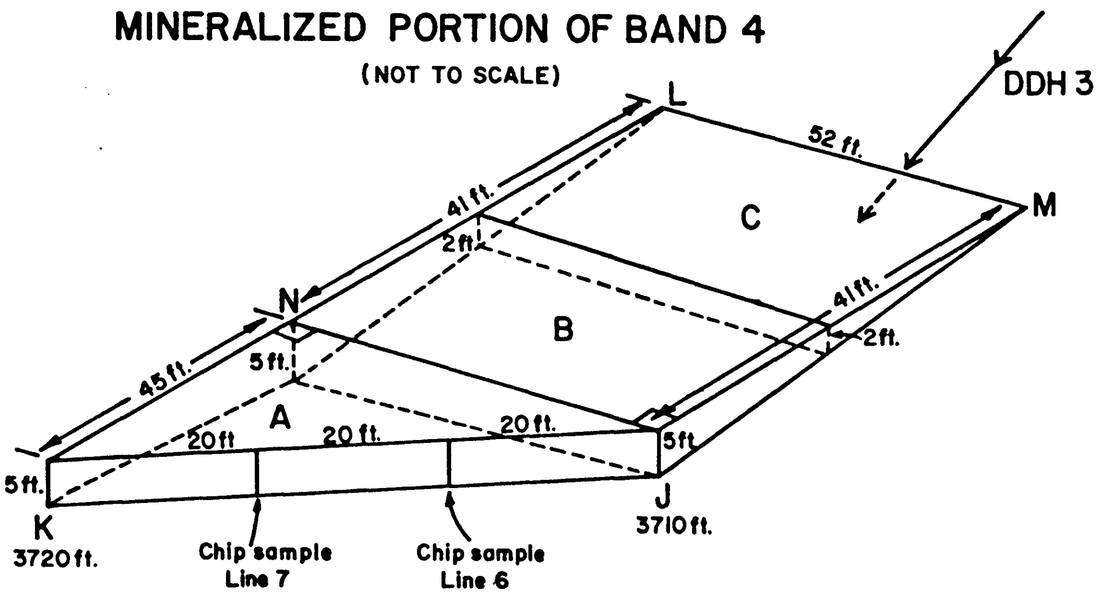
Volume of I I' H H' G = 1,120 cu.ft.

Tonnage truncated pyramid = 16,134 tons

FIGURE 9

MINERALIZED PORTION OF BAND 4

(NOT TO SCALE)



Volume of A+B = 8,500 cu.ft.

Tonnage = 779 tons

VIII DRILL PROGRAM

a) Introduction

Three primary and two secondary drill sites have been planned. A total of 12 drill holes have been projected to give a total length of 1,490 feet. The drill program is flexible to allow for additional and/or alternative holes to be drilled as drilling progresses. Sections and logs have been constructed for each planned hole but these are only rough guides of what to expect.

Access to the planned drill sites will require construction of 2 short cat roads. The road to give access to drill site T (plotted on map) is planned to start 20 feet southeast of station 16 and to cut back angling down slope through stations 24, 4 and 21. This road as well as giving access to the bench south of the pit should expose any extension of Band 4 to the southwest. The road to give access to sites S and R is planned to start 20 feet north of station 10 and to cut back angling down slope towards station 3. This road should expose Band 5.

Maximum flexibility of the drilling program will be retained if the following sequence is followed. First set up at site R and drill holes 1, 2 and 3. If no additional holes are planned for this site move to site S and drill holes 4, 5, 6, 8, and 9. If the cat road to sites S and R exposed mineralization, site another drill hole back along the road, otherwise move round top of the pit to site T. From site T drill holes in sequence 10, 11 and 12. It may prove worthwhile to set up on the access road near station 1 and drill additional holes based on the results of drilling from site T.

It is important to prepare drill sites as well as possible and to locate them accurately to minimize drill costs. Water will have to be piped at least 1000 feet. A good source of water with sufficient head should be located so that if possible the cat can help make access to it.

Most of the inclined drill holes are planned to intersect layering at close to 90°, if core bedding/angles are consistently less than 60° layering is shallowing or steepening with depth and sections should be reinterpreted before casing is pulled.

Site R

Three holes are planned for Site R. Figure 10 illustrates the location with respect to Band 1, Band 3 and Band 4. Piercing points of the surfaces of these Bands are plotted on the map and labelled in Figure 10.

Hole 1	0-25 ft. projected to intersect 9 ft. of mineralization
Vertical	equivalent to a present thickness of 5 ft.
	25 ft. base of skarn Band 1
	25 - 60 ft. schist mixed with skarn (Band 2) and alaskite
	60 ft. fault
	60 ft.+ intrusives.
	100 ft. maximum projected length.
Hole 2	0-15 ft. Band 1 projected to intersect 5 ft. of mineralization
145/42°	equivalent to a present thickness of 5 ft.
	15 - 50 ft. 60% alaskite 30% skarn (Band 2) 10% schist.
	50 - 78 ft. Band 3 projected to intersect 15 ft. mineralization equivalent to 15 ft. present thickness.

78 - 100 ft. schist and quartzite
100 - 105 ft. Band 4?
150 ft. maximum projected length.

Hole 3 145°/63°
0 - 18 ft. Band 1, projected to intersect 5 ft. mineralization equivalent to 5ft present thickness.
18 - 50 ft. 60% alaskite 30% skarn (Band 2) 10% schist
50 - 90 ft. Band 3, projected to intersect 15 ft. mineralization equivalent to 15 ft present thickness.
90 - 113 ft. quartzite and schist
113 - 118 ft. Band 4?
150 ft. maximum projected length.

Depending on the results of drilling it may be useful to spot more holes to intersect Band 3.

Site S

Six holes are planned for Site S, Figure 11 illustrates the location of holes 4, 5, and 6, Figure 12 illustrates the location of holes 7 and 8 and Figure 13 illustrates the location of hole 9.

Hole 4 145°/40°
0-26 ft. Band 1, wollastonite skarn some quartz-garnet skarn and alaskite. Projected to intersect 6 ft. mineralization equivalent to 6 ft. present thickness.
26-50 ft. mostly alaskite
50 to 79 ft. Band 3, projected to intersect 12 ft. mineralization equivalent to a present thickness of 12 ft.
79-110 ft. quartzite plus schist
110-115 ft. Band 4?
150 ft. maximum projected length

Hole 5	0-21 ft. Band 1, wollastonite skarn, quartz-garnet skarn, alaskite. Projected to intersect 6 ft. mineralization equivalent to 6ft. present thickness.
	21-60 ft. mostly alaskite
	60-93 ft. Band 3, projected to intersect 10 ft. mineralization equivalent to 10 ft. present thickness.
	03-113 ft. schist and quartzite
	113-123 ft. Band 4?
	150 ft. maximum projected length
Hole 6 Vertical	0-46 ft. wollastonite skarn, quartz-garnet skarn and alaskite. Projected to intersect 8 ft. mineralization equivalent to 4.5 ft. present thickness.
	46 - 55 ft. alaskite, minor skarn (Band 2)
	55 ft. fault
	55-100 ft. biotite-quartz monzonite
	100 ft. maximum projected length.
Hole 7 233°/62°	0-63 ft. wollastonite skarn, quartz-garnet skarn. Projected to intersect 3.2 ft. mineralization equivalent to 2 ft. present thickness.
	63-74 ft. alaskite, minor skarn
	74 ft. fault
	74 ft - 100 ft. biotite-quartz monzonite
	100 ft. maximum projected length.
Hole 8 233°/43°	0-31 ft. wollastonite skarn, quartz-garnet skarn. Projected to intersect 4.2 ft. mineralization equivalent to present thickness 2 ft.

31 - 94 ft. mostly alaskite, minor skarn

94 ft. fault

94 ft. - 120 ft. biotite-quartz monzonite

120 ft. maximum projected length

Hole 9 165°/62°
0-36 ft. Band 1, projected to intersect 2.5 ft. mineralization equivalent to 2.5 ft. present thickness.

36 to 130 ft. alaskite, minor skarn possibly mineralized (Band 2)

130-160 ft. Band 3, projected to intersect about 8 ft. mineralization equivalent to 8 ft. present thickness.

160-130 ft. schist and quartzite

180 ft. maximum projected length.

Site T

Three holes are planned for Site T, figure 14 illustrates the location of these holes. They are primarily designed to check the down dip extension of Band 4.

Hole 10 146/55°
0-42 ft. Band 3, projected to intersect 15 ft. mineralization equivalent to 15 ft. present thickness.

42-60 ft. schist

60-69 ft. Band 4, projected to intersect about 4 ft. mineralization equivalent to 4 ft. present thickness

69-100 ft. schist and quartzite

100 ft. maximum projected length.

Hole 11 Vertical

0-70 ft. Band 3,projected to intersect 26 ft. mineralization equivalent to 15 ft. present thickness.

70-105 ft. quartzite and schist

105-114 ft. Band 4,projected to intersect about 8 ft. mineralization equivalent to about 4 ft. present thickness.

114 - 140 ft. quartzite and schist

140 ft. maximum projected length.

Hole 12 180°/60°

0-45 ft. Band 3,projected to intersect about 20 ft. mineralization equivalent to 15 ft. present thickness.

45 - 65 ft. quartzite and schist

65 ft. - 75 ft. Band 4, projected to intersect about 6 ft. of mineralization equivalent to 4 ft. present thickness.

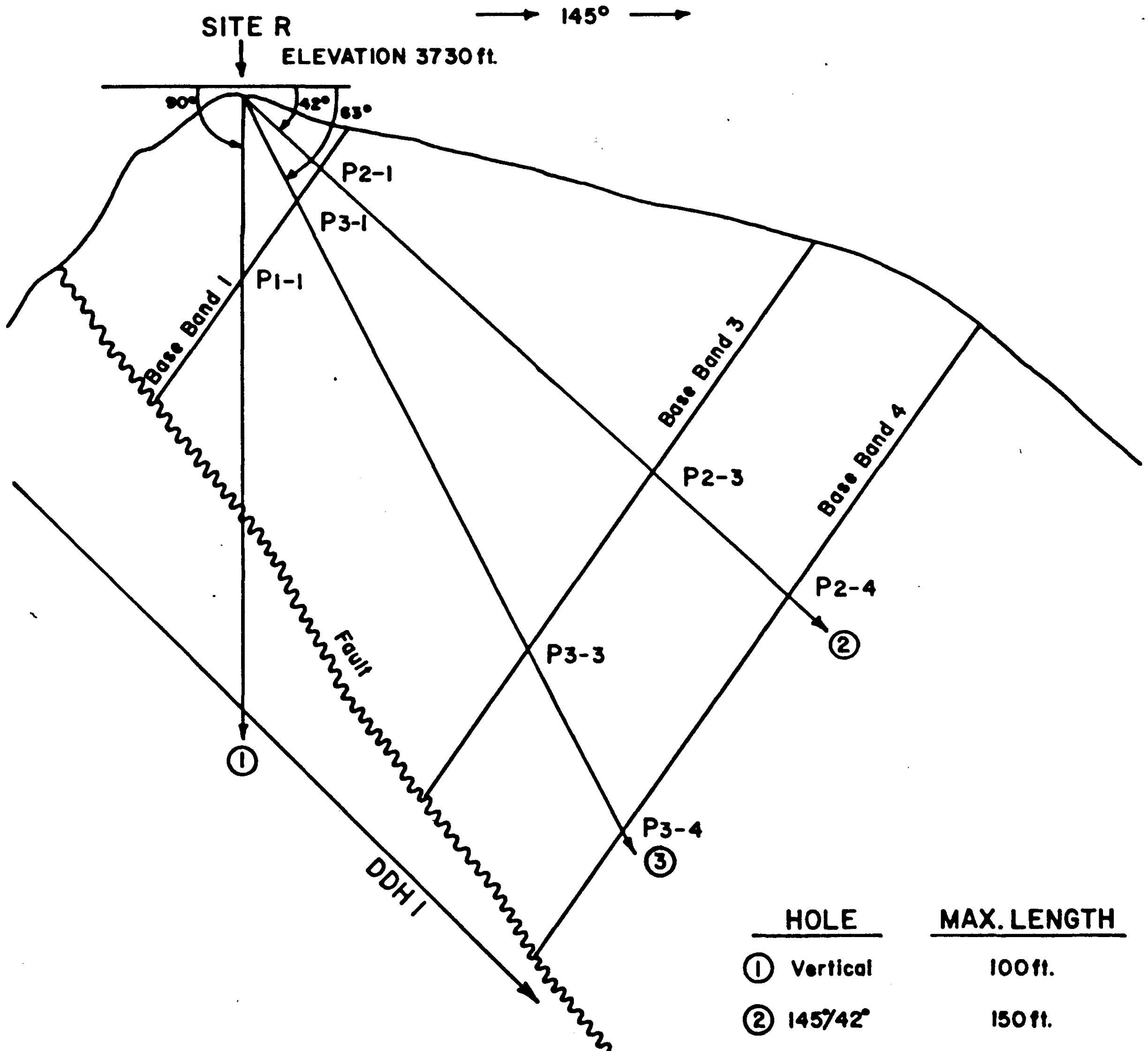
75 - 100 ft. quartzite and schist

100 ft. maximum projected length.

FIGURE 10

VERTICAL SECTION TRENDING 145° THROUGH SITE R

(SCALE 1cm = 10ft.)

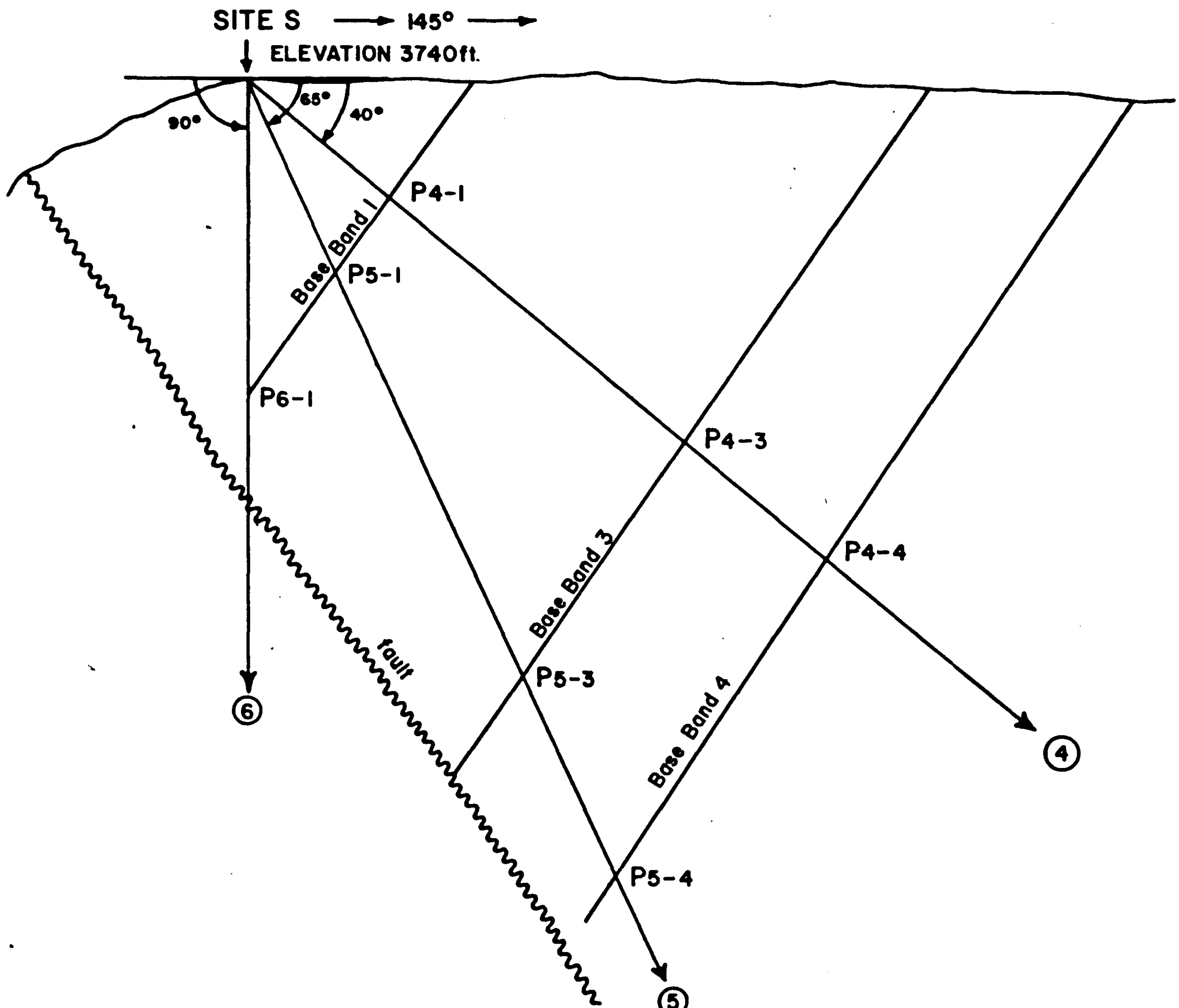


HOLE	MAX. LENGTH
① Vertical	100 ft.
② 145°/42°	150 ft.
③ 145°/63°	150 ft.

FIGURE 11

VERTICAL SECTION TRENDING 145° THROUGH SITE S

(SCALE 1cm = 10ft.)



HOLE	MAX. LENGTH
④ 145°/40°	150ft.
⑤ 145°/65°	100ft.
⑥ Vertical	100ft.

FIGURE 12

VERTICAL SECTION TRENDING 233° THROUGH SITE S

(SCALE 1cm = 10ft.)

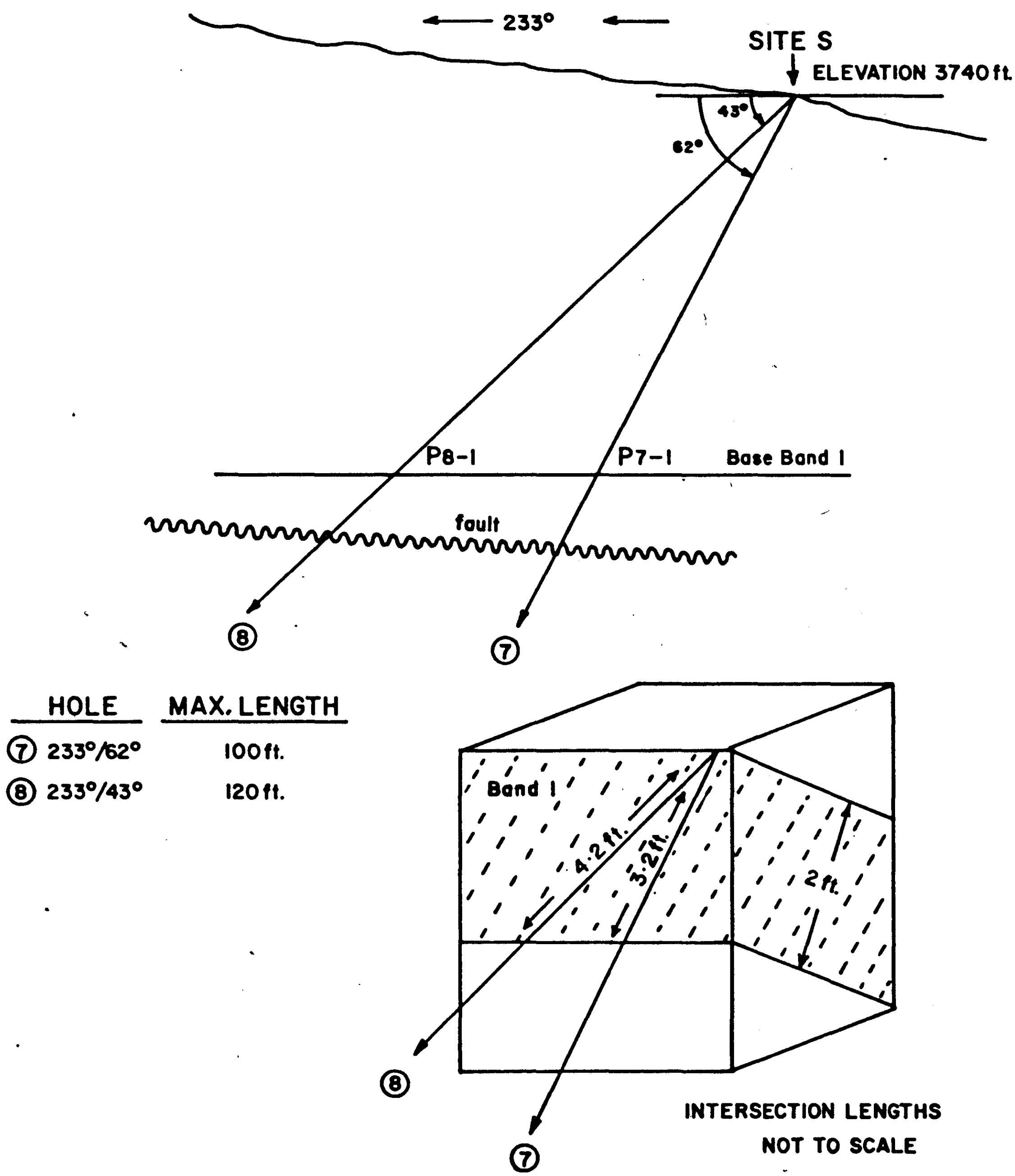
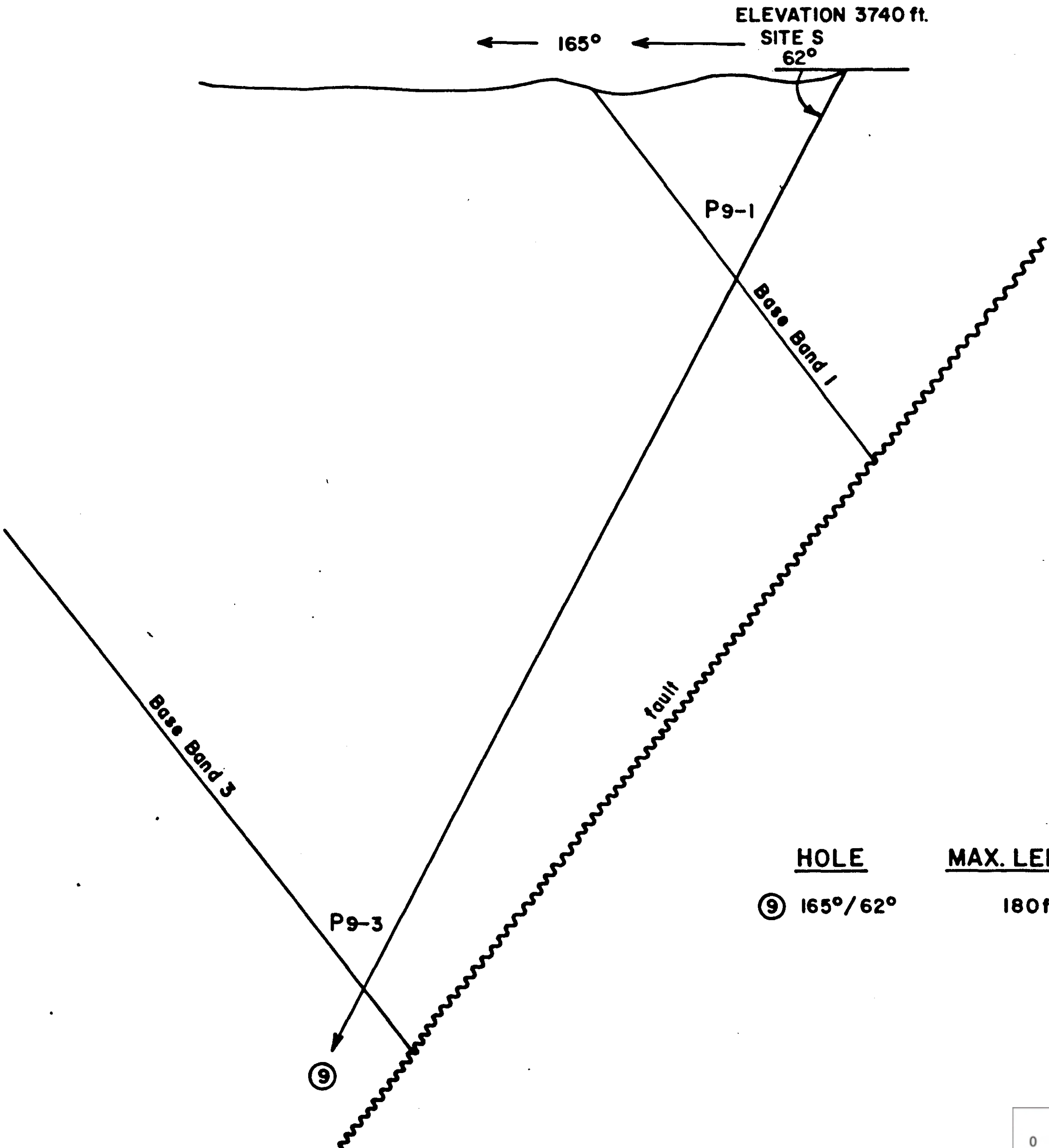


FIGURE 13

VERTICAL SECTION TRENDING 165° THROUGH SITE S

(SCALE 1cm = 10ft.)

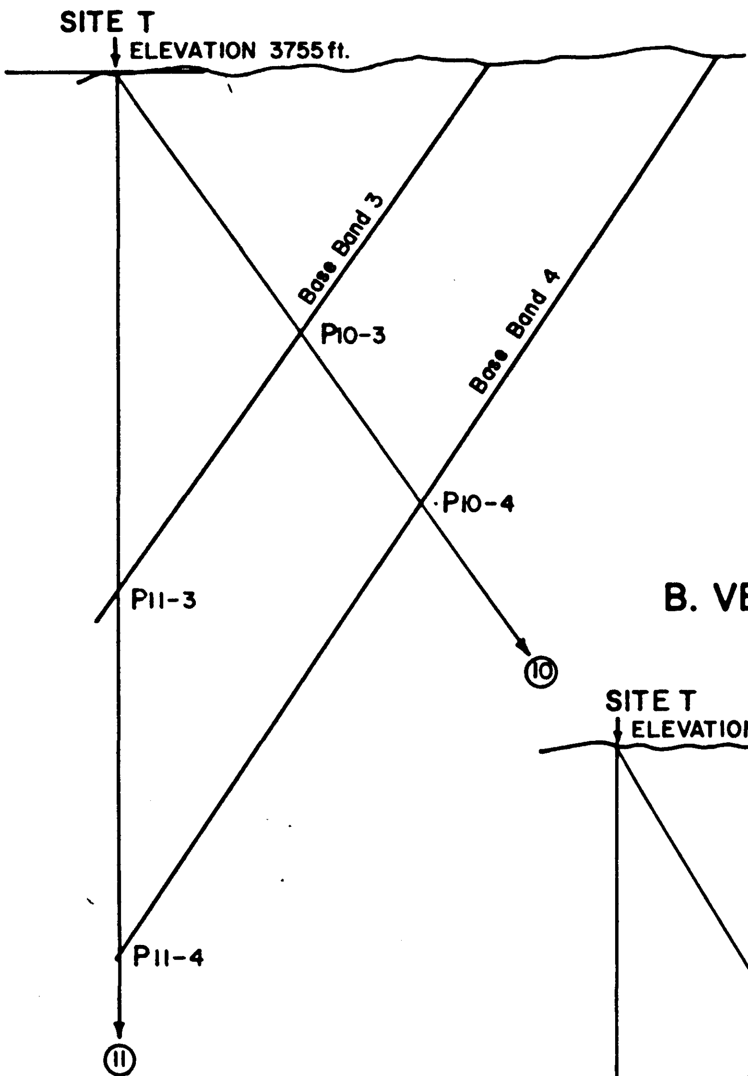


HOLE	MAX. LENGTH
9 165°/62°	180ft.

FIGURE 14

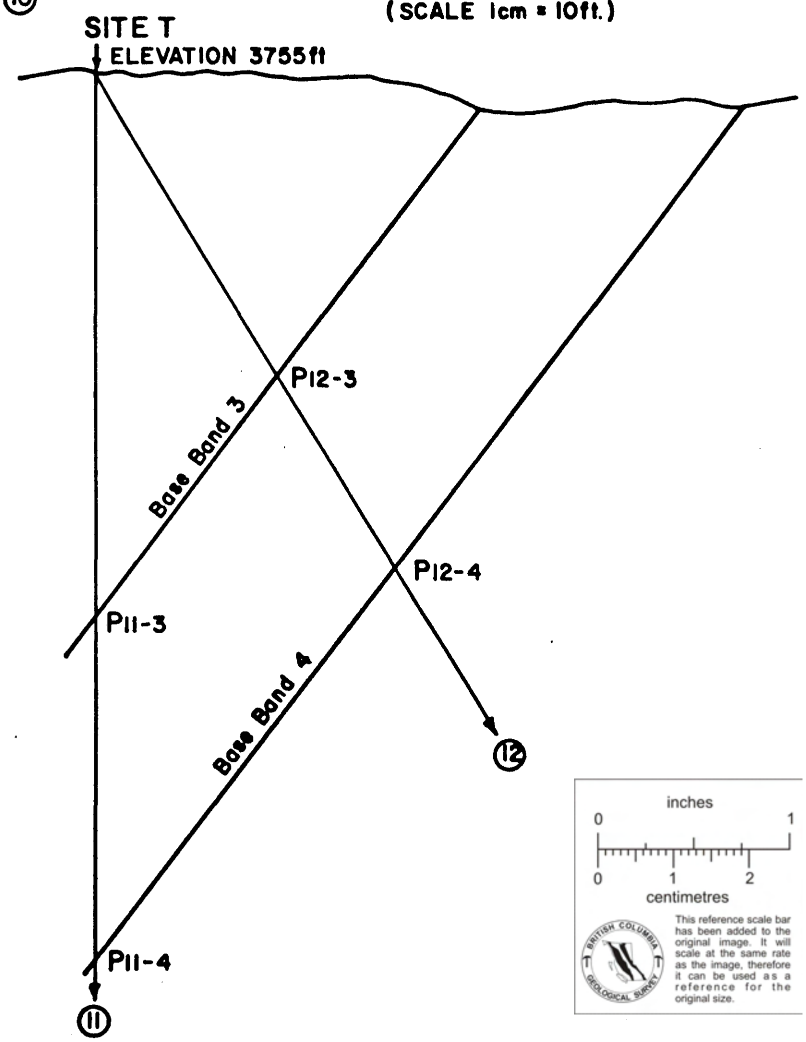
A. VERTICAL SECTION TRENDING 146° THROUGH SITE T

(SCALE 1cm = 10ft.)



B. VERTICAL SECTION TRENDING 180° THROUGH SITE T

(SCALE 1cm = 10ft.)



HOLE		MAX. LENGTH
10	146°/55°	100 ft.
11	Vertical	140 ft.
12	180°/60°	100 ft.

IX SUMMARY OF ORE TONNAGE POSSIBILITIES

Scheelite mineralization occurs in 6 areas which are listed below:

1) Skarn Band 1
2) Skarn Band 3
3) Skarn Band 4
4) Loose boulders in pit
5) Stockpile at Clearwater
6) Boulders below pit

1. Skarn Band 1 has a geologically probable tonnage of 2,763 tons at about 1%.
2. Skarn Band 3 has a geologically probable tonnage of 16,134 tons at about 1.5%
3. Skarn Band 4 has a geologically probable tonnage of 779 tons at 1.5%.
4. A 76 ft. lamping line through the pit indicated an average grade for loose material of 1%. The area of the pit is 3675 sq. ft., if a thickness of 3 ft. is assigned to the loose material then this provides 1002 tons at 1%.
5. The stock pile at Clearwater has previously been estimated at least 1500 tons. A grade estimate was made by lamping traverse lines across the pile at night and a value of about 1.5% obtained.
6. There is probably about 500 tons of better than 1% ore contained in boulders on the slope between Maxwell Creek and the pit. Unfortunately they can probably not be recovered.

Total reserves:

	2,763 tons	1.0%
	16,134 tons	1.5%
	779 tons	1.5%
	1,002 tons	1.0%
	1,500 tons	1.5%
equivalent to	20,923 tons	at 1.5%

It is difficult to attach a certainty to this figure. It has been derived by assuming a degree of predictability for grade and skarn thickness that may not be justified. The test of the assumption will be how well drill results fit the predictions made in the proposed drill hole logs.

X CONCLUDING REMARKS

The Gotcha mineral claims cover a small area of high grade scheelite mineralization. The mineralization occurs in skarns adjacent to an alaskite body which is partially surrounded by quartz monzonite. The grade of scheelite mineralization changes abruptly from outcrop to outcrop as does the specific type of skarn. It is difficult therefore to see much consistency in the surface geology and even more difficult to predict with any degree of certainty what the mineralization is doing at depth. Despite these problems an attempt has been made to estimate geologically probable ore reserves. A drill program has been outlined that should prove or disprove the existence of these reserves.

XI RECOMMENDATIONS

1. Arrange for a cat to open up two short access roads to three drill sites R, S, T.
2. At the same time, cut a trench 400 ft. long trending 320° from DDH 6 to expose skarn bands next to the projected intrusive contact.
3. Drill holes in the sequence suggested at each site. Spot new holes and/or shorten planned holes depending on amount and location of mineralization intersected. About 2000 ft. of drilling will probably be required to adequately prove out the tonnages outlined.

Respectfully submitted,

B. D. RYAN, B.Sc., Ph.D.

RESUME OF QUALIFICATIONS

B. D. RYAN, B.Sc. Hon., P.hD.

1967	B.Sc. Hon. Geology, U.B.C.
1967-1973	N.R.C. Graduate scholarship
1973	P.hd. Geology, U.B.C.
1973-1975	N.R.C. Post Doctoral Fellowship
1976-1977	U.B.C. Sessional Lecturer Geology
1977	Consultant
1978-1979	Research Associate, U.B.C.

During the period 1965 to 1973, I have had temporary employment with the G.S.C., Inco., Anaconda and Union Carbide.

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A SUMMARY REPORT
AND
COMPILATION OF DATA
GOTCHA TUNGSTEN PROPERTY
CLEARWATER, B. C.

UNITED MINERAL SERVICES LTD.

MAY 1979

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cont'd ....

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MAPS

ATTACHMENTS

SUMMARY

(1) The Gotcha Claim Group, owned by United Mineral Services Ltd., covers an area of tungsten mineralization which was discovered and preliminarily explored by Union Carbide of Canada Mining Ltd. in 1972-1973, further delineated by NCA Minerals Corporation during January 1978 and by United Mineral Services Ltd. during the summer of 1978.

(2) The claims are located approximately 20 miles northeast of Clearwater, B. C. and are serviced by a year round logging road.

(3) Two scheelite mineralized skarn bands, the Upper Band and the Lower Band, have been the focus of most exploration activity. A third scheelite mineralized skarn band was discovered during development stripping undertaken during the 1978 field season.

(4) Using exploration data obtained from work programs on the property tonnages of the Upper and Lower Bands can be stated as:

INDICATED TONNAGE

Upper Band	6000 tons @ 2.0%	or	12,000 s.t.u.
Lower Band	6100 tons @ $1.95%		11,900 s.t.u.

PROBABLE TONNAGE

Lower Band	900 tons @ 2.0%	or	1,800 s.t.u.

POSSIBLE TONNAGE

Lower Band	1100 tons @ 2.0%	or	2,200 s.t.u.

The total estimated reserves excluding float ore is 14,100 tons with an estimated total content of 27,900 s.t.u. WO_3.

(5) Further ore reserves are likely to be found on the property. Extensions for the Upper and Lower Bands exist both down-dip and to the southwest. Extension for the Lower Band also exists to the north-east. Additional ore may be found within the untested third scheelite skarn band that was discovered in 1978.

(6) Metallurgical testing has shown that the ore is ammenable to gravity or flotation concentration techniques. Recoveries in the 80% range can be expected.

(7) A program consisting of surveying, structural mapping and diamond drilling followed by feasibility studies is proposed.

INTRODUCTION

The Gotcha Claim Group is comprised of the Gotcha and Gotcha 2 Mineral Claims containing 1 and 9 units respectively. The Claims were staked by United Mineral Services Ltd. during March 1977 to cover an area of tungsten mineralization outlined and partially delineated in 1972 and 1973 by Union Carbide Canada Mining Ltd. During July 1977 United Mineral Services undertook limited trenching on the property and made a preliminary economic evaluation from data available.

In September 1977, United Mineral Services Ltd. entered into an option agreement with NCA Minerals Corporation. A percussion drilling program was carried out during January 1978 followed by a preliminary cost study by Mr. J. P. Elwell, P.Eng. on behalf of NCA Mineral Corporation. The option agreement between NCA Mineral Corporation and United Mineral Services Ltd. was terminated in March 1978.

United Mineral Services Ltd. undertook a program of development between the months of May and November, 1978. This program consisted of upgrading road access for heavy equipment use, stripping, rock trenching and preliminary metallurgical testing of two hundred tons of ore.

The results of work performed on the Gotcha Property are outlined in this report.

LOCATION AND ACCESS

The Gotcha and Gotcha 2 Mineral Claims are situated in the Kamloops Mining District with the main area of economic interest located approximately 30 meters above Maxwell Creek and 4.8 kilometres northwest of the confluence of Maxwell Creek with the Raft River. The property lies at the 4000 foot elevation at latitude 51°51.3' and longitude 119°42.1'.

Year round access to the property is by logging road which leaves the Yellowhead Highway (No. 5) 6.5 kilometres east of Clearwater and follows first the Raft River for 35.4 kilometres and then Maxwell Creek for the final 4.8 kilometres.

RESULTS OF PREVIOUS EXPLORATION

(A) During the summer of 1972, Union Carbide Exploration Corporation undertook a program of geological mapping, sampling and a total of 1769.3 feet of diamond drilling (diamond drill holes 1 to 8). This program was followed by an additional 1436 feet of diamond drilling during the 1973 field season (diamond drill holes 9 to 11). With the information available,Mr. D. L. Cook, P.Eng. postulated that there were two distinct scheelite bearing skarn bands present on the property with a tonnage possibility of 10,000 tons with a content of 15,000 short ton units of WO_3. Union Carbide's work is described in detail in a report by D. L. Cook, P.Eng. included in this report as Appendix A.

(B) During the summer of 1977, United Mineral Services Ltd. undertook a trenching program which substantiated the two band hypothesis presented by Mr. Cook and revealed that the Upper Band of skarn has sections which grade in excess of 3% WO_3. United Mineral Services Ltd. wrote a report entitled "A Geological Evaluation and Preliminary Economic Evaluation of the Gotcha Mineral Claim" using Union Carbide's data and the geological information obtained from their trenching program. With the information available at that time United Mineral Services Ltd. confirmed that at least two scheelite bearing skarn zones were present on the property and it was estimated that an overall tonnage of 9000 tons with a content of 14,000 stu of WO_3 was indicated. The report prepared by United Mineral Services Ltd. is included as Appendix B.

(C) During January 1978, NCA Minerals Corporation undertook a percussion drilling program in order to establish a greater degree of certainty for the tonnage and grade of portions of the scheelite bearing skarn in the Upper and Lower Bands. The results of this program are reported in "Progress Report - Exploration of the Gotcha Claims" by Mr. J. P. Elwell, P.Eng. and included as Appendix C. From the percussion drill hole results it was estimated that the Upper and Lower Bands contained 14,000 s.t.u. as drill indicated ore with an additional 3700 s.t.u. as probable and possible.

With the results obtained from the percussion drill program a Preliminary Cost Study was carried out by Mr. J. P. Elwell, P.Eng. for NCA Minerals Corporation in March 1978. He recommended that metallurgical tests should be made to determine the actual recovery and grade of concentrate that could be expected from production. He also recommended that development drifts be driven over both the Upper and Lower bands to open up the part of the ore zones which would be mined by underground methods and also provide access for further exploration of the mineral zones beyond the limits delineated by drilling. Mr. Elwell's report is included as Appendix D.

Following the reports of Mr. Elwell an independent review of the data available and an examination of the property was carried out by Mr. B. Hicks, P.Eng. of Brodie Hicks Engineering Ltd. The results of his findings are included as Appendix E. Mr. Hicks recommended that additional development should be carried out to up-grade the reserves and that a better understanding of the metallurgy of the ore be obtained.

(D) From the period of May 1978 to August 1978, United Mineral Services Ltd. had three separate metallurgical tests carried out by Bacon, Donaldson & Associates Ltd. as well as a semi-quantitative spectrographic analysis carried out by the Department of Mines and Petroleum Resources. The results of these tests are found as Appendix F.

The scheelite ore was found to respond well on bench scale tests using tabling and flotation techniques. Tabling of the ore was found to yield a 71% recovery and a 50% WO_3 concentrate. Flotation tests indicated a 80% recovery and a 11% to 36% WO_3 concentrate. A jig used in conjunction with a flotation circuit yielded a 16% concentrate and recovered 35% of the total 85.6% WO_3 recovered.

Semi-quantitative analysis of the ore showed that there are no deleterious impurities.

In order to obtain an indication of ore grades after dilution from mining and the response of the ore to a larger scale mill test, United Mineral Services Ltd. processed 200 tons of ore through a flotation mill located at Lumby, B. C. Results of this test are found in Appendix G. Head grades varied from .97% to 2.99% and it is reasonably justified to assume an overall average grade of 1.5% for the ore with an assumed dilution of 25%. Concentrate grades varied between 20.8 and 43.4% WO_3. Recoveries were low (approximately 50%) due to problems of controlling the grind.

The results of the tests show that the ore is easily ammenable to concentration by gravity or flotation methods.

Development work undertaken by United Mineral Services Ltd. during the 1978 field season established a greater understanding of the nature of the distribution of the scheelite mineralization within the skarn zones and the established grade of mineralization after dilution due to open pit mining methods. An overall grade of approximately 2% WO_3 can be assigned to the Upper Band and it appears that a tonnage greater than the original 3000 tons estimate given to the Upper Band can be reasoned. The work performed has also established additional areas in which ore can be obtained by open pit mining methods.

REGIONAL GEOLOGY AND STRUCTURE

The Gotcha and Gotcha 2 claims are located in an area in which metasedimentary rocks of the Shuswap Metamorphic Complex have been intruded by a variety of granitic dykes, stocks and sills. The metasedimentary assemblage consists of quartz-mica schist, garnet-mica schist, marble, muscovite-chlorite (biotite) schist, amphibolite, quartzite and metasedimentary gneisses. These rocks have undergone polyphase deformation and the metamorphic assemblage belongs to the upper amphibolite or hornblende hornfels facies.

Principal deformation and metamorphism of the Complex occurred in a time interval between Upper Triassic and Upper Jurassic. A general north to northwesterly trend of major and minor structures, including fold axes, lineations and compositional layering exists in the metasedimentary rocks in the northern portion of the Maxwell Creek area. A change to a predominately northeasterly trend of major and minor structures is found on the Gotcha Claim group. Large scale anticlinoria and synclinoria as well as smaller scale isoclinal folds and angular folds are recognized structural features of the Shuswap Metamorphic Complex and it is evident that such folding can be expected to be found within the Maxwell Creek area.

The metasedimentary rocks have a sequence that is lithologically similar to the Lower Cambrian Hamill quartzite - Badshot limestone succession and are tentatively assigned as correlatives of these formations.

Granitic rocks that intrude rocks of the Shuswap Metamorphic Complex include medium-grained biotite granodiorite, alaskite, quartz monzonite, quartz diorite and biotite granodiorite. Pegmatites represent a late - stage intrusive event and intrude all other granitic rocks. These intrusives have been assigned an Upper Cretaceous age and a K/Ar age date from an alaskite located in the Upper Skarn Band yielded an age of 64 m.y. (accuracy 3%) placing the time of intrusion on the Gotcha claims as Lower Tertiary. The emplacement of the intrusives within the metasedimentary rocks has resulted in the formation of contact metamorphic aureoles that contain large masses of tactite. It is within portions of these tactite zones that scheelite mineralization is found.

Faulting post-dates skarn and intrusive formation and these faults trend northeast and northwest and may be accompanied by strongly developed gouge zones.

A general summary of the geological events that occurred within the Maxwell Creek area are as follows:

(1) Lower Cambrian (?)
Deposition of a series of interbedded quartzites, limestones, and pelites.

(2) Upper Triassic to Upper Jurassic
Polyphase deformation and the formation of an amphibolite grade metamorphism of the sedimentary succession.

(3) Upper Cretaceous and Lower Tertiary
Intrusion of a varierty of intermediate to acid intrusive rocks.

(4) Lower Tertiary
The formation of tactite bodies within calcareous beds of the sedimentary succession.

(5) Post Lower Tertiary
Disruption of the lithologies by northeasterly and northwesterly low to high angle faulting.

GEOLOGY OF THE GOTCHA CLAIM

The work carried out on the Gotcha property since 1972 has established a series of northeasterly trending areas of metasedimentary rocks that occur as pendants with generally west to northwesterly dips. The area in which these pendants are found has been traced for approximately 400 metres (1200 feet) to the southwest of Maxwell Creek and the width of this area is approximately 200 metres (600 feet). At the southwesterly portion of this area the metasedimentary rocks are bounded on both sides by granitic rocks. The pendants lie within and are separated by intrusive rocks and are cut by numerous sills. Contact metamorphism has occurred along this northeasterly trend and a variety of contact metamorphic mineral assemblages have been produced. The calcareous rocks show stages of development from original marble to a coarsely crystalline quartz-garnet-epidote-vesuvianite rock.

Of the variety of skarn assemblages that occur on the property three important assemblages predominate.

(1) Massive garnet-quartz-epidote-vesuvianite skarn.
This skarn type consists of coarsely crystalline garnet, quartz and vesuvianite with varying amounts of accompanying epidote, sphene and apatite. This assemblage is widespread on the property as evidenced in both outcrop and diamond drill hole intercepts.

(2) Diopside-clinozoisite-tremolite-quartz skarn.
This skarn type is generally fine grained and can display a banded texture. It appears that this skarn type attains a continuity of composition and can be correlated in outcrop exposures and between diamond drill holes.

(3) Wollastonite-garnet-calcite skarn.

This skarn type is medium to coarse grained and appears to have a variable distribution throughout the property. The presence of wollastonite indicates formation at low pressure (1 to 2 kilobars) and high temperatures (500 to 700 degrees Centigrade) with the availability of SiO_2. Wollastonite-garnet-calcite skarn occurs along the northeasterly edge of the area of metasedimentary rocks.

Scheelite Mineralization

Tactite zones that have been delineated on the property are commonly composed of varying proportions of the three skarn assemblages described. The tactite has a widespread horizontal and vertical distribution as seen in both surface outcrop and diamond drill hole intercepts.

Within these tactite zones varying amounts of scheelite mineralization can be observed. Skarn types (1) and (2) host the most significant concentrations of scheelite while skarn type (3) has not been found to contain any appreciable amounts of scheelite. Skarn type (1) hosts pervasive late-stage silification that is accompanied by coarse grained scheelite. Quartz segregations frequently are noted to occur as irregular veinlike masses within and bordering the skarn. The quartz bodies yield no scheelite but nearby in other parts of the same zone scheelite may be concentrated. In areas of skarn type (1), abundant concentrations of scheelite are frequently found where quartz is abundant. The garnet-quartz-scheelite association of skarn type (1) appears to be the most productive and widespread skarn assemblage, however, the diopside-epidote-quartz-scheelite association of skarn type (2) can contain unusually high grade concentrations of scheelite as noted in the area of the Lower Band.

Within the tungsten-bearing zones of tactite there are areas in which no scheelite occurs, and the nature of the distribution of zones in which scheelite deposition occurred must be appreciated in the evaluation of the tactite zones.

In general it appears that the formation of the various skarn assemblages found on the Gotcha Claim has been in progressive stages. At an advanced stage of contact metamorphism, skarn types (1), (2) and (3) have been formed. The wollastonite stage of skarn formation has not been accompanied by tungsten deposition of any importance. The formation of the silicates of the garnet and epidote group when accompanied by abundant excess quartz represent a stage at which scheelite mineralization may be expected to form in greater abundance.

FORM OF SCHEELITE ZONES

The results from past exploration and development work have outlined two scheelite bearing zones that have been denoted as the Upper Band and the Lower Band. Scheeelite mineralization is found to dilate within the skarn assemblages to form irregular lensoid masses. The shapes of these mineralized zones are particularly well illustrated in the reports by Mr. J. P. Elwell, (Appendix D, Estimated Tonnage for the Lower Band) and Mr. D. Cook (Appendix A, Estimated Tonnage for the Upper Band, p.9). Determining the actual dimensions for the mineralized zones has been done on a basis of grade, mineable widths, and the degree of confidence that can be assigned to the width given at any particular point. It should be noted that in the evaluation by United Mineral Services Ltd. (Appendix B, Figure 3B) that an estimated tonnage of 3000 tons with a grade of 1% WO_3 was calculated for the Upper Band. The tonnage estimate for the Upper Band was calculated for two sections - a frontal block of 2000 tons and a rear block of 1000 tons. After stripping off the overburden covered area it was found that the surface topography was more pronounced than that depicted in the original estimate. This can be seen in the fact that from the 3692' elevation, the rear of the frontal block extends up for a distance of approximately 75 feet and is mineralized over a width of 20 feet. The grade of the slope is not as pronounced as originally depicted and the estimated tonnage given originally as 2000 tons can reasonably be justified as being in the neighbourhood of 5000 tons. As previously mentioned the grade of the material tested was approximately 2% before dilution.

The rear portion of the tonnage estimate for the Upper Band has been shown to constrict and pinch down to a width of approximately 2 feet, however, scheelite mineralization has been noted over a length of 14 feet (Appendix H - Diamond Drill Logs - D.D.H. 2).

From the exploration and development work done to date it has been shown that the scheelite bearing skarn zones have lensoidal geometry within larger zones of skarn assemblage minerals. The boundary to the limit of scheelite mineralization may be abrupt as in the case with the Lower Band or may be diffuse as in the case of the Upper Band.

POSSIBILITIES FOR ORE CONTINUATION

Tonnage calculations for the Upper Band have been made using the 3692 foot elevation as a cut-off. It is obvious from the results obtained from the percussion hole drilling performed in January 1978 that this is an arbitrary level and that scheelite mineralization is known to extend down at least to the 3676 level in the Lower Band. Extensions of the Lower Band are expected to be found to the northeast of the limited of percussion hole drilling and also at depth.

Extensions of both the Upper and Lower Band to the southwest are by no means eliminated and the possibilities of finding additional scheelite bearing skarn zones are good. This possibility has been shown by the discovery of an additional scheelite bearing skarn zone during the development work carried out during the 1978 field season. This zone is approximately 50 feet to the southeast of the Lower Band and is approximately 3 feet wide and contains greater than 2% WO_3 (visual estimate).

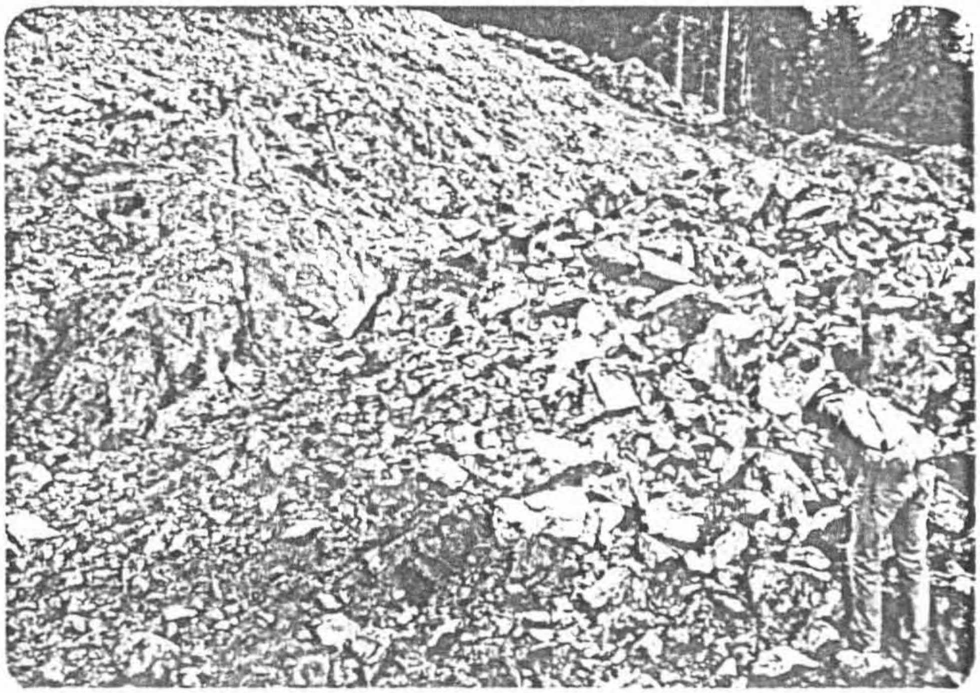
Diamond drilling performed by Union Carbide has indicated appreciable thicknesses of skarn assemblage minerals that are host to the scheelite mineralization in the Upper and Lower Bands. A more definitive geological model will help delineate those areas of the property where additional zones of scheelite mineralization can be expected to occur.

PROPOSED EXPLORATION PROGRAM

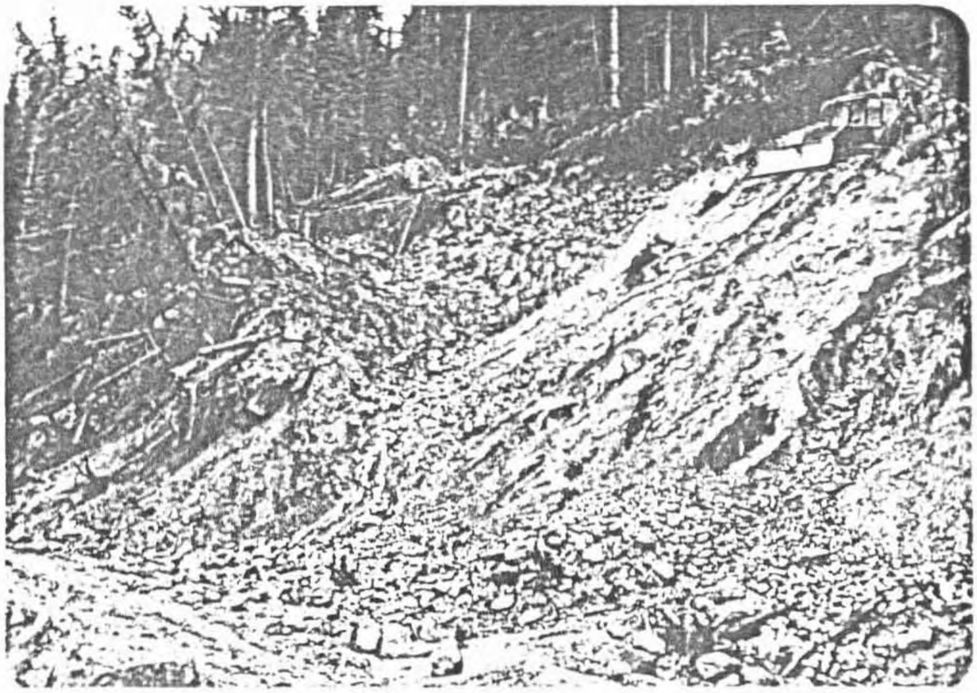
A program involving detailed geological mapping and surveying, taking into account the structural features that are evident on the property, should be undertaken. The area involved in this mapping program would focus on the geology between Diamond Drill Holes, 9, 10, 11, in the southwest, 4 and 8 in the southern portion and 2, 7, 5, 3, and 1 in the northern portion of the area. Upon completion of the mapping program and a revised structural interpretation of the geology, a drilling program should be undertaken. The drill program would be designed to more fully delineate the scheelite mineralization in the Upper and Lower Bands and test the geological possibilities that exist for the occurrences of additional zones of scheelite mineralization that may occur in the southwestern portion of the property. This recommended program is envisioned to entail approximately one month of geological field work followed by approximately 4000 feet of diamond drilling.

In addition, previous soil sampling has been able to locate areas of scheelite mineralization and it is suggested that a more widespread soil sampling program may be useful in delineating areas of scheelite mineralization that are overburden covered. Such a program would likely entail close spaced sampling. An initial survey over the known mineralized zones at different sample intervals would determine the optimim soil sample interval required. Care would have to be taken in determining the type of soil sampled in order to interpretate the results of such a survey.

A feasibility study should follow drilling to determine the approach to be taken to place the Gotcha property into commercial production.

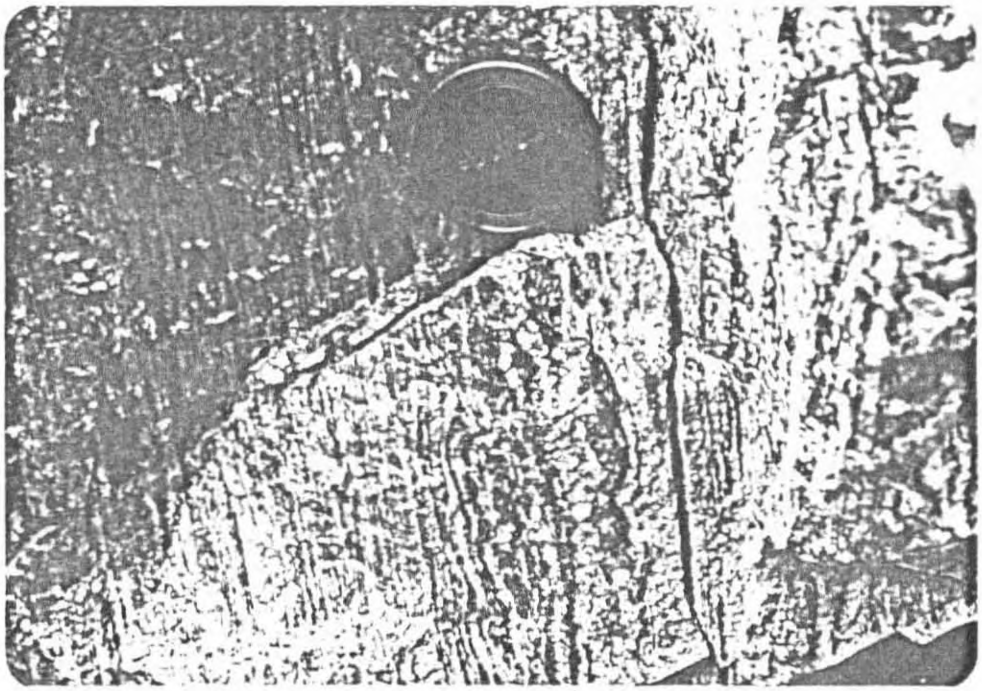


(1) View of Southwestern Portion of Open Pit - Upper Band

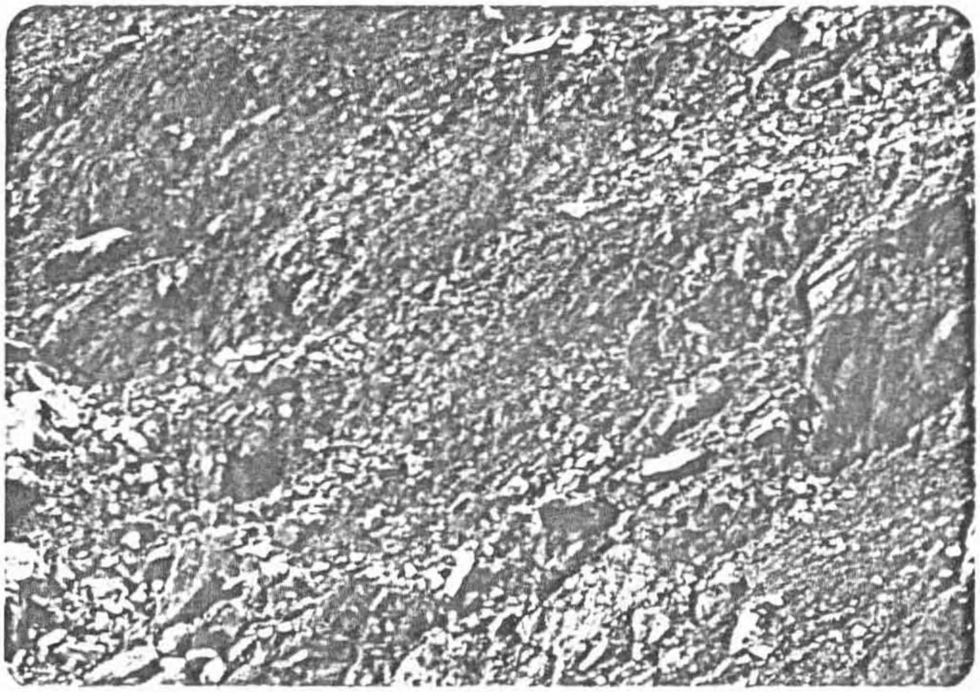


(2) Trace of Outcrop of Lower Band

(3) Coarse Scheelite in Quartz-Garnet-
Epidote-Vesuvianite Skarn - Upper Band



(4) Banded Dioside Skarn



(5) Hangingwall Fault Zone - Upper Band

APPENDIX A

82M/13E

GEOLOGICAL REPORT

ON THE

BOULDER CLAIM GROUP

KAMLOOPS MINING DIVISION

LATITUDE 51°50', LONGITUDE 119°41.5'

By

D. L. Cook, P. Eng.

For

UNION CARBIDE CANADA MINING LTD.

Work Completed During Period

August 1st, 1972 - November 13th, 1972.

March 30th, 1973.

CONTENTS

GEOLOGICAL REPORT

On The

BOULDER CLAIMS GROUP, KAMLOOPS MINING DIVISION, B. C.

INTRODUCTION

The Boulder Claim Group was staked by Union Carbide Exploration Corp., during July and August 1972 to cover what appeared to be the source area of scheelite encountered in down-stream panning and in boulders in Maxwell Creek.

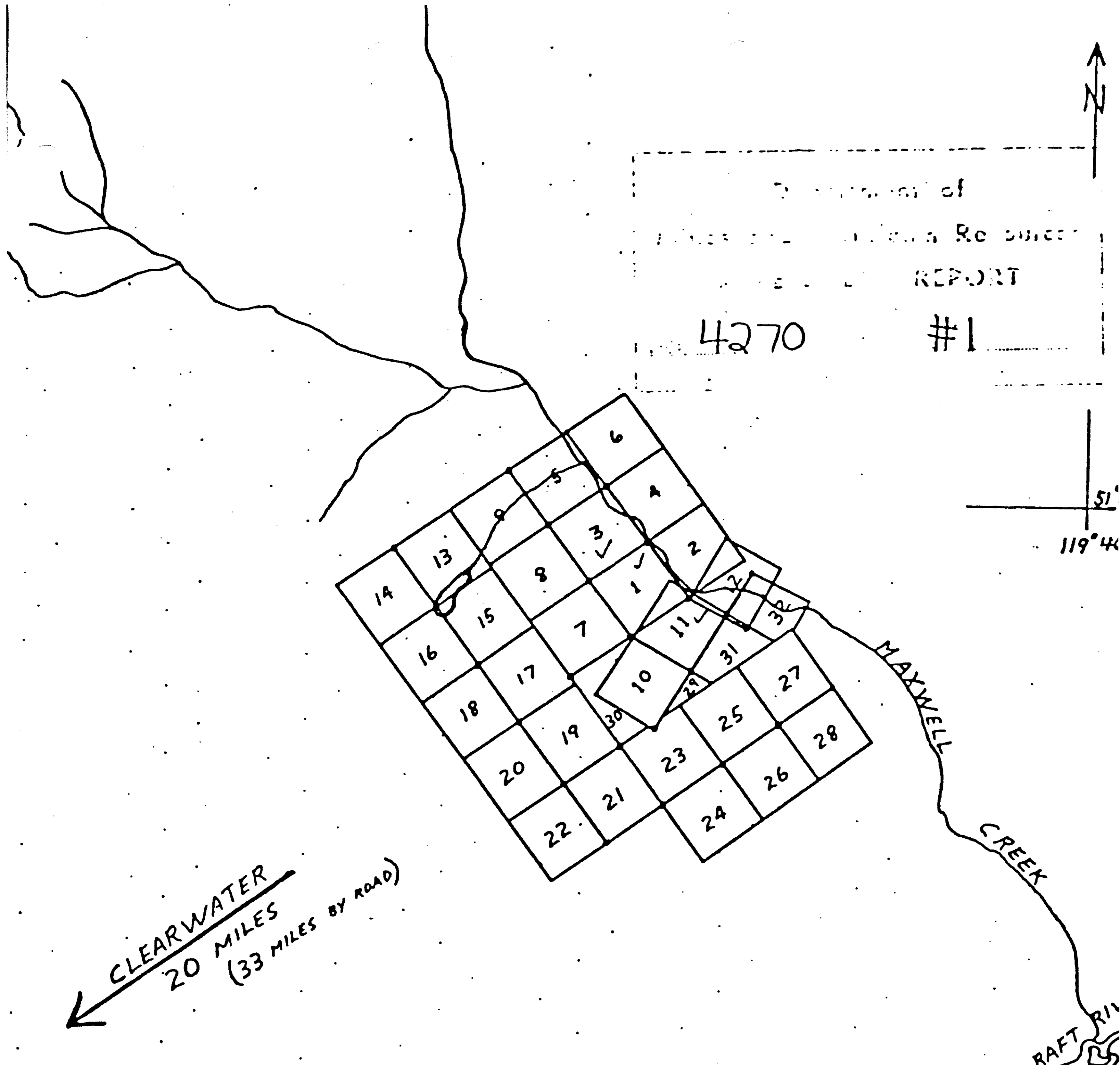
The present report outlines the results of the geological examination carried out on the property commencing August 1st, 1972.

Geological mapping, sampling and drilling were completed between August and November 1972 by B. D. Ryan and others under the supervision of D. L. Cook, P. Eng.

OWNERSHIP

The claims staked in the name of Union Carbide Exploration Corporation are as follows:

Name	Location Date	Recording Date	Record Number
Boulder 1 - 11	27 July 1972	1 August 1972	121089 - 121099
Boulder 12	1 August 1972	7 August 1972	121344
Boulder 13 - 22	19 August 1972	30 August 1972	121862 - 121871
Boulder 23 - 28	20 August 1972	30 August 1972	121872 - 121877
Boulder 29, 31 & 32	27 August 1972	30 August 1972	121878, 121880 & 121881
Boulder 30	26 August 1972	30 August 1972	121879



UNION CARBIDE EXPLORATION CORP.

LOCATION MAP

BOULDER CLAIMS

DRAWN BY: B.G.D.	DATE: APRIL 9. 1973
SCALE: 1" TO 5/9 MILE	1 : 36000

I.T.S. 82 M/13 E

All were recorded with the Mining Recorder for the Kamloops Mining Division at Kamloops. No other claims are held in the immediate area by any other company. (See accompanying claim map).

LOCATION

The claims are located 20 miles N. E. of Clearwater on the W. side of upper Maxwell Ck. and 80 miles N. of the town of Kamloops. They are in the Kamloops Mining Division and in map area 82M/13E of the National Topographic Series.

ACCESS

By logging road along Raft River and Maxwell Creek for 25 miles which leaves the Yellowhead Highway (No. 5), 4 miles E. of Clearwater.
A 'cat' road gives access to drill-sites. A helicopter pad has been cut in the bed of Maxwell Creek.

TOPOGRAPHY

The claims and surrounding area are heavily forested with steep, rounded, flat-topped hills up to about 6000' which is also the tree line. The only peak above the tree-line in the immediate area is Raft Mtn. (8040').
The flat hill-tops are swampy with numerous small lakes, conditions which will undoubtedly lead to problems in reconnaissance drainage sampling.

REGIONAL GEOLOGY AND STRUCTURE

The most 'detailed' mapping in this region was by R. B. Campbell in 1962 and 1963 (Adams Lake Map Sheet: G.S.C. No. 48-1963). However, this work is of a very general nature with extensive areas of rocks being undifferentiated both lithologically and structurally.
On and around the Boulder claims, the rocks are of the Shuswap Metamorphic Complex described, but not mapped, as consisting of the following rock assemblages:

1. Metasedimentary gneisses of varied type.

2. Amphibolite
3. Quartz-mica schist
4. Quartzite
5. Marble and Skarn
6. Pegmatite
7. Granitic rocks

Numbers 3 to 7 have been identified on the property.

Structurally, the Shuswap Metamorphic Complex is the heart of the core zone of the S. part of the E. Fold Belt of S. British Columbia. Rocks are generally in the upper amphibolite or hornblende-hornfels facies. The Complex is flanked on the N. by the Caribou Mountains Sub-province, on the E. by the Kootenay Arc (physiographically the Selkirk Mountains) and on the W. by the Intermontane Zone (Physiographically the interior plateau).

The metasedimentary rocks and schists are intruded by an enormous number of dykes, sills, and small irregular bodies of the granitic rocks. Only the larger of these granitic rocks have been mapped by Campbell, well to the S. of the Boulder claims. These are described as unfoliated or weakly foliated, mainly medium-grained biotite granodiorite. The granitic dykes and sills are described as mainly in the N. part, i.e., around the Boulder claims. The mapping on the Boulder claims, although limited, suggests one of these larger granitic bodies (unrecognized by Campbell) occurs in that area.

The pegmatites intrude all the other granitic rocks.

The metasedimentary gneiss contains a lower sequence that is generally similar in lithology, though not in detail, to the Lower Cambrian Hamill quartzite-Badshot limestone succession in the Kootenay Arc.

This lower quartzite-carbonate sequence remained resistant to metamorphism (i.e. relative to the more pelitic rocks) forming marbles and schistose quartzites. The metasedimentary rocks of the Boulder claims are thought to be of this succession.

The principal deformation and metamorphism of the Complex occurred in post-Late Triassic or Early Jurassic time. It began in the E. part of the Complex with intense metamorphism and migmatization accompanying large scale east-west trending interfolding of the core and mantle. Such folds permitted the local rise of migmatitic core synchronous with a northwesterly arching along the E. edge of the Complex, producing a series of gneiss domes at about 50 mile intervals. The final deformation consisted of warping and development of some N.W. trending folds.

PROPERTY GEOLOGY

In general, the area of claims drilled and geologically mapped (claims 1, 3, & 11) is a series of north to north-northeast trending pendants of west to northwest dipping metasediments, (marbles, skarns, quartzites and schists) lying in intrusive rocks (mainly leucocratic quartz monzonite, biotite quartz monzonite and pegmatite). The pendants merge and diverge both horizontally and vertically which complicate the interpretation.

The Metasediments

Marbles and Skarns: This succession of quite variable rocks, has a probable stratigraphic thickness of about 140' but is interupted by a number of thin beds of quartz-mica schist, quartzite and all variations between.

The variations within the skarn are thought to reflect differences in the lithology of the original limestone as composition within a given bed does not seem to change unless perhaps on a regional scale. This continuity of composition can be seen not only in outcrop but in a general way between drill-holes e.g. the banded diopside skarn bands seem to correlate in holes 1, 2 & 5.

If looked at on an even broader scale however, there does seem to be some variation along beds as the amount of wollastonite (previously identified on this property as tremolite) and calcite increases southward between the site of drill-hole No. 1 to a marble outcrop about 300' S. of the 'cat' road/logging road junction, a distance of about one-third of a mile.

Calcite as marble, or in skarn, with variable diopside, garnet and wollastonite, occurs only in the S. part of the mapped area. This is interpretted as an indication of decreasing reactive and additive solutions from N. to S. This gradient does not correspond to distance from intrusive contact as the intrusive is common throughout the mapped area. The suggestion is that solutions (of a reactive and additive nature) originated from the intrusive somewhere to the N. possibly in the vicinity of the scheelite mineralization.

The skarn with high incidence of wollastonite indicates a high level of silica, possibly in the original limestone as detrital quartz, or derived from the intrusive.

The very general correlation between the incidence of both calcite and wollastonite suggest the latter may be a function of distance from the source of reactive and additive solutions and therefore has derived its silica from this source. Vein and patchy quartz obviously from pervading solutions, is common. This is thought to be post-wollastonite. The other skarn types have been recognized on the property; one predominantly coarse and with dominant mineral as garnet; the other predominantly fine and banded, with dominant mineral as diopside. Both types however, have the minerals of the other, as well as variable content of idocrase, quartz and occasional wollastonite and scheelite; the latter up to 2.60% WO_3 over stratigraphic thickness up to 12.3'.

Quartzites and Schists: These are the lithological end members of a gradational series of rock types varying from fine-grained 'clean' quartzite through biotite quartzite, quartzitic biotite schist to biotite-rich quartz schist.

These metasedimentary rocks are thought to be equivalent to the Lower Cambrian Hamill quartzite-Badshot limestone succession in the Kootenay Arc.

Intrusive Rocks:

The main intrusive rocks are fine to coarse-grained leucocratic quartz monzonite with variable content of muscovite; fine to coarse-grained biotite quartz monzonite; minor amounts of biotite quartz diorite and biotite granodiorite; muscovite, quartz, feldspar pegmatite.

The leucocratic quartz monzonite and the biotite granitics have been seen in sharp contact with each other and in one case the first appeared to be digesting and therefore intruding the second.

It is suggested that the biotite-rich granitics are the outer phase or phases of the migmatizing intrusive with the leucocratic quartz monzonite being a later phase which has breached the earlier phase. However the evidence for this is recognized as limited

The contacts of the intrusive with the metasediments are usually parallel to the compositional banding but may be irregular or in rare cases show digestion and/or partial stoping of the sediments.

The intrusive then has apparently invaded the stratified rocks primarily along their bedding producing an irregular contact appearing in section as a myriad of narrow apophyses. Most of the stoping and digestion of the stratified rocks has then occurred at the advancing front of these apophyses with a sharp and relatively non-reactive contact paralleling the beds away from the reactive loci. This explanation would explain why drilling cut so many narrow intersections of intrusive and rarely cut the reactive loci of stoping and digestion.

Muscovite, quartz, feldspar, pegmatite in irregular masses and as dykes or veins are seen pervading all the granitic rocks.

STRUCTURE OF THE PROPERTY

What little is known of the structural geology of the region has been described above. Just how the area of the Boulder claims fits into this overall picture is not very clear.

The only structural feature mapped by Campbell in the claim area is a strike and dip of beds at 329/80N.E. This is not consistent with our own mapping on the claims where strikes and dips are mainly NE/35-70 S.W. in the north, south and east parts and W-E/40-75N in the western part.

Obviously we have not mapped enough area in order to understand the relationship of the claim area to the regional geology.

Considerable evidence for faulting has been encountered in drill-holes as well as some lesser evidence on the surface. Because of the frequency of faulting and poor outcrop it has not been possible (with two exceptions) to correlate between holes and outcrop in order to determine the aspect of the faulting beyond one dimension, or their relative displacement. The exceptions occur at drill-holes 1 and 3, (see geological map and sections).

SCHEELITE MINERALIZATION

Scheelite mineralization in place was seen at two locations on surface and in the core as follows:

	Location	Grade (% WO_3	Rock-type
DDH 2.	124.5' - 126.5'	1.07]	Coarse garnet, quartz idocrase diopside skarn
"	At 128.5'	Minor]	
"	At 129.5'	Minor]	
"	At 138.5'	>1]	
DDH 3.	15' - 19'	2.06	Fine-grained banded diopside, quartz skarn.
"	19' - 27.3'	2.86	Medium to coarse-grained garnet, quartz diopside skarn.

Location	Grade % WO_3	Rock-type
DDH 3. 27.3' - 32'	0.19	Fine-grained unbanded diopside quartz skarn and medium to coa grained garnet, diopside quart skarn.
DDH 5. At 70'	Minor	Limey, coarse garnet idocrase diopside skarn.
" 116.3' - 116.5'	> 0.5	Coarse quartz, garnet diopside, idocrase skarn.
Outcrop near DDH 1.	> 1	Coarse garnet quartz idocrase diopside skarn.
Outcrop near DDH 3.	> 1	Fine-grained banded diopside sk

Of the most significant mineralization, most of it occurs within the coarse garnet quartz, idocrase, diopside skarn. The exception is a band of fine-grained, banded, diopside, quartz skarn intersected in hole 3 and outcropping near the collar of hole 3. This fine-grained band overlies the mineralized coarse-grained skarn.

The present interpretation of the geology (see sections) indicates that the known mineralization is closed off by intrusive down the dip of beds in the area of holes 2, 3 and 5. To the N. and E. the depressed topography would appear to limit the mineralization in that direction. However, elsewhere (i.e. S. and E.) there remains untested ground between drill holes where there may be a shoot or shoots of ore extending away from the known mineralization.

Faulting will undoubtedly compound the problem of locating these possible extensions.

The first question to be asked is whether the intersections and outcrop of scheelite-bearing skarn are of one or more continuous horizons.

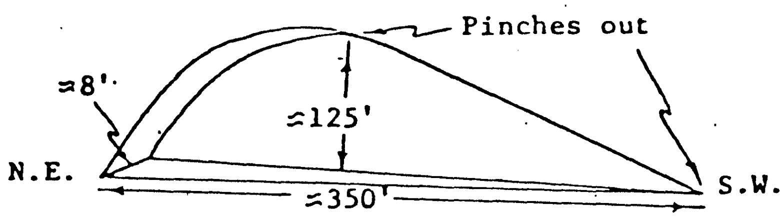
The confinement of the scheelite to one or possibly two units would help to establish the continuity of mineralization and improve the outlook for extensions. If no pattern of continuity can be seen between drill-holes and outcrop, then patchy mineralization (or continuous mineralization offset by faulting) would be suggested.

When examining a three dimentional model built up from drill-hole sections and the geological map, the mineralization in hole 2 and near hole 1 seem to correlate, but that in and near hole 3 does not.

Either the block of ground between holes 1 and 2 has been offset by faulting (and faulting is known to be quite common in the area drilled) or there are two mineralized horizons represented.

The pattern of scheelite-bearing boulders shows two distinct lines which when related to the outcrop at holes 1 and 3 strongly suggest two separate bands of mineralization.

Assuming two distinct bands, then the upper band (intersected by hole 2 and outcropping near hole 1) would appear to pinch out towards hole 2 as it is about 8' thick near hole 1, but only 2' thick in hole 2. It is further closed off by hole 5, Maxwell Creek, and at depth, probably by the intrusive. Thus this mineralized band may take the following form:



i.e. 5000 stu of 1% WO_3

Possibilities for extension exist to the S.W. where there is always the possibility of the body dilating again. This ground has not been tested.

The lower band (intersected by hole 3 and possibly represented by the E. line of boulders) would appear to be closed off by holes 2, 5 and 7, the intrusive down dip, and the depressed land surface at Maxwell Creek. This mineralized band would therefore probably have a similar volume to the upper band but about twice the grade, i.e. ≃10,000 stu. if of 2% WO_3.
Total possible reserves for the two bands might therefore be 15,000 stu. of between 1 and 2 % WO_3.

RECOMMENDATIONS

The possibilities for ore continuation are as follows:

1. Down dip extensions: Although the intrusive appears to close off down dip extensions of 'ore', it has nevertheless been seen to have a very irregular contact with its intruded rocks and significantly deeper extensions of the metasedimentary pendants containing scheelite may occur.
2. Being in the Shuswap Complex where structures are known to be complex, the same lithological setting may be repeated in any direction, not only on the property, but in the surrounding area. The most obvious area of interest in this regard is the on-strike continuation of the mineralized bands in metasedimentary pendants which may occur N.E. of Maxwell Creek.
3. The possibility of two distinct mineralized bands has been suggested. This can be tested by further drilling: See below.

The following recommendations are made:

1. Complete the following drilling at -45° declination, a total of about 1500' of drilling.

(a) One hole near the collar of hole 5 to test for the intrusive contact and a possible extension of the suggested lower band of mineralized skarn.

(b) One hole between the collars of holes 1 and 5 or 1 and 3 to test for the continuation of mineralization found in hole 2 and in outcrop at hole 1. While realizing that this hole will not add significantly to tonnage even if it proves the continuation of mineralization between holes 3 and hole 1 outcrop, it would reveal something of the degree of persistance of mineralization which may be expected elsewhere.

(c) One hole 250' S.W. of holes 2 and 7 to test for the possible extension of the mineralization found in hole 2.

(d) One hole between 150' and 300' W.S.W., of hole 4 to test the ground in that area. If skarn is encountered in this hole then further holes to the W. should be planned.

2. Carry out the following with the purpose of finding a similar mineralized situation or situations to that already known.

(a) The panning of soils in the area of the known mineralization detected the mineralization, but only within 100' to 200' of it. It is recommended then that panning on say a 50' grid should be carried out over the extent of the claim area.

(b) Based on the results of this panning, further 'cat' trenching should be undertaken.

(c) The outcrop exposed by this trenching should be mapped and further drilling may be proposed.

3. The region surrounding the Boulder claims should be thoroughly prospected and mapped. This should include the Raft Mountain occurrence 10 miles to the S.W.

683-6451

Brian Norris

BOULDER CLAIM GROUP - ASSESSMENT COST, 1972.

Personnel

D. L. Cook, P. Eng., Field Examinations (Sept. 30, Oct. 1 & 28, 1972) Map Preparation (1 day) Report Preparation (1 day) Logging Drill Core (8 days) 14 days @ $40.	$ 560.00
B. D. Ryan, Geologist, Field Examination (August 1-4, 27-31, Sept. 1, 1972) Map Preparation (3 days) Report Preparation (1 day) 13 days @ $35.	455.00
H. Abendroth, Geologist, Field Examinations and Short-hole percussion drilling (Aug. 4 - 25, 1972) Diamond Drilling Supervision (Oct. 13 - 30, 1972) 40 days @ $35.	1,400.00
B. Dimitroff, Geologist, Diamond Drill Supervision (Oct. 17-19, 1972) 3 days @ $35.	105.00
P. Burt, Senior Field Assistant, Field Examinations (August 24 - 29, 1972) Diamond Drill Supervision (Sept. 7 - 15, 1972 and Oct. 31 - Nov. 21, 1972). 37 days @ $30.	1,110.00
D. Oatway, Senior Field Assistant, Map Preparation (1 day) Field Examinations and short-hole percussion drilling (August 4 - 29, 1972) 27 days @ $30.	810.00
Eight Field Assistants for a total of 107 man days between August 7 and Sept. 30, 1972. 107 days @ $30.	3,210.00

633-2651

APPENDIX B

A

GEOLOGICAL EVALUATION

AND

PRELIMINARY ECONOMIC EVALUATION

OF THE

GOTCHA 2 MINERAL CLAIM

BY

UNITED MINERAL SERVICES LTD.

MAY, 1977

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PLATES

FIGURES

APPENDIX

GEOLOGICAL EVALUATION OF GOTCHA 2 CLAIM

1. INTRODUCTION

The Gotcha 2 mineral claim was staked by United Mineral Services Ltd. during March 1977 to cover an area of tungsten mineralization, outlined and partially delineated in 1972 and 1973 by Union Carbide Canada Mining Ltd.

This report outlines and interprets the results of geological examinations carried out on the property during 1972 and 1973 by B. Ryan and B. Norris under the supervision of D.L. Cook. Personal communication with B. Ryan and B. Norris and other Union Carbide personnel concerning the geology and exploration of the property along with study of their maps and drill logs has enabled a satisfactory interpretation of the geology of the property and its economic potential. Union Carbide's data is included in this report as Appendix A.

2. LOCATION

The claim is located 20 miles northeast of Clearwater on the west side of upper Maxwell Creek and 80 miles north of the town of Kamloops.

Access to the claim is by a year-round logging road which leaves the Yellowhead Highway (No. 5) 4 miles east of Clearwater and follows alongside Raft River and Maxwell Creek for 25 miles.

3. PREVIOUS EXPLORATION

The Gotcha 2 mineral claim was originally staked by Union Carbide Exploration Corporation as the Boulder Claim Group in July and August, 1972. During the 1972 field season geological mapping, sampling and a total of 1769.3 feet of diamond drilling was completed in diamond holes number 1 through 8.

During the 1973 field season a total of 1,436 feet of diamond drilling was completed in diamond holes number 9 through 11. The results of the mapping and diamond drilling were not encouraging for the discovery of the large size of deposit in which Union Carbide Canada Mining Ltd. would be interested in; hence the Boulder claims were subsequently allowed to lapse.

4. PROPERTY GEOLOGY

The Gotcha 2 mineral claim is underlain by rocks of the Shuswap Metamorphic Complex and granitic rocks of Mesozoic age (see Figure 1).

SUMMARY

(1) The Gotcha 2 Claim, owned by United Mineral Services Ltd., covers an area of tungsten mineralization outlined and partially delineated by Union Carbide Canada Mining Ltd. in 1972-1973.

(2) The Claim is located 20 miles northeast of Clearwater, B.C.

(3) Coarse grained scheelite is found within coarse garnet-diopside skarn and fine banded diopside skarn.

(4) Two scheelite mineralized skarn bands exist and at least one other is possible.

(5) Using Union Carbide drill and geological data, tonnages of the Upper and Lower Bands can be stated as:

Upper Band	2,000 tons @ 1% WO_3	INDICATED ORE
	1,000 tons @ 1% WO_3	INFERRED ORE
Lower Band	900 tons @ 1.9% WO_3	INDICATED ORE
	5,100 tons @ 1.9% WO_3	INFERRED ORE
	6,500 tons @ $>$.5% WO_3	POSSIBLE ORE

(6) About 1,700 feet of drilling would move most of this ore into a measured classification.

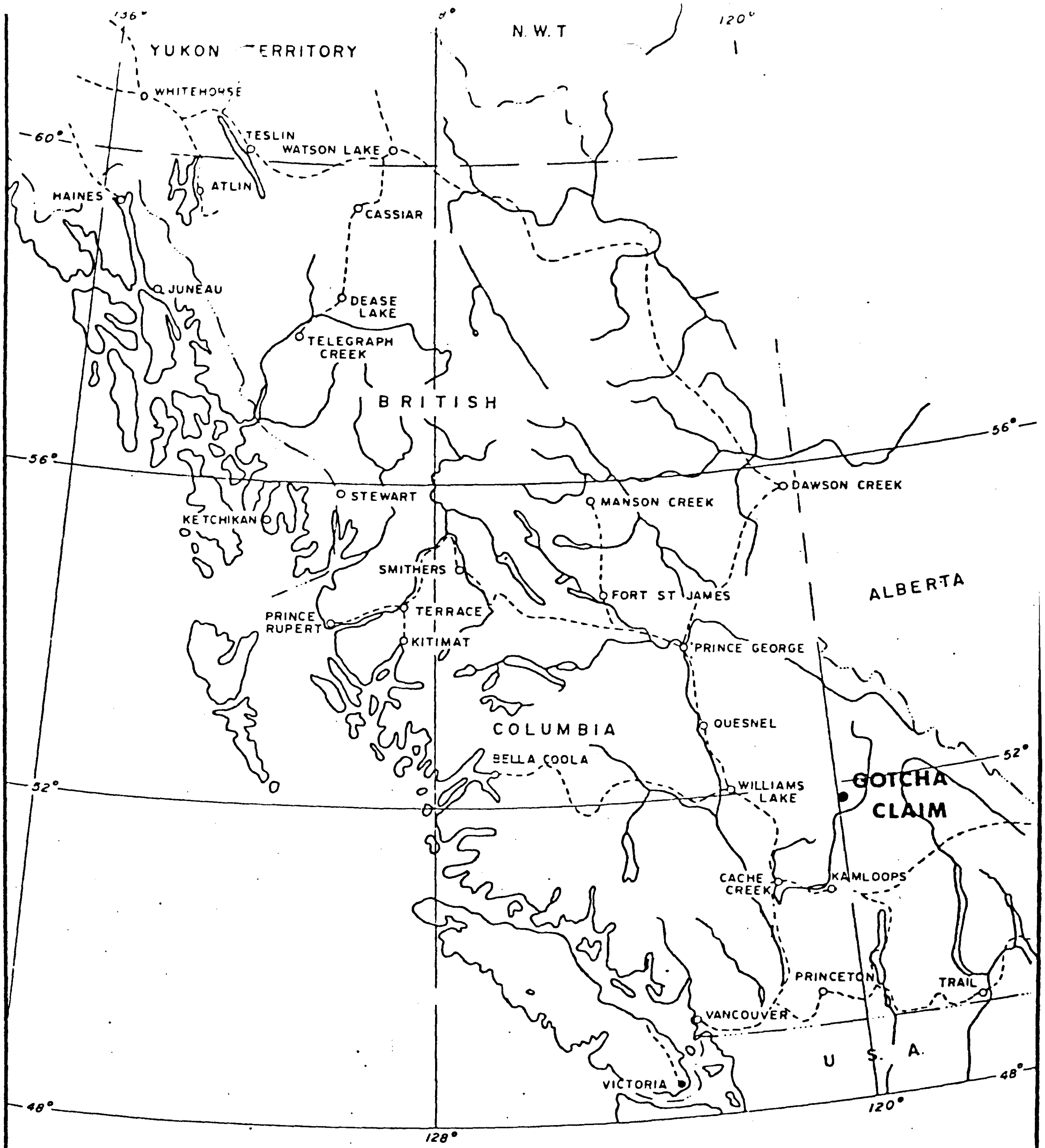
(7) This exploration would cost about $29,000.

(8) If all ore is proven it could be mined/milled within one year using a 50-75 ton per day gravity milling complex.

(9) The capital cost of milling assets is calculated to be $100,000.

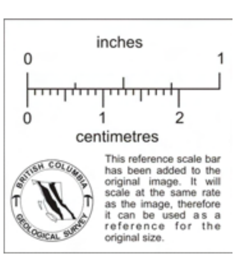
(10) Calculations suggest an after tax profit of $500,000-$600,000+ if all ore is proven.

(11) Working capital required in the early stages of operation would be in the order of $200,000.



UNITED MINERAL SERVICES LTD.

GENERAL LOCATION SKETCH



SCALE: 1" = 125 MILES

PLATE 1

The metamorphic rocks exposed consist of an assemblage of metasedimentary gneisses, quartzite, marble and skarn and minor amphibolite. These rocks have been intruded by dykes, sills and small irregular bodies of granitic rocks. Intrusion and emplacement of the granitic rocks appears to be in large part controlled by structures in the metamorphic rocks.

Generally, the area that has been mapped by Union Carbide consists of a series of north to north-northeast trending pendants of west to northwest dipping metasediments lying in intrusive rocks (see Figure 2; Figure 2A). The metasedimentary rocks within the pendants have been overturned to the northwest and plunge at low angles to the northeast (see Figure 1; Appendix A - Isoclinal Fold Hypothesis in Diamond Drill Hole Number 10). It is apparent that the metasediments have been folded into overturned isoclinal folds that have been intruded by granitic rocks along their core areas.

Possible faulting and internal folding of the metasediments as well as intrusion of granitic rocks hinder the interpretation of the continuity of assemblages within the pendants.

5. SKARN ASSEMBLAGES AND SCHEELITE MINERALIZATION

An economically important marble and skarn succession of quite variable lithologies has a probable stratigraphic thickness of 40 plus feet and is interrupted by thin beds of quartz - mica schist and quartzite (see Appendix A - Diamond Drill Hole Number 10).

The continuity of the composition of the skarn on the local scale is good; that is, correlation of different skarn bands can be made between holes 2 and 3 and 5 and surface outcrops. On a broader scale the skarn assemblages are predominately localized to the area of interest (see Figure 1; Appendix A - Geology of Boulder Claims) and give way to marble in a southerly direction.

Three skarn assemblages predominate and consist of the following:

(a) Calcite - wollastonite (tremolite) skarn.

(b) Coarse garnet-diopside skarn.

(c) Fine-banded diopside skarn.

Scheelite is found associated with assemblages (b) and (c) and the most significant mineralization is usually found within coarse garnet-diopside skarn and occurs as grains (1mm - 5mm in diameter) and porphyroblasts (greater than 5mm in diameter).

6. ESTIMATES OF ORE TONNAGE

Two scheelite mineralized skarn bands, the Upper Band and the Lower Band, have been outlined by Union Carbide's field work.

The volumes of these bands can be reasoned from data available to date. Union Carbide suggests "Total possible reserves for the two bands might be 15,000 STU of between 1 and 2% WO_3" (see Appendix A). A more definitive working model can be reasoned as follows:

UPPER BAND

Figure 3A - Plan of Geology at the 3692 foot level indicates that the Upper Band is partially digested by the intrusive approximately 100 feet from its 3692 foot surface exposure. D.D.H. No. 2 intersected the Upper Band however at the 3692 foot level, indicating continuity of the Upper Band for 200 feet on the 3692 foot level. Drill hole No. 5 intersected the upward continuation of the Upper Band at 22.5 feet, displaying that the Upper Band has good overall continuity. ?

Scheelite mineralization within the Upper Band outcrops near the collar of D.D.H. No. 1 and appears to pinch out towards D.D.H. No. 2 and is further closed off by D.D.H. No. 5. The Upper Band is also closed off at depth by the intrusive. Possibilities for extension exist to the southwest where there is the possibility of the Upper Band expanding again.

The total mineralized volume that can be outlined in the Upper Band has been determined by the length of the surface expression of the mineralized band (as indicated by geochemical results, and the mineralized boulder train, see Figures 4 and 5); the distance to the 3692 foot elevation intersection in D.D.H. No. 2 and the distance to the surface expression from D.D.H. No. 2 (see Figure 3). The dimensions of this volume are shown on Figure 3B.

The total volume in this prism is approximately 30,000 cubic feet or 3,000 tons (1 ton equals 10 cu. ft.) of ore with an estimated grade of 1% WO_3. Of this 3,000 tons it is reasonable to classify approximately 2,000 tons as indicated ore and 1,000 tons as inferred ore (see Appendix B - Classification of Ore).

LOWER BAND

The plan of geology at the 3692 foot level indicate that the Lower Band can only extend along strike for approximately 250 feet before it is replaced by intrusive. The Lower Band, however, has been intersected in D.D.H. No. 3 and the down-dip extension has been intersected in D.D.H. No. 5.

The total mineralized area that can be outlined in the Lower Band has been determined by the length of the surface expression of the mineralized band (see Figures 4 and 5), the distance to the intersection in D.D.H. No. 5 and the distance to the surface expression from D.D.H. No. 5 (see Figure 3). The volume outlined in this wedge has been broken down into two parts (see Figures 3C and 3D). Block 1 approximates 6,000 tons of which 5,100 tons is considered inferred tonnage and 900 tons is considered to be indicated tonnage each with an estimated grade of 1.9% WO_3.

Block 2 is a possible extension of the mineralized Lower Band and contains a possible 6,500 tons. The dimensions of this particular block do not have a reasonable degree of confidence at present due to lack of data. For example, the down-dip extension in D.D.H. No. 5 is less than .5 feet consisting of greater than .5% WO_3, hence there must be considerable dilation of the Lower Band in this area and this has not been taken into account. Block 2 does point out however that for a small horizontal advance from the end of Block 1 a large volume can be attained.

In summary the volumes outlined by the Upper and Lower Bands can be stated as follows:

Upper Band		2,000 tons @ 1% WO_3	Indicated
		1,000 tons @ 1% WO_3	Inferred
Lower Band	Block 1	900 tons @ 1.9% WO_3	Indicated
		5,100 tons @ 1.9% WO_3	Inferred
	Block 2	6,500 tons @ greater than .5%	Possible

7. EXPLORATION

A. DIAMOND DRILLING

It is proposed that 6 diamond drill holes be put down to test the continuity and variations in grade of the Upper and Lower Bands. If the results of the initial diamond drill program prove satisfactory then a more detailed survey by X-Ray diamond drilling will be required to confirm continuity.

Total length of the initial diamond drilling would be approximately 700 feet. X-Ray drilling would be dependant on the results of the initial diamond drilling and would be a maximum of 1,000 feet.

B. FURTHER PROPERTY EXAMINATION

The possibility of finding additional mineralized areas similar to that already known exist in the southeastern portion of the area of interest (see Figure 1 and possible band in Figure 4). This area could be examined in the following manner.

(a) A 50 foot grid over the area of possible mineralization should be soil sampled for scheelite content.

(b) Based on the results of this panning, cat trenching should be undertaken.

(c) The outcrop exposed by this trenching should be mapped and further drilling may be proposed.

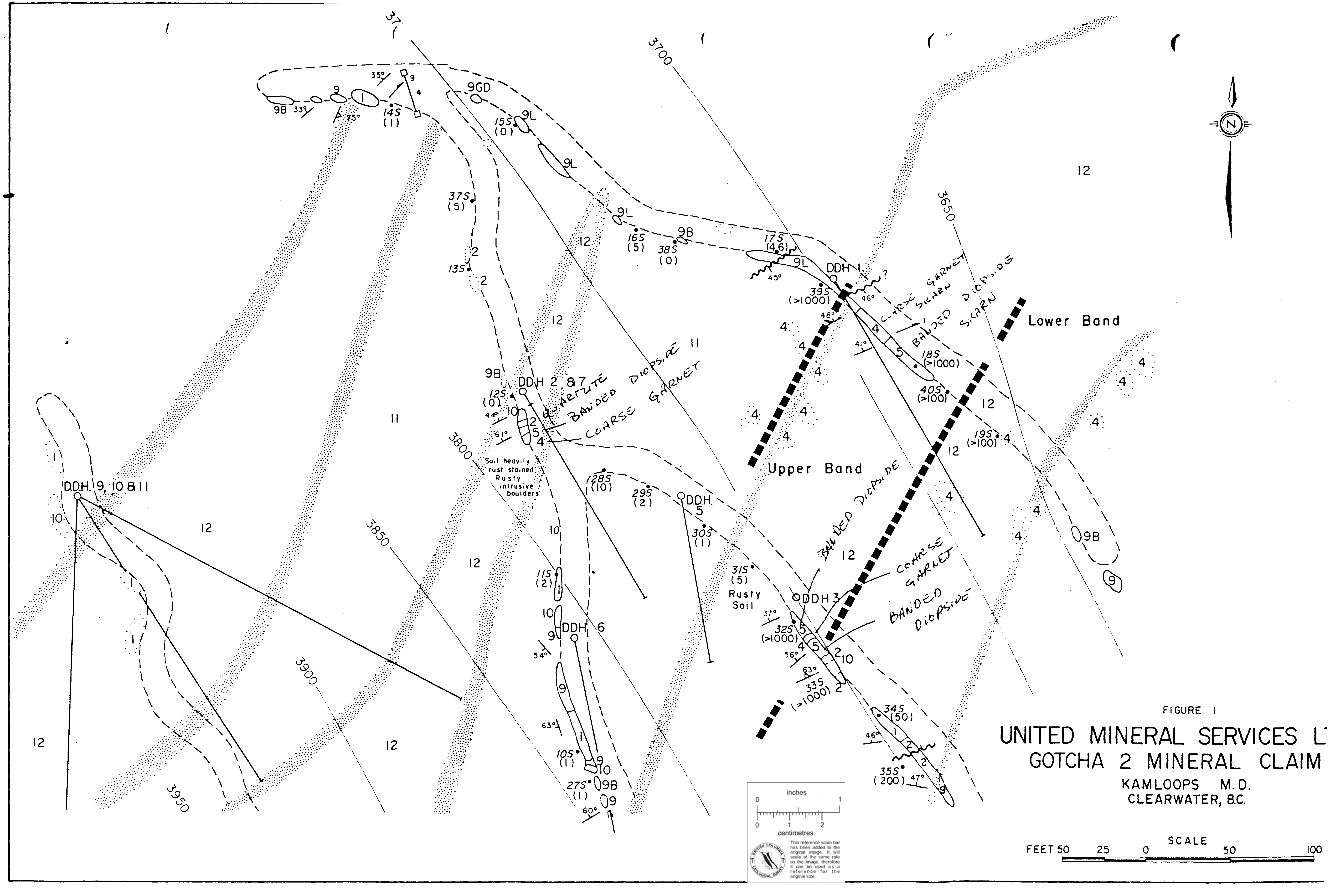


FIGURE I

UNITED MINERAL SERVICES LT
GOTCHA 2 MINERAL CLAIM
KAMLOOPS M.D.
CLEARWATER, B.C.

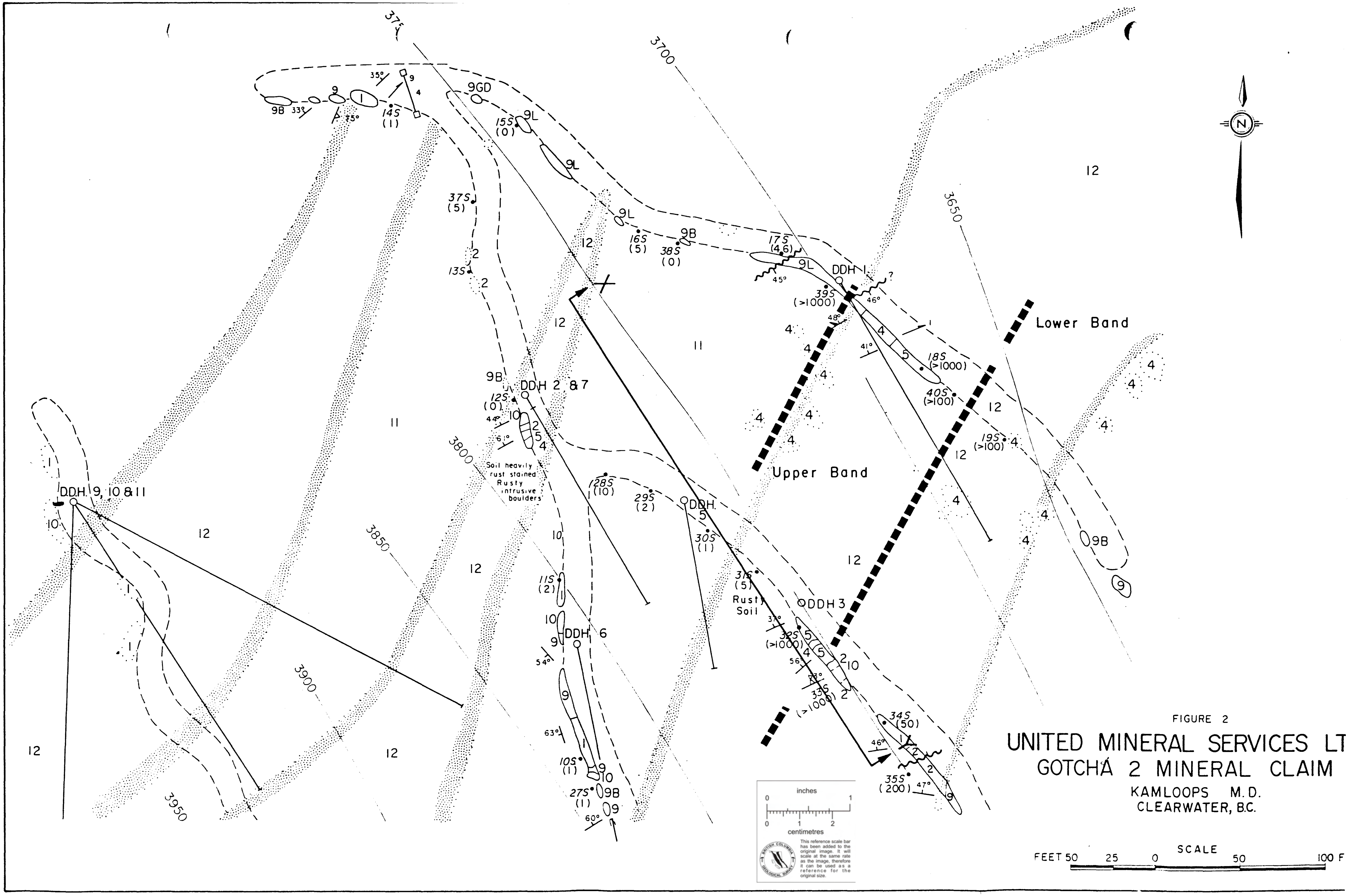


FIGURE 2
UNITED MINERAL SERVICES LT
GOTCHA 2 MINERAL CLAIM
KAMLOOPS M.D.
CLEARWATER, B.C.
SCALE
FEET 50 25 0 50 100 F
Upper Band
Lower Band
DDH 1
DDH 2 & 7
DDH 3
DDH. 5
DDH 6
DDH. 9, 10 & 11
Soil heavily rust stained Rusty intrusive boulders
Rusty Soil
3700
3650
3800
3850
3900
3950
inches
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.

FIGURE 2A

UNITED MINERAL SERVICES LTD.

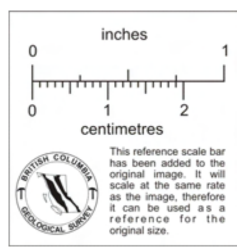
GOTCHA 2 MINERAL CLAIM

KAMLOOPS M.D.
CLEARWATER, B.C.

CROSS-SECTION X-Y

SCALE (HORIZ. & VERT.)

FEET 50 25 0 50 100 FEET



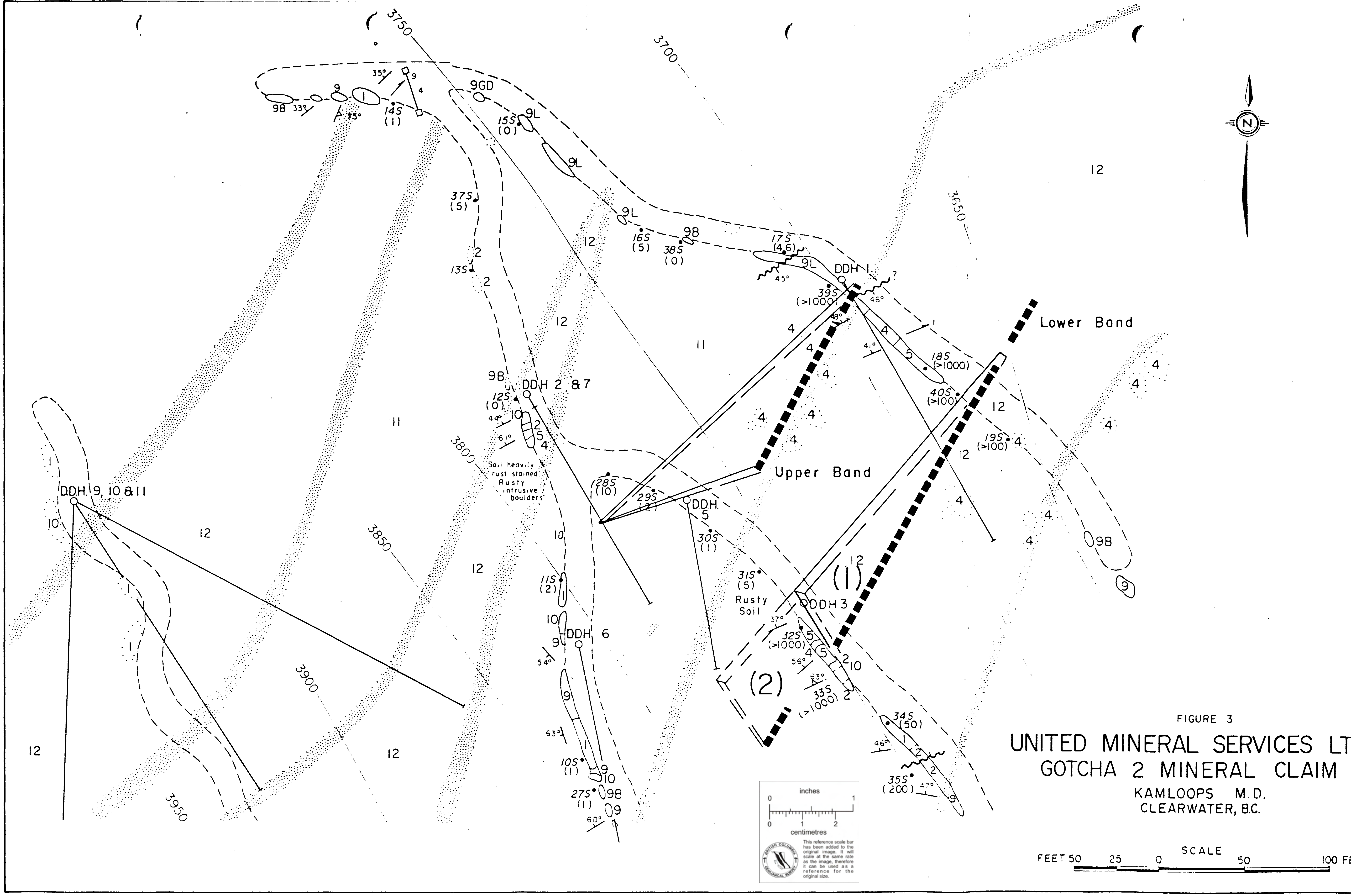


FIGURE 3
UNITED MINERAL SERVICES LT
GOTCHA 2 MINERAL CLAIM
KAMLOOPS M.D.
CLEARWATER, B.C.
SCALE
FEET 50 25 0 50 100 FT
Lower Band
Upper Band
DDH 1
DDH 2 & 7
DDH 3
DDH. 5
DDH 6
DDH. 9, 10 & 11
Soil heavily rust stained Rusty intrusive boulders
Rusty Soil
3750
3700
3650
3800
3850
3900
3950
(1)
(2)
inches
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.

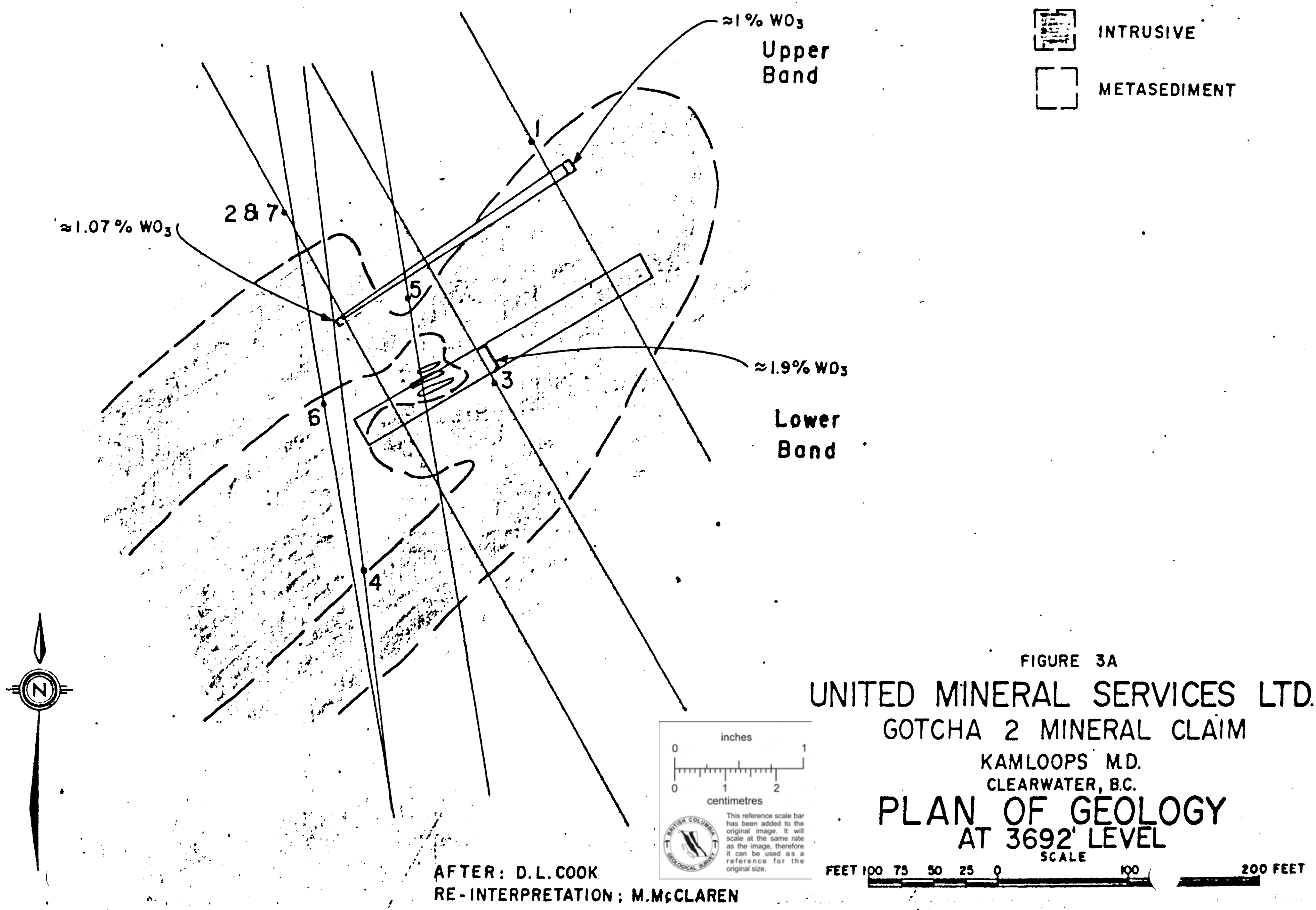


FIGURE 3A

UNITED MINERAL SERVICES LTD.

GOTCHA 2 MINERAL CLAIM

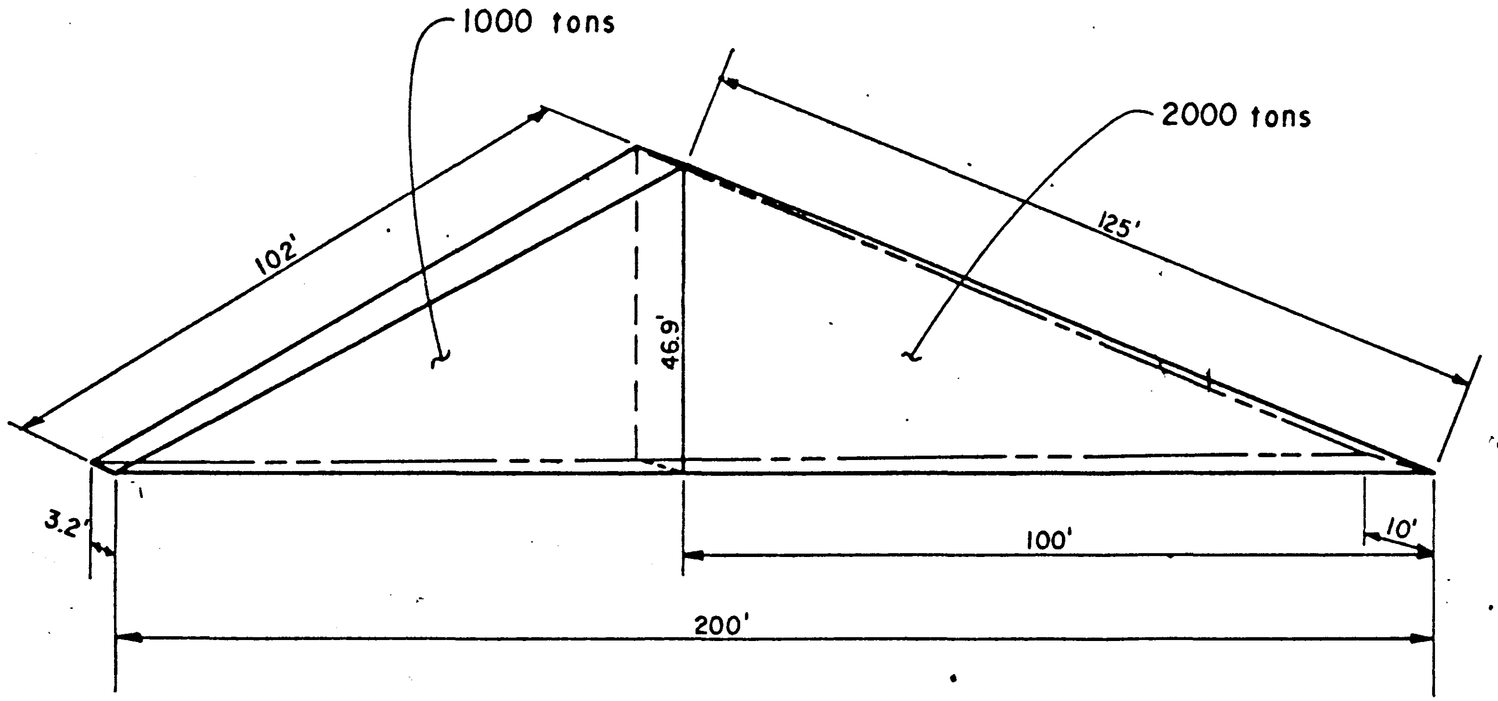
KAMLOOPS M.D.

CLEARWATER, B.C.

PLAN OF GEOLOGY AT 3692' LEVEL

AFTER: D.L. COOK

RE-INTERPRETATION; M.McCLAREN



TONNAGE ≃ 3000 tons

FIGURE 3B

UNITED MINERAL SERVICES LTD

GOTCHA 2 MINERAL CLAIM

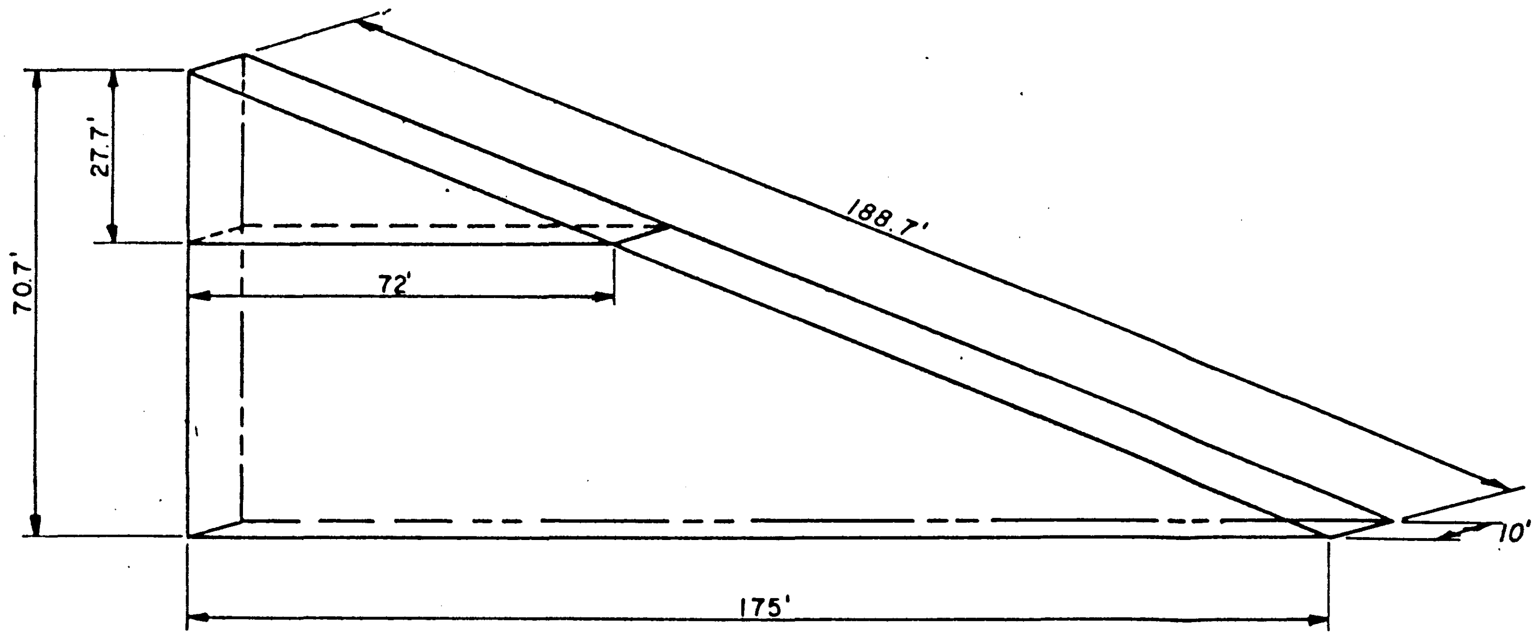
KAMLOOPS M.D.

CLEARWATER, B.C.

UPPER BAND

SCALE

FEET 30 20 10 0 3 60 FEET



TONNAGE ≃ 6000 tons

FIGURE 3C

UNITED MINERAL SERVICES LTD.

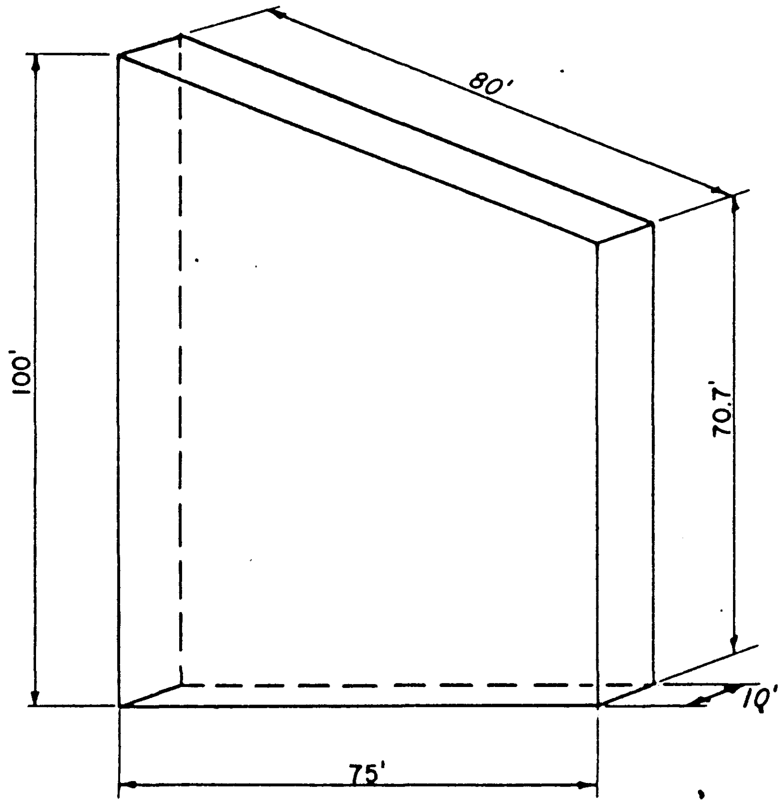
GOTCHA 2 MINERAL CLAIM

KAMLOOPS M.D.
CLEARWATER, B.C.

LOWER BAND-BLOCK I

SCALE

FEET 30 20 10 0 30 60 FEET



TONNAGE ≃ 6500 tons

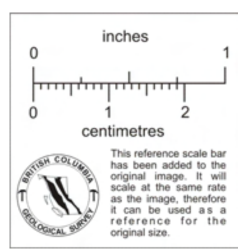


FIGURE 3D

UNITED MINERAL SERVICES LTD

GOTCHA 2 MINERAL CLAIM

KAMLOOPS M.D.
CLEARWATER, B.C.

LOWER BAND-BLOCK 2

SCALE

FEET 30 20 10 0 30 60 FEET

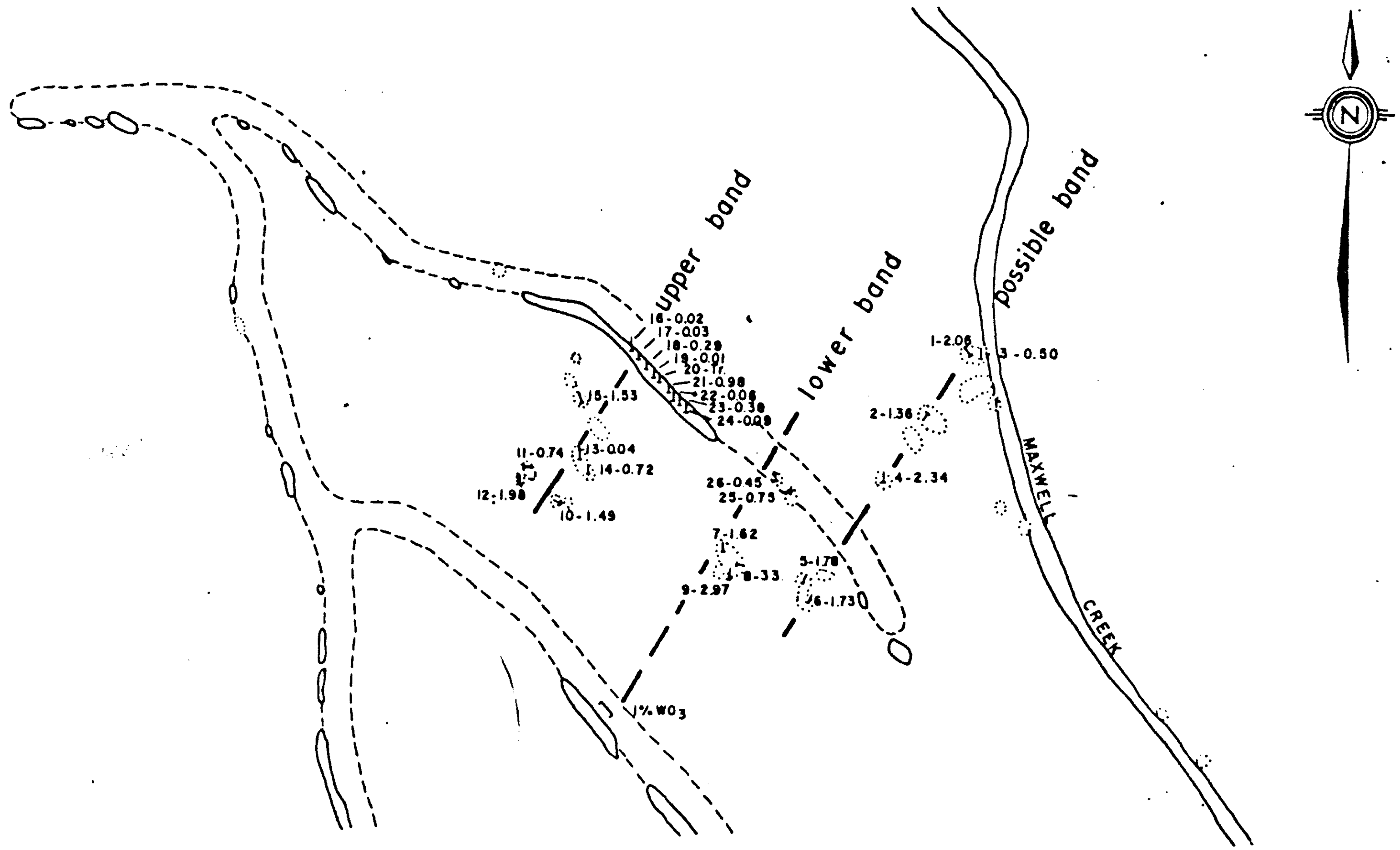


FIGURE 4

UNITED MINERAL SERVICES LTD.

GOTCHA 2 MINERAL CLAIM

KAMLOOPS M.D.

CLEARWATER, B.C.

SCALE

FEET 100 75 50 25 0 100 200 FEET

LEGEND

BOULDER

OUTCROP

SHORT PERCUSSION HOLE

(AFTER: D.L. COOK)

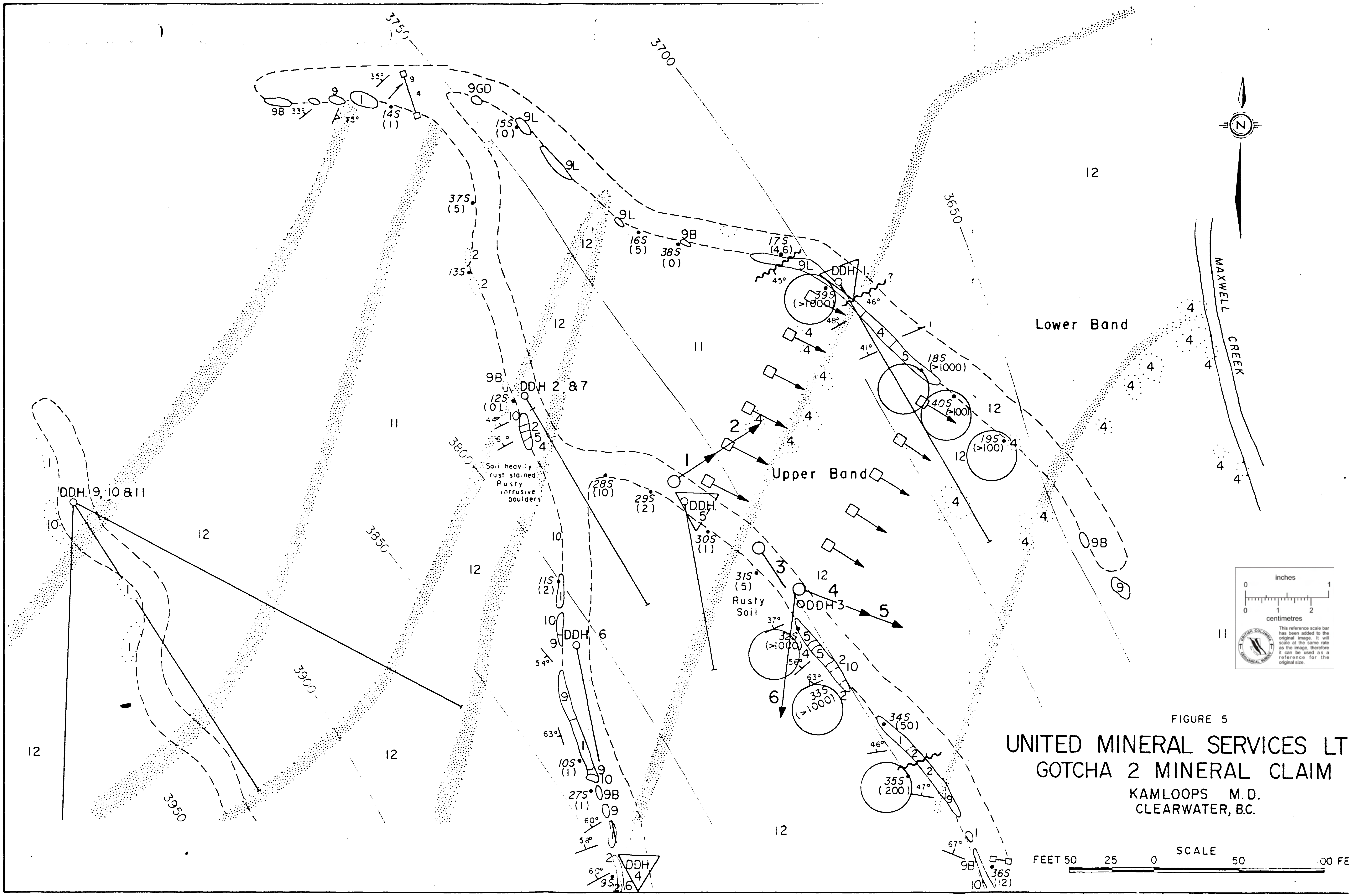


FIGURE 5

UNITED MINERAL SERVICES LT
GOTCHA 2 MINERAL CLAIM
KAMLOOPS M.D.
CLEARWATER, B.C.

PRELIMINARY ECONOMIC EVALUATION OF GOTCHA 2 CLAIM

1. INTRODUCTION

Proforma Income Statements for a small scale mining operation at the Gotcha 2 Claim are developed in this section to determine if the expected return warrants further exploratory drilling outlays. The two Income Statements produced use reasonable estimates of pertinent variables.

Proforma Income Statement 1 depicts profits that can be reasonably expected if all Indicated, Inferred and Possible Ore is proven and mined/milled as follows:

Phase 1 - Block 1, Lower Band, open pitted	(6,000 tons)
Phase 2 - Upper Band, open pitted	(2,000 tons)
Phase 3 - Block 2, Lower Band, underground mining	(6,500 tons)
Phase 4 - Upper Band - underground mining	(1,000 tons)

Proforma Income Statement 2 depicts profits that can be reasonably expected if:

a) only indicated reserves are mined/milled

b) indicated + inferred reserves are mined/milled

c) indicated + inferred + possible reserves are mined/milled.

2. DISCUSSION OF VARIABLES USED AND COST DEVELOPMENT

A. EXPLORATION EXPENSE

Exploration expenditures of $29,000 are required to move present ore classifications into a measured ore class. Exploration for other bands can be undertaken at a later date.

CALCULATION OF EXPLORATION EXPENSE

Diamond Drilling (700 ft. @ $15/ft.)	$10,500
X-Ray Drilling (1,000 ft. @ $10/ft.)	10,000
Cat Trenching (120 hrs. @ $40/hr.)	4,800
Cat Mob-Demob	200
Geological Supervision	2,500
Bulk Sampling-Testing	1,000
TOTAL EXPLORATION EXPENSES	$29,000

B. MARKETS AND SMELTER PRICES

Canada Tungsten (North Vancouver) and Union Carbide (Upper Scheelite-California) are the two main buyers of tungsten concentrate. Canada Tungsten will pay for concentrate, grading 60%+ WO_3, the London Metal Exchange Price less an inflated marketing cost which may be 30% of the LME price. The current LME price for tungsten is $175-$180(US) per short ton unit of 1% WO_3. Union Carbide buys concentrate at scheduled prices, currently $120(US) per short ton unit in concentrate form averaging 60% WO_3. Unlike Canada Tungsten, Union Carbide buys concentrate grading less than 60% WO_3 at reduced prices (see Appendix C). Union Carbide will negotiate large shipments and quality concentrate at higher prices. Payment is made within 14 days by Union Carbide while Canada Tungsten may take 2-3 months to market concentrate.

Concentrate is sold at <u>$120.00(US)</u> per STU for development of the statements.

C. RECOVERY

Recovery of scheelite is assumed to be 70%. A survey of 10 gravity mills reported in Transactions Volume 46 - Canadian Institute of Mining and Metallurgy - 1943 shows recoveries typically range from 65 to 85%. Metallurgical testing at the exploration stage will provide a more accurate figure. The coarseness of scheelite grains suggests a high recovery may be possible.

CALCULATION OF NET SMELTER RETURN FOR INCOME STATEMENT 1

(Dilution ?)

	PHASE 1	PHASE 2	PHASE 3	PHASE 4
Tons of Ore	6,000	2,000	6,500	1,000
Average Grade (%)	1.9	1	1	1
Recovery (%)	70	70	70	70
Grade Recovered (%)	1.33	.7	.7	.7
Grade of Concentrate (%)	60	60	60	60
Short Ton Units WO_3 (STU)	7,980	1,400	4,550	700
Smelter Price per STU ($US)	120	120	120	120
Net Smelter Return ($US)	957,600	168,000	546,000	84,000
Accumulated Total ($US)	957,600	1,125,600	1,671,600	1,755,560
Net Smelter Return ($CAN-3%)	986,328	173,040	562,380	89,040
Accumulated Total ($CAN)	986,328	1,159,368	1,721,748	1,810,788
Net Smelter Return ($CAN) with 80% Recovery	1,127,232	197,760	642,720	98,880

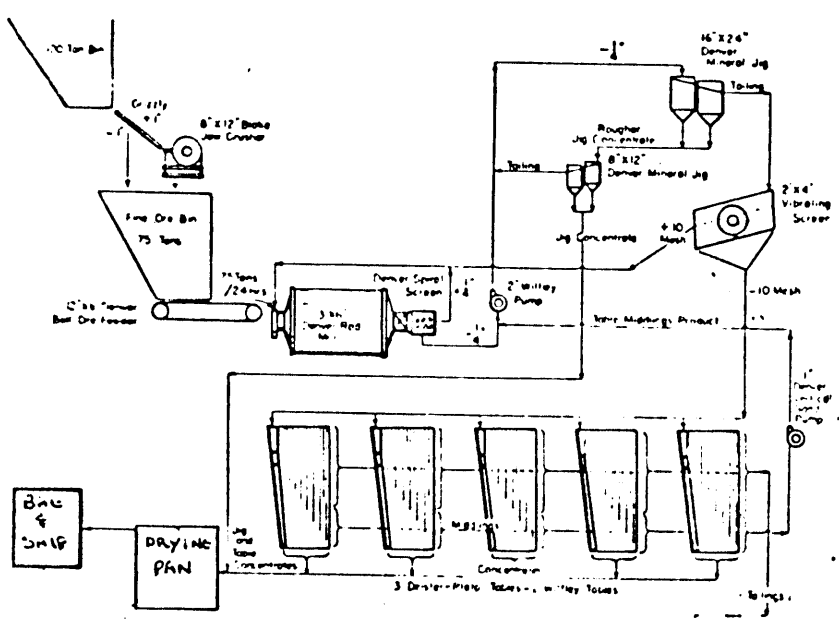
CALCULATION OF NET SMELTER RETURN
FOR INCOME STATEMENT 2

	INDICATED ORE	INFERRED ORE	POSSIBLE ORE
Tons of Ore	2,900	6,100	6,500
Average Grade (%)	1.28	1.75	1
Recovery (%)	70	70	70
Grade Recovered(%)	.89	1.23	.70
Grade of Concentrate (%)	60	60	60
Short Ton Units WO_3 (STU)	2,581	7,503	4,550
Smelter Price per STU ($US)	120	120	120
Net Smelter Return ($US)	309,720	900,360	546,000
Accumulated Total ($US)	309,720	1,210,080	1,756,080
Net Smelter Return ($CAN-3%)	319,012	927,371	562,380
Accumulated Total ($CAN)	319,012	1,246,383	1,808,763
Net Smelter Return ($CAN) with 80% Recovery	365,609	1,055,544	642,720

D. CAPITAL COST

It is assumed ore would be concentrated using a 50-75 ton per day gravity separation mill. This size mill should run all indicated, inferred and possible ore classified to date within one year. Equipment dealers in Vancouver indicate machinery is available and from these sources at least, equipment cannot be leased or rented.

The mill flow envisioned is diagramatically depicted as:



The cost of milling assets installed is assumed to be $100,000.

CAPITAL COST CALCULATION FOR 50-75 TON PER DAY MILL

CAPITAL ASSETS	COSTS
2 Ore Bins	$ 2,000
Grizzly	500
Jaw Crusher	2,000
Ore Feeder	500
Rod Mill	25,000
2 Jigs @ $4,000 per	8,000
4 Tables @ $6,000 per	24,000
3 Pumps @ $2,000 per	6,000
Generator	5,000
Screens & Misc.	2,000
Total	$75,000
Installation	25,000
TOTAL INSTALLED ASSETS	$100,000

E. MINING COSTS

Mining costs are developed from current contract mining rates. Block 1, Lower Band (6,000 tons) is assumed open pitable as is indicated ore from Upper Band (2,000 tons). Inferred ore from Upper Band (1,000 tons) and Block 2, Lower Band (6,500 tons) is assumed to be mined totally by underground methods.

CALCULATION OF MINING COSTS AT CONTRACT RATES FOR INCOME STATEMENT 1

	PHASE 1	PHASE 2	PHASE 3	PHASE 4
Tons Open Pit Ore	6,000	2,000		
Tons Underground Ore			6,500	1,000
Mob-Demob	$ 6,000	$ 6,000	$ 6,000	$
Open Pit Costs @ $12/ton	72,000	24,000		
Underground Costs				
Drifting @ $127/ft. (10'x10')			10,000	12,000
Raising, Stoping @ $75/ft.			10,000	4,000
Slashing @ $1.40/cu. ft.			57,000	17,000
Total Mining Costs	$78,000	$ 30,000	$ 83,000	$ 33,000
Accumulated Total	78,000	108,000	191,000	224,000

CALCULATION OF MINING COSTS
AT CONTRACT RATES
FOR INCOME STATEMENT 2

	INDICATED	INFERRED	POSSIBLE
Tons Open Pit Ore	2,900	5,100	
Tons Underground Ore		1,000	6,500
Mob-Demob	$ 6,000	$ 6,000	$ 6,000
Open Pit Costs @ $12/ton	34,000	62,000	
Underground Costs			
Drifting @ $127/ft.		12,000	10,000
Raising, Stoping @ $75/ft.		4,000	10,000
Slashing @ $1.40/cu.ft.		17,000	57,000
Total Mining Cost	$40,000	$101,000	$ 77,000
Accumulated Total	40,000	141,000	224,000

F. MILLING - OPERATING COSTS

Milling and operating cost estimates are based on a 50 TPD operation.

CALCULATION OF MILLING - OPERATING COSTS
FOR INCOME STATEMENT 1

	PHASE 1	PHASE 2	PHASE 3	PHASE 4
Tons of Ore	6,000	2,000	6,500	1,000
Tons of Diluted Ore (10%)	6,600	2,200	7,200	1,100
Days Milling @ 50 TPD	132	44	144	22
Months Milling	4.4	1.5	4.8	.7
Wages (2 shifts x 3 men x $2,000/mo.)	$ 52,800	$ 18,000	$ 57,600	$ 8,400
Cook (@ $1,000/mo.)	4,400	1,500	4,800	700
Fuel (4 gal./hr./ 20 hrs. x $1.00/gal.)	10,500	3,500	11,000	1,700
Vehicles (2 @ $500/mo.)	4,400	1,500	4,800	700
Camp Rental	10,000	4,000	10,000	2,000
Insurance	2,000			
Legal & Audit	2,000			
Consulting	10,000			
Supervision	6,000	2,500	7,000	1,500
Environment Repair	10,000	2,000	5,000	2,000
Total Operating Costs	$112,000	$ 33,000	$100,000	$ 17,000
Accumulated Total	112,000	145,000	245,000	262,000

CALCULATION OF MILLING - OPERATING COSTS FOR INCOME STATEMENT 2

	INDICATED	INFERRED	POSSIBLE
Tons of Ore	2,900	6,100	6,500
Tons of Diluted Ore (10%)	3,200	6,600	7,150
Days Milling @ 50 TPD	64	132	143
Months Milling	2.1	4.4	4.8
Wages (2 shifts x 3 men x $2,000/mo.)	$25,000	$ 52,800	$ 57,600
Cook (@ $1,000/mo.)	2,100	4,400	4,800
Fuel (4 gal./hr. x 20 hrs. x $1.00/gal.	5,100	10,500	11,000
Vehicles (2 @ $500/mo.)	2,000	4,400	4,800
Loader rental	2,000	4,000	3,000
Camp Rental	6,000	10,000	10,000
Insurance	2,000		
Legal & Adit	2,000		
Supervision	6,000	7,000	7,000
Environment Repair	5,000	10,000	5,000
Total Operating Costs	$57,200	$103,100	$103,200
Accumulated Total	57,200	170,300	273,500

G. TRANSPORTATION COSTS

Transportation costs are developed from carrier current quotations. Transportation costs For concentrate delivered to Canada Tungsten, North Vancouver, and delivered to Union Carbide, Upper Scheelite-California, are both shown.

CALCULATION OF TRANSPORTATION COSTS FOR INCOME STATEMENT 1

	PHASE 1	PHASE 2	PHASE 3	PHASE 4
Tons of Concentrate	133	23	77	12
Tons of Contained WO_3	80	14	46	7
Trucking Clearwater-Kamloops @ $144/22 ton load	$ 864	$ 144	$ 576	$ 144
Kamloops-Vancouver @ $172/22 ton load	1,032	172	688	172
Total Cost Delivered to Canada Tungsten	$1,896	$ 316	$1,264	$ 316
Accumulated Total	1,900	2,200	3,500	3,800
Train Vancouver-Lawz ($6.12/100 lbs. min. 50,000 lbs.)	16,524	3,060	9,425	3,060
Lawz-Upper Scheelite ($.215/100 lbs. min. 60,000 lbs.)	774	129	387	129

continued

	PHASE 1	PHASE 2	PHASE 3	PHASE 4
Duty ($.25 per lb. contained WO_3)	40,000	7,000	23,000	3,500
Customs Broker, Loading-Unloading	4,000	1,000	3,000	1,000
Total Cost Delivered to Union Carbide	$63,194	$11,505	$37,076	$ 8,005
Accumulated Total	63,200	74,700	111,800	119,000

CALCULATION OF TRANSPORTATION COSTS
FOR INCOME STATEMENT 2

	INDICATED	INFERRED	POSSIBLE
Tons of Concentrate	43	124	77
Tons of Contained WO_3	26	75	46
Trucking Clearwater-Kamloops @ $144/22 ton load	$ 288	$ 864	$ 576
Kamloops-Vancouver @ $172/22 ton load	344	1,032	688
Total Cost Delivered to Canada Tungsten	$ 632	$ 1,896	$ 1,264
Train Vancouver-Lawz ($6.12/100 lbs. min. 50,000 lbs.)	6,120	15,300	9,180
Lawz-Upper Scheelite ($.215/100 lbs. min. 60,000 lbs.)	258	516	387
Duty ($.25 per lb. contained WO_3)	13,000	37,500	23,000
Customs Broker, Loading-Unloading	2,000	4,000	2,000
Total Cost Delivered to Union Carbide	$22,010	$59,212	$35,831
Accumulated Total	22,000	81,200	117,000

H. TAXES

Taxes applicable to the operation would be B.C. Mineral Resource Tax, B.C. Income Tax and Federal Income Tax. These are calculated from projected operating profits.

CALCULATION OF TAXES FOR INCOME STATEMENT 1

	PHASE 1	PHASE 2	PHASE 3	PHASE 4
B.C. Mineral Resource Tax				
Net Smelter Return	$986,328	$173,040	$562,380	$89,040
Less Mining	78,000	30,000	83,000	33,000
Milling	112,000	33,000	100,000	17,000
Transport	1,900	316	1,264	316
Operating Profit	794,428	109,724	378,116	38,724
Less Depreciation (30%)	29,000			
Depletion	30,000			
Subtotal A	735,428	109,724	378,116	38,724
Less Processing Allowance (15% of Subtotal A)	110,314	16,459	56,717	5,808
Taxable Profit	625,114	93,265	326,499	32,916
B.C. Mineral Resource Tax (17.5%)	$109,395	$16,321	$57,137	$5,760
Provincial Income Tax				
Operating Profit	$794,428	$109,724	$378,116	$38,724
Less Depreciation	30,000			
Depletion	10,000			
B.C. Resource Tax	109,395	16,321	57,137	5,760
Taxable Profit	645,033	93,403	326,979	32,964
Provincial Income Tax (15%)	$96,755	$14,010	$49,047	$4,945
Federal Income Tax				
Operating Profit	$794,428	$109,724	$378,116	$38,724
Less Depreciation	30,000			
Mineral Resource Profit	764,428	109,724	378,116	38,724
Less Mineral Resource Allowance (25%)	191,107	27,431	96,029	9,681
Exploration	29,000			
Taxable Production Profit	544,321	82,293	288,087	29,043
Federal Income Tax (36%)	$195,956	$29,625	$103,711	$10,455

CALCULATION OF TAXES
FOR INCOME STATEMENT 2

	INDICATED ORE	INFERRED ORE	POSSIBLE ORE
B.C. Mineral Resource Tax			
Net Smelter Return	$319,012	$927,371	$562,380
Less Mining	40,000	101,000	77,000
Milling	57,200	103,100	103,200
Transport	22,010	59,212	35,831
Operating Profit	199,802	664,059	346,349
Less Depreciation	29,000		
Depletion	30,000		
Subtotal A	140,802	664,059	346,349
Less Processing Allowance (15% of Subtotal A)	30,832	99,609	51,952
Taxable Profit	109,970	564,450	294,397
B.C. Mineral Resource Tax (17.5%	$19,245	$98,779	$51,519
Provincial Income Tax			
Operating Profit	$199,802	$664,059	$346,349
Less Depreciation	30,000		
Depletion	10,000		
B.C. Resource Tax	19,245	98,779	51,519
Taxable Profit	140,557	565,280	294,830
Provincial Income Tax (15%)	$21,084	$84,792	$44,225
Federal Income Tax			
Operating Profit	$199,802	$664,059	$346,349
Less Depreciation	30,000		
Mineral Resource Profit	169,802	664,059	346,349
Less Mineral Resource Allowance (25%)	42,451	166,015	86,587
Exploration	29,000		
Taxable Production Profit	98,351	498,044	259,762
Federal Income Tax (36%)	$35,406	$179,296	$93,514

3. PROFORMA INCOME STATEMENT 1

Assumptions: (1) Indicated, Inferred, Possible Ore Proven.
(2) 70% Recovery.
(3) Concentrate Grades 60%+ WO_3.
(4) Concentrate Sold to Can-Tung @ $120(US) per STU.

	PHASE 1	PHASE 2	PHASE 3	PHASE 4
Total Capital Cost	$100,000			
Net Smelter Return ($Can)	$986,328	$173,040	$562,380	$89,040
Less Mining	78,000	30,000	83,000	33,000
Milling	112,000	33,000	100,000	17,000
Transport	1,900	316	1,264	316
Operating Profit	794,428	109,724	378,116	38,724
Less Exploration Expense	29,000			
Depreciation	30,000			
Taxable Income	735,428	109,724	378,116	38,724
Less Taxes				
B.C. Mineral Resource Tax	109,395	16,321	57,137	5,760
B.C. Income Tax	96,755	14,010	49,047	4,945
Federal Income Tax	195,956	29,625	103,711	10,455
NET PROFIT	$333,322	$49,768	$174,221	$17,564
Accumulated Total	$333,322	$383,090	$557,311	$574,875
+ Sale of Assets				50,000
Total Return				$625,000
Working Capital Required	$220,900	$63,316	$178,264	$50,316
Working Capital Available	0	573,528	683,252	1,067,368

4. PROFORMA INCOME STATEMENT 2

Assumptions: (1) Indicated Ore Proven, Indicated + Inferred Proven, Indicated + Inferred + Possible Ore Proven.
(2) 70% Recovery.
(3) Concentrate Grades 60%+ WO_3.
(4) Concentrate Sold to Union Carbide @ $120 (US) per STU.

	INDICATED ORE		INFERRED ORE		POSSIBLE ORE
Total Capital Cost	$100,000				
Net Smelter Return	$319,012		$927,371		$562,380
Less Mining	40,000		101,000		77,000
Milling	57,200		103,100		103,200
Transport	22,010		59,212		35,831
Operating Profit	199,802		664,059		346,349
Less Exploration Expense	29,000				
Depreciation	30,000				
Taxable Income	140,802		664,059		346,349
Less Taxes					
B.C. Mineral Resource Tax	19,245		98,779		51,519
B.C. Income Tax	21,084		84,792		44,225
Federal Income Tax	35,406		179,296		93,514
NET PROFIT	$65,067		$301,192		$157,091
Accumulated Total	$65,067		$366,259		$523,350
+ Sale of Assets	50,000		50,000		50,000
Total Return	$115,000	or	$416,000	or	$575,000
Working Capital Required	$148,210		$263,312		$216,031
Working Capital Available	0		170,802		834,861

5. CONCLUSIONS

(1) The project could be abandoned after $17,000 of exploration expense.

(2) Net profits are very sensitive to changes in recovery. A 10% increase in recovery would increase net profits by greater than 20%.

(3) Cost of transportation of concentrate to Union Carbide are about 17% of total operating costs, but are negligible to Canada Tungsten.

(4) Canada Tungsten would be the better market because of possible higher smelter prices and low transportation cost if sufficient working capital is available to withstand their payment period of 2-3 months.

(5) All ore classified at present could be milled within one year.

(6) Operating costs average about $17 per ton ore. Significant savings could likely be made if a custom mill could be found within transporting distance.

(7) The effective tax rate of about 55% would be reduced somewhat by small business incentives.

(8) The project would require $100,000 for aquisition of assets and at least $200,000 working capital.

(9) The return on capital is estimated to be about 600% if all ore is proven.

(10) The return/risk ratio is high for this project.

A P P E N D I X C

JAMES P. ELWELL, P.ENG.
CONSULTING MINING ENGINEER

PHONE: 682-2120
RES: 922-2551

1020 - 510 W. HASTINGS ST.
VANCOUVER 2, B.C.

February 14, 1978

NCA Minerals Ltd.
P. O. Box 371
Station "A"
VANCOUVER, B. C.

Dear Sirs:

PROGRESS REPORT - EXPLORATION ON THE GOTCHA CLAIMS

In my report on the Gotcha mineral claims dated September 14, 1977, the conclusions stated were that the previous drilling by Union Carbide had indicated 15,000 s.t.u. of WO_3 in the two known mineral zones with good possibilities for increasing this tonnage by further exploration along strike and to depth. The initial recommendations made at this time called for an additional 1,000 feet of drilling and some bulldozer trenching.

As it was not possible to carry out this program until January of this year, it was found that diamond drilling would be difficult and extremely expensive due to the severe winter conditions in this area. It was decided therefore, to substitute percussion drilling for the diamond drilling, using a Track mounted Atlas Copco overburden drill, the drill cuttings from each 5 foot section of each hole being bagged for examination and assay.

This program was carried out during the period January 9 to 17, a total of 975 feet of drilling being completed in 18 holes. The program was under the supervision and management of Mr. Murray McLaren, B.Sc., geologist, who had spent considerable time on the property during the summer of 1977, and was familiar with the geology, and structural controls of the mineral deposit. The results of this work are summarized below.

Percussion Drilling

Lower Band

Hole No.	Bearing °	Dip °	Depth in ft.
1		vert.	70
2		vert.	80
3	256	56	75
4	270	66	65

No. 9
For Identification
Exam of McLaren

..../2

Hole No.	Bearing °	Dip °	Depth in ft.
5	310	70	80
6	214	70	90
7	N	48	50
8	N	55	45
9	N	51	95
10	30	60	55
11	230	52	80

Upper Band

Hole No.	Bearing °	Dip °	Depth in ft.
12	310	60	25
13	236	50	30
14	292	54	20
15	186	54	20
16	290	50	30
17	270	43	35
18	340	55	25

The holes were sampled in 5 foot sections, a visual estimate of the scheelite content was made by the intensity of fluorescence under ultra violet light, and all samples indicating even minor fluorescence were submitted for chemical assay.

Excellent drilling results were achieved on the Lower Band, the ground being competent, and sample recovery was estimated to be practically 100%. The mineral zone of the Upper Band was found to be highly fractured and impossible to drill to depth and obtain acceptable sample return, so for the time being, the tonnage estimate for this band is being based on the drill results of Union Carbide.

A 1" to 50' scale plan showing the roads, trenches, D.D. and percussion drill holes, and a 1" to 25' detail plan of the drill holes and mineral bands accompanies this report.

Ore Estimates

Lower Band

The drilling to date has indicated that the Lower Band is roughly lenticular in shape, tending to pinch to the southwest along strike and also to depth, with the widest part being 17 - 20 feet. In order to calculate the tonnage in this band to the limits of the drilling, it has been broken down into a number of irregular, blocks with tonnages calculated for each segment. These blocks are depicted in the three dimentional drawings which accompany this report.

Drill indicated tonnage consists of blocks intersected by one or more drill holes.

Probable tonnage is that which has not been intersected by a drill hole, but is directly adjacent to a drill indicated block.

Possible ore represents a block which is a reasonable geological extension of the known ore, but which has not been confirmed by drilling.

On the above bases, the Lower Band has been calculated to contain.

Drill indicated		5,500 tons
Probable		600 tons
Possible		1,100 tons
	TOTAL	7,200 tons

Union Carbide's estimate of grade for the Lower Band is 2% WO_3. At this grade the drill indicated tonnage alone would represent 11,000 s.t.u. WO_3, and if the possible and probable blocks are included, a total of 14,400 s.t.u. WO_3 are indicated.

Upper Band

Due to bad ground conditions the percussion drilling on the band did not extend the ore limits beyond that indicated by Union Carbide which was estimated at 5,000 tons at 1% WO_3 or 5,000 s.t.u. United Mineral Services re-calculated the tonnage of this block as 3,000 s.t.u. of drill indicated ore, making no allowance for extensions beyond the limits of drilling. A three dimensional block diagram of this body, prepared by United Mineral Services, accompanies this report.

Float Ore

On the surface there are a large number of massive blocks of well mineralized float which probably originated from the Lower Band. Many of the larger pieces have been drill sampled by Union Carbide, and a concervative estimate of float material in sight is 1,000 tons with an average grade of 2% WO_3 or 2,000 s.t.u.

Summary

Drill indicated ore:

Upper Band		3,000 s.t.u.
Lower Band		11,000 s.t.u.
Float		2,000 s.t.u.
	Total	16,000 s.t.u.

Probable and Possible ore:

Upper Band	2,000 s.t.u.
Lower Band	1,700 s.t.u.
Total	3,700 s.t.u.

Total Drill Indicated, Probable and Possible - all zones = 16,000 + 3,700
= 19,700 s.t.u. WO_3

Using a price of $163.00 Can. per s.t.u. WO_3 the deposit would have a gross value of $2,608,000 in reasonably assured or drill indicated ore, or $3,211,000 if the probable and possible ore is included, without further exploration.

Recommendations

A preliminary feasibility study should be initialed immediately. This should consist of ore analysis, metalurgical tests, investigations into plant design, operating and capital costs, etc. In addition, in anticipation of the feasibility study being favourable, application should be made immediately for a production permit to avoid delays at a later date.

Yours truly,

J. P. ELWELL, P.Eng

JPE:pr

Attachments

Plan of Trenches, D.D. & Percussion Holes
Detail Plan of Mineral Zones
Three dimentional drawings of Lower Band Ore Blocks
Three dimentional drawings of Upper Band Ore

APPENDIX D

D

G

PERCUSSION DRILLING RESULTS
AND PRELIMINARY COST STUDY

GOTCHA PROPERTY, CLEARWATER AREA, B. C.

FOR

NCA MINERALS CORP.
P. O. Box 371
Station "A"
Vancouver, B. C.

BY

J. P. Elwell, P.Eng.
1030 - 510 West Hastings Street
Vancouver, B. C.
V6B 1L8

March 2, 1978

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MAPS

APPENDIX

PERCUSSION DRILLING RESULTS AND PRELIMINARY COST STUDY, GOTCHA PROPERTY, CLEARWATER AREA, B. C.

Summary

An analysis of the assay results of the percussion drilling program on the Gotcha scheelite property indicate the total reserves in the mineral zones without further exploration to be 23,900 s.t.u. WO_3 of which 19,900 s.t.u. are drill indicated, and the remainder classed as probable and possible.

The net value of the drill indicated reserves only after recovery of capital and deduction of 10% royalty is $1,974,200 on a 75% recovery basis and $2,260,760 with 85% recovery.

Total operating costs including mining, milling and overhead are estimated at $37,35/ton. Allowing for 25% dilution, the total tons to be extracted to yield the 19,900 s.t.u. is 11,500. Using these figures the indicated net profit per ton before taxes on a 75% recovery basis is $149.59, and on an 85% recovery basis is $174.45.

It is recommended that metallurgical tests should be proceeded with to determine the actual recovery rate and grade of concentrate that can be expected from actual production.

Development drifts should be driven over both the Upper and Lower bands to open up the part of the ore zones which will be mined by underground methods and also to provide access for further exploration of the mineral zones beyond the limits delineated by drilling.

Introduction

On February 14, 1978 the writer submitted a progress report on the exploration of the scheelite mineral deposits on the Gotcha claims located in the Clearwater area of the Kamloops Mining Division. This report covered the percussion drilling program completed in January 1978, and a preliminary ore estimate was compiled on the basis of a visual estimate of the grade of the drill hole samples along with the results of the previous diamond drilling by Union Carbide Corp.

A preliminary cost study and profit estimate was submitted on February 20, based on the above ore reserve data and cost estimates believed to be conservative. Since this data, the assays of the percussion drilling samples have been received and this report consists of an updated ore estimate and cost analysis based on the assay data.

Drill Hole Assays

The drill cuttings from each hole were taken in 5 foot sections for examination by ultra-violet light, and then grouped into sections of 10 feet or more according to the intensity of flourescence, as being submarginal, marginal, or ore grade. These grouped samples were assayed for % WO_3 by Can-Test Ltd., Vancouver. The results are tabulated as follows:

Hole No.	Footage From	Footage To	% WO_3
1.	20	30	0.03
	30	50	1.58
	50	60	0.85

Hole No.	Footage From	Footage To	% WO_3
2.	15	20	0.03
	20	25	0.03
	25	75	0.04
4.	45	60	0.06
5.	10	30	0.57
	30	45	0.14
	45	75	0.03
7.	25	35	0.03
	35	50	1.07
9.	25	50	4.30
	50	75	5.07
	75	95	1.47
10.	40	50	0.89
11.	10	30	2.84
	35	60	0.12
	60	75	0.10
12.	0	10	1.12
	10	20	1.25
13.	0	15	3.23
14.	0	10	3.05

Hole No.	Footage From	Footage To	% WO_3
17.	0	15	0.06
	15	30	0.35
18.	5	20	0.14

The samples from holes 3, 6, 8, and 16 were not submitted for assay, as they showed only very minor flourescence, and their location indicated that they were in either the footwall or hanging wall of the mineralized structure. No sample was recovered from Hole No. 15. The bearing, dip, and total length of each hole is tabulated in Appendix "A" of this report, and their location is shown on the detail plan. Copies of the Can-Test assay certificates are included as Appendix "B".

Hole No. 2 resulted in surprisingly low assays, considering it was parallel, and only about 20 feet from Hole No. 1 which averaged 1.37% WO_3 over 30 feet, and it is suspected that the structure may have rolled so that the drill hole remained in the hanging wall its entire length, or possibly, the mineral band has been split by a horse of waste at this point.

Ore Reserves

1. Lower Band

   In the block diagrams which accompany this report, each block designated Drill Indicated has been cut by one or more drill holes and the grade has been calculated by taking the average of the weighed averages of the drill hole sections within it.

From the cubic volume of each block and the average mineral content, the short ton unit equivalent has been arrived at. These are tabulated below, and are also shown on the block diagrams.

Block	Tons	Avg. Assay % WO_3	S.T.U. Equivalent
2	800	0.68	544
4	600	0.57	342
5	900	1.62	1456
7	700	1.84	1287
8, 9, 10	2200	3.76	8272
		Total drill indicates	11,900 s.t.u.

2. Upper Band

Holes #12 to #18 were drilled to test the Upper Band. The ground in this area was found to be badly shattered, and it was not possible to drill beyond about 30 feet and still obtain satisfactory sample returns, and in one case, hole #15, no sample was returned.

Holes #16, 17 and 18 were in the footwall of the structure and showed only low values in WO_3, but the assays from holes #12, 13 and 14 averaged 2.28% WO_3, therefore, for the purposes of the ore estimate, the original tonnage figure of 3,000 has been maintained, but an average grade of 2.0% WO_3 is used, giving a total of 6,000 s.t.u. WO_3 for this zone.

3. Float Ore

The estimate for the float ore remains the same as in the report of February 14, 1978 or 2,000 s.t.u.

4. Probable and Possible Ore

The block diagrams for the Lower Band show 900 tons classed as probable ore and 1,100 tons classed as possible ore. If a provisional assay value of 2% WO_3 is assigned to these, a total of 4,000 s.t.u. WO_3 is indicated.

5. Summary

Estimated Reserves in S.T.U. WO_3

	Drill Ind.	Probable	Possible	Total
Upper Band	6,000			6,000
Lower Band	11,900	1,800	2,200	15,900
Float	2,000			2,000
Totals	19,900	1,800	2,200	23,900

6. Additional Ore Possibilities

In the Lower Band, no ore has been projected beyond a reasonable zone of influence of the present drilling, but from a geological stand point, considerable additional ore is expected to be found downslope to the northeast and also at depth, and the possibility for extension of the zone to the southeast ore are far from being eliminated.

The Upper Band zone is still very incompletely outlined by drilling and there are indications that it may prove to be of equal grade and size to the Lower Band, but without further confirmed data, the original conservative estimate must stand.

Cost Study

On the bases of revised ore estimated above, the cost study and indicated return, from a mining operation on the property, has been updated from the report of February 20, which is attached as Appendix "C" to this report.

1. Ore Reserves - Drill Indicated only = 19,900 s.t.u.
   Current market value @ $C. 160/s.t.u. = $3,184,000.

		75% Rec.	85% Rec.
2.	Recoverable value	$2,388,000	$2,706,400
3.	Less 10% Royalty	$ 238,800	$ 270,640
4.	Net value to company	2,149,200	2,435,760
5.	Capital cost estimated		175,000 (1)
6.	Net after recovery of capital (total amortization in one year)	$1,974,200	$2,260,760

7. Indicated tons of ore to be mined to realize.

above net -	9,200
Dilution, 25%	2,300
	11,500 tons

8. Net value/ton (75% rec.) = $186.89
   (85% rec.) = $211.80

9.	Mining costs (100 tons/day basis)	\$21.75/ton (2)
10.	Milling costs (100 tons/day basis)	8.00/ton (3)
11.	Mine Development	2.60/ton (4)
12.	Overhead	5.00/ton (5)
13.	Total operating costs before taxes	\$37.35/ton

14. Indicated net profit per ton before taxes

    75% rec. bases = \$186.98 - \$37.35 = \$149.59

    85% rec. bases = \$211,80 - \$37.35 - \$174.45

15. Indicated total net profit before taxes assuming only 11,500 tons will be found.

    75% rec. bases = \$149.59 x 11,500 = \$1,720,285

    85% rec. bases = \$174.45 x 11,500 = \$2,006,175

16. Pre Production Cash Requirements (6)

Purchase of capital equipment and plant construction	\$175,000
Pre production stripping	10,000
Engineering, administration, etc.	15,000
	\$200,000

Conclusions and Recommendations

The exploration program to date has indicated a small, but highly profitable orebody with excellent possibilities for substantial additional reserves to be developed by further exploration.

The next phases of work recommended for the property are:

(1) Drive development adits over the ore in both the Upper and Lower bands. These should be located to serve as haulage tunnels for the part of the orebody which will be mined by underground methods, and provide access for further exploration.

(2) Metallurgical tests should be carried out on representative ore samples to determine the actual recovery and concentrated grade which can be achieved. It is expected that the concentrated grade will fall somewhere between the 75% and 85% limits used in this report.

March 2, 1978 J. P. Elwell, P.Eng.

Footnotes:
(1) Appendix "C" - Paragraph 5.
(2) Appendix "C" - Paragraph 6.
(3) Appendix "C" - Paragraph 7
(4) Appendix "C" - Paragraph 9
(5) Appendix "C" - Paragraph 10
(6) Appendix "C" - Page 4

CERTIFICATE

I, James Paul Elwell, of 4744 Caulfield Drive, West Vancouver, B. C., do hereby certify that:

1. I am a Consulting Mining Engineer residing at 4744 Caulfield Drive, West Vancouver, B. C., and with an office at 1030 - 510 West Hastings Street, Vancouver, B. C. V6B 1L8.

2. I am a graduate in Mining Engineering from the University of Alberta in 1940, and am a Registered Professional Engineer in the Province of British Columbia.

3. I have no personal interest, directly or indirectly in the properties or in NCA Minerals Corp. securities, nor do I expect to receive directly or indirectly any interest in such property or securities.

4. The findings in the report are from data obtained from the reports and maps acknowledged.

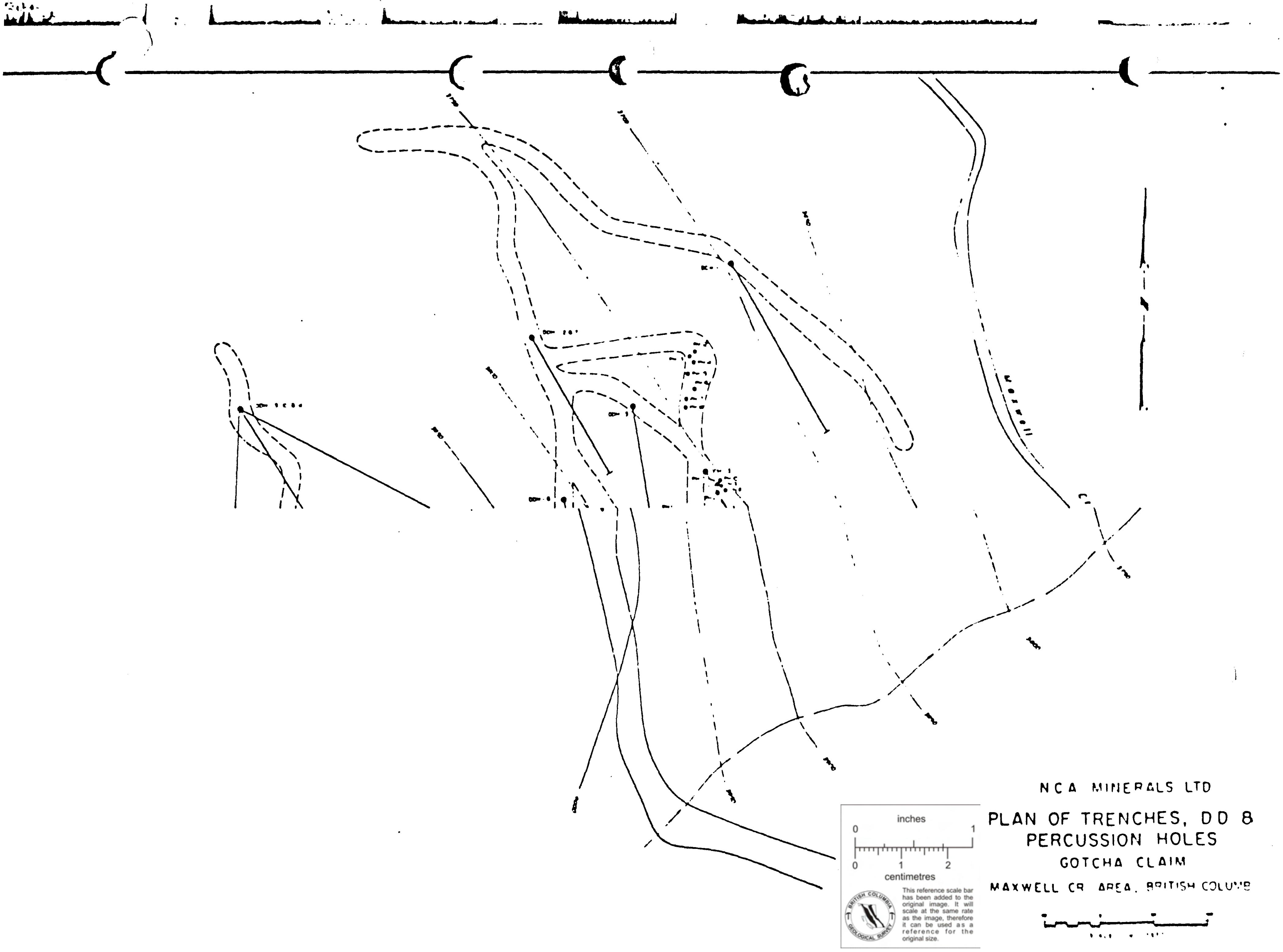
DATED at VANCOUVER, BRITISH COLUMBIA, this 2nd day of March, 1978.

JAMES PAUL ELWELL, P.Eng.

J. P. Elwell, P.Eng.

RESUME OF QUALIFICATIONS

1946 - 1950:	Employed by Cerro de Pasco Corp., Peru, as Mine Foreman and General Mine Forman in underground lead-zinc-copper mines using square set, cut-and-fill, and shrinkage stoping methods.
1950 - 1953:	Volcan Mines Co., Peru as Mine Superintendant to General Superintendant in 300 ton/day underground lead-zinc mine using cut-and-fill stoping method. Also acting manager of 190 ton/day gold mine and cyanide mill. Mining by room and pillar method.
1953 - 1960:	Minas de Matahambre, Cuba, as Mine Superintendant to General Manager. Mine produced 1000 tons/day from underground stopes to the 4,000 level using square-set and cut-and-fill methods.
1960 - 1967:	Registered Professional Engineer, Province of B. C. Independant Mining Consultant for various mining exploration projects in B. C., Yukon, N.W.T. and South America.
1967 - 1969:	Employed as Mining Expert by United Nations in Mexico; duties consisted of making an economic evaluation of the various mining properties, both metallic and non-metallic in the State of Oaxaca, Mexico, and making recommendations on properties which showed economic potential.
1969 - Present:	Continued as Self-employed Mining Consultant on exploration projects in Canada and Mexico.

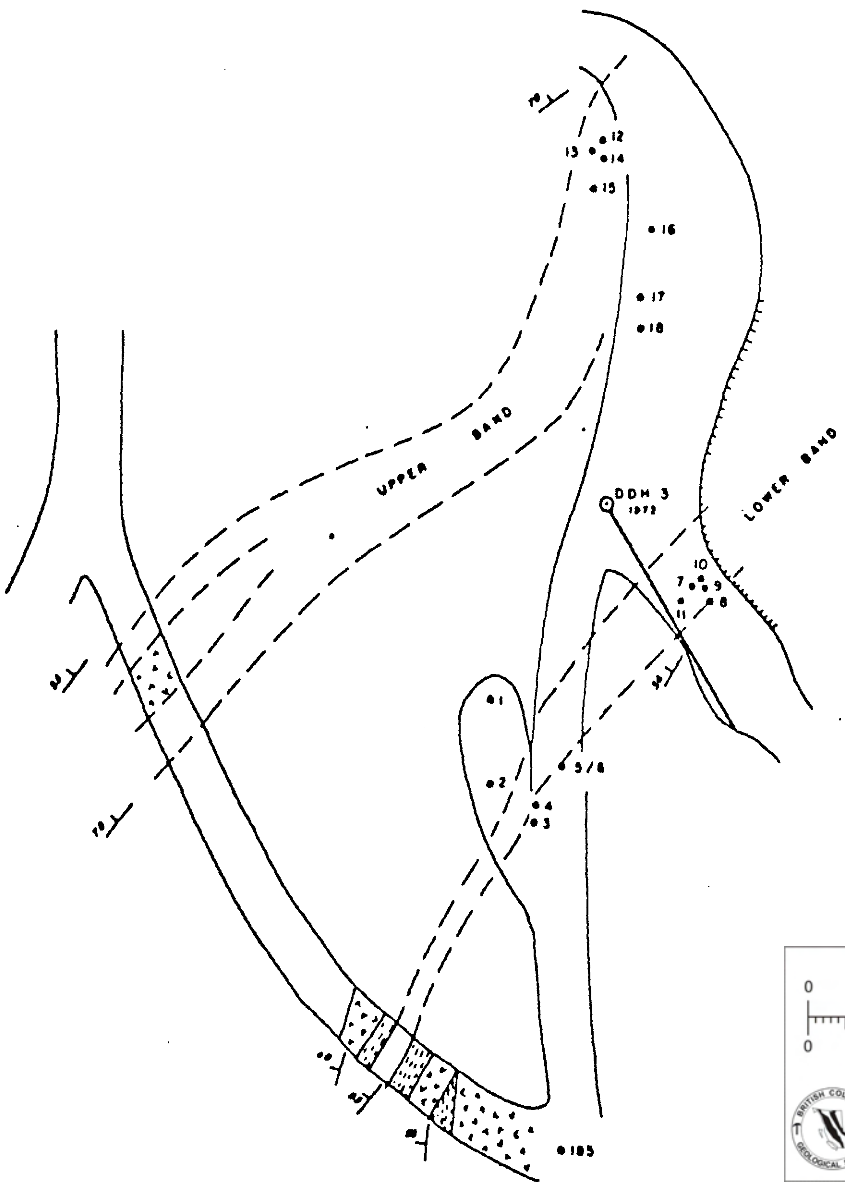


NCA MINERALS LTD

PLAN OF TRENCHES, DD & PERCUSSION HOLES

GOTCHA CLAIM

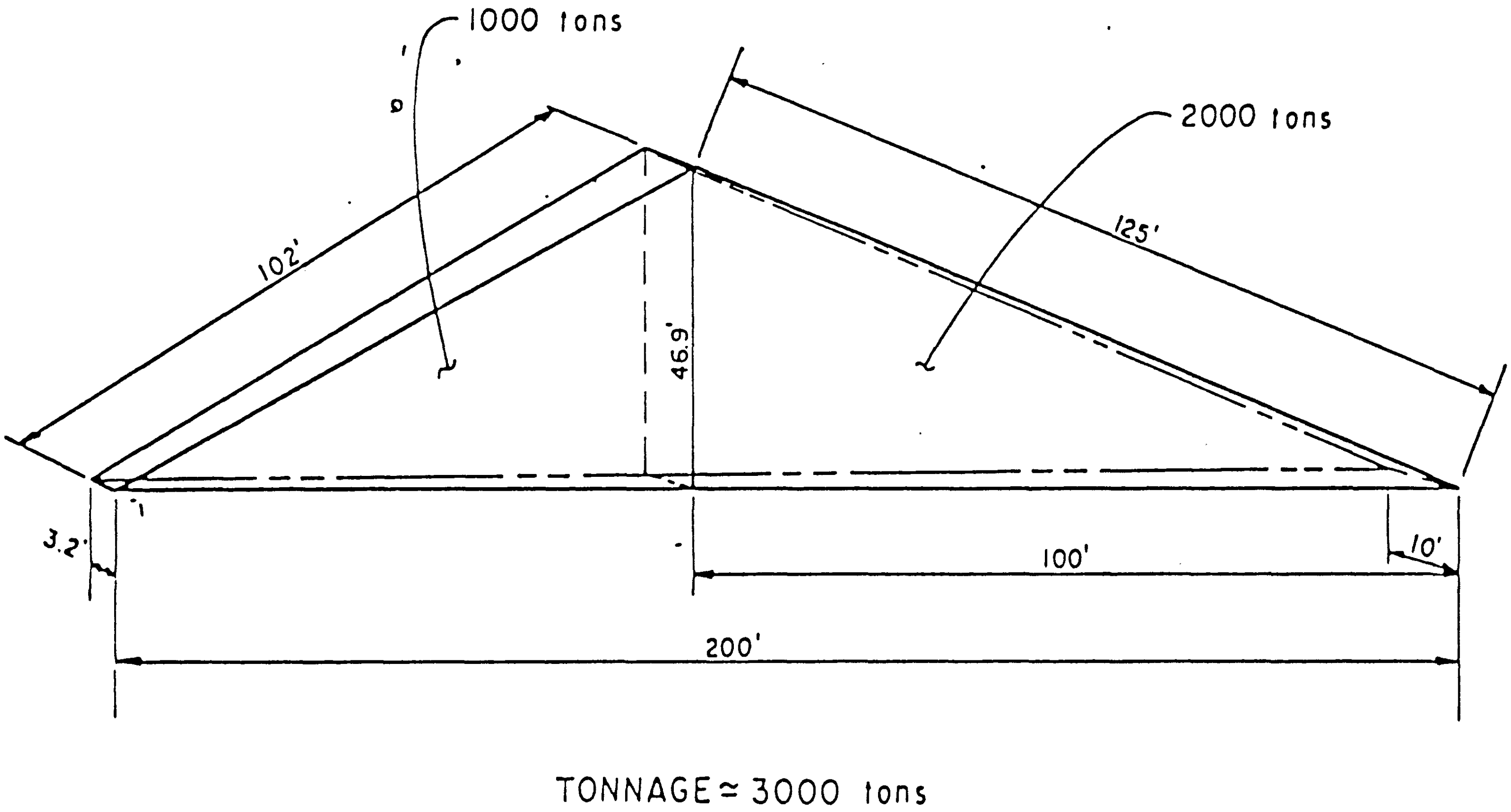
MAXWELL CR AREA, BRITISH COLUMB



NCA MINERALS LTD

DETAIL PLAN OF MINERAL ZONES SHOWING PERCUSSION AND D.D. HOLES

SCALE IN FEET



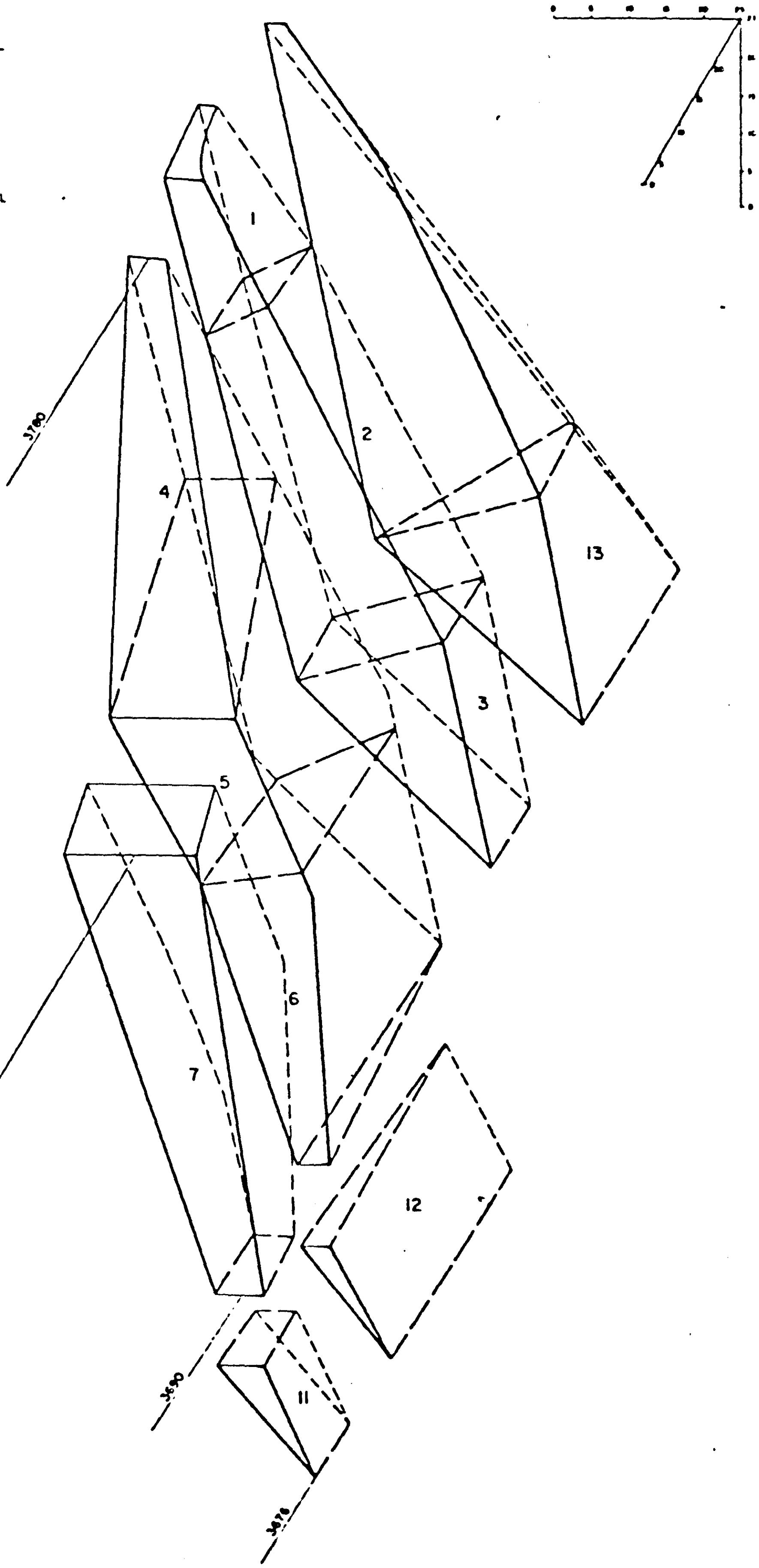
NCA MINERALS CORP.

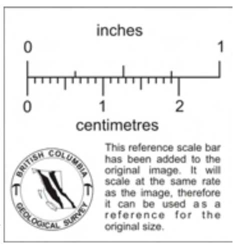
ESTIMATED TONNAGE FOR UPPER BAND GOTCHA MINERAL CLAIM

FEBRUARY, 1978

FEET 30 20 10 0 30 50 FEET

BLOCK	ESTIMATED TONNAGE	CLASS	ASSAY % WO_3	SHORT TON UNIT EQUIVELANT
1	100 T	PROBABLE		
2	800 T	DR IND	0 68	544
3.	300 T	PROBABLE		
4	600 T	DR IND	0 57	342
5	900 T	DR IND	1 62	1456
6	500 T	PROBABLE		
7	700 T	DR IND	1·84	1287
11	50 T	POSSIBLE		11,900 TOTAL
12	60 T	POSSIBLE		
13.	1000 T	POSSIBLE		





NCA MINERALS CORP.

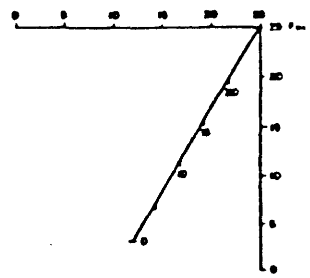
ESTIMATED TONNAGE FOR LOWER BAND

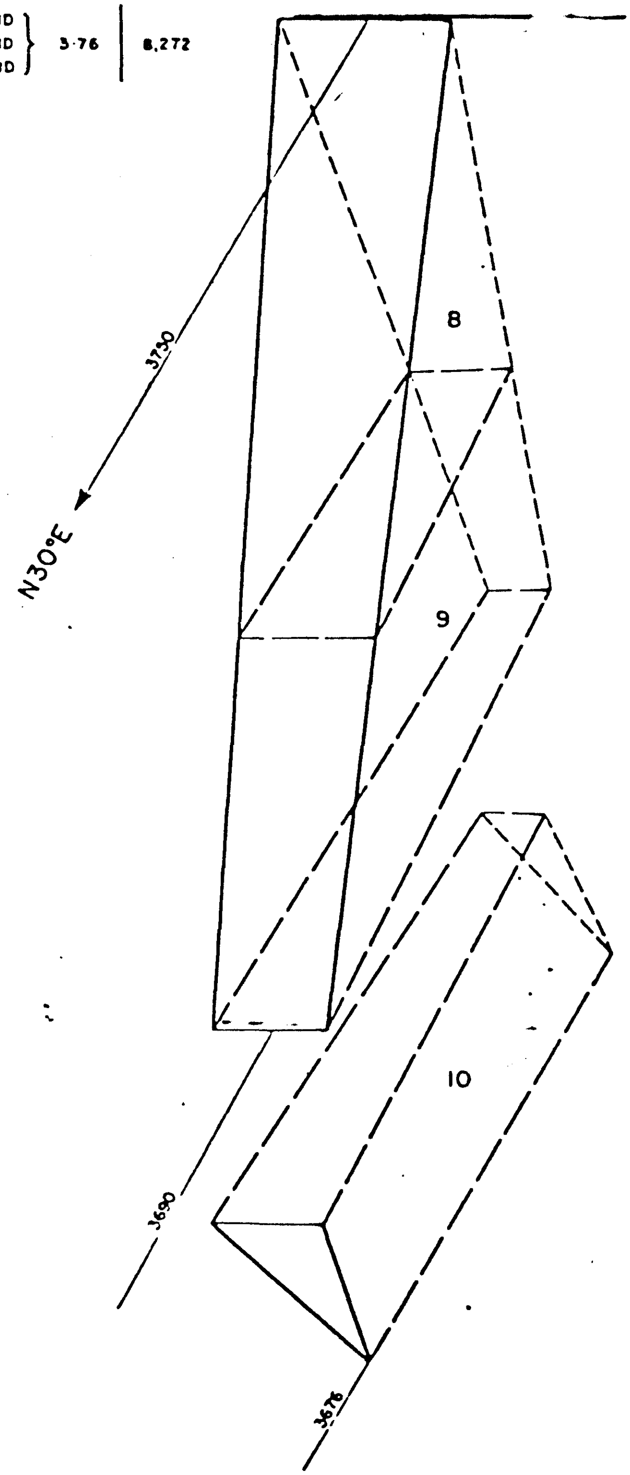
GOTCHA MINERAL CLAIM

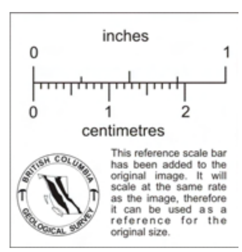
FEBRUARY 1978.

BLOCK	ESTIMATED TONNAGE	CLASS	ASSAY % WO_3	SHORT TON UNIT EQUIVELANT
8.	700 T	DR IND	3·76	8,272
9.	1300 T.	DR IND		
10	200 T.	DR IND		

DRILL IND 5200 TONS
PROBABLE 900 TONS
POSSIBLE 1110 TONS







NCA MINERALS CORP.

ESTIMATED TONNAGE FOR LOWER BAND

GOTCHA MINERAL CLAIM

FEBRUARY 1978.

APPENDIX "A"

Percussion Drilling

Lower Band

Hole No.	Bearing °	Dip °	Depth in ft.
1		vert.	70
2		vert.	80
3	256	56	75
4	270	66	65
5	310	70	80
6	214	70	90
7	N	48	50
8	N	55	45
9	N	51	95
10	30	60	55
11	230	52	80

Upper Band

Hole No.	Bearing °	Dip °	Depth in ft.
12	310	60	25
13	236	50	30
14	292	54	20
15	186	54	20
16	290	50	30
17	270	43	35
18	340	55	25

): C

NCA Minerals Corporation

P.O. Box 371

Vancouver, B.C.

V6C 2N2

ttention: Mr. D. McLeod

can test ltd.
1650 PANDORA STREET, VANCOUVER, B.C. V5L 1L6

Teleph 254 7238
54210

Certificate of Assay

File No. 4575C 1 of 3

Date Feb.26/78

We hereby Certify that the following are the results of assays made by us upon submitted Percussion Drill Cores samples.

Sample Identification	GOLD	SILVER	Tungsten					
	Ounces Per Ton	Ounces Per Ton	Percent WO_3	Percent	Percent	Percent	Percent	Percent
mposite 1 PH 1 20'-30'			0.03					
2 30'-50'			1.58					
3 50'-60'			0.85					
4 PH 2 15'-20'			0.03					
5 20'-25'			0.03					
6 25'-75'			0.04					
7 PH 5 10'-30'			0.57					
8 30'-45'			0.14					
9 30'-45'			0.03					
10 PH 7 25'-35'			0.03					

Note: Pulps retained three months.

Rejects retained two weeks.

Form No. 13 C

CAN TEST LTD.

Provincial Assayer

To: NCA Minerals Corporation

P.O. Box 371

Vancouver, B.C.

V6C 2N2

Attention: Attn: Mr. D. McLeod

can test ltd.
1650 PANDORA STREET, VANCOUVER, B.C. V5L 1L6
Telephone 254 7278
454210

Certificate of Assay

File No. 4575C

Date Feb.26/78

We hereby Certify that the following are the results of assays made by us upon submitted Percussion Drill Cores samples.

Sample Identification	GOLD	SILVER	Tungsten					
	Ounces Per Ton	Ounces Per Ton	Percent WO_3	Percent	Percent	Percent	Percent	Percent
Composite 11 PH 7 35'-50'			1.07					
12 PH 9 25'-50'			4.30					
13 50'-75'			5.07					
14 75'-95'			1.47					
15 PH 4 45'-60'			0.06					
16 PH 10 40'-50'			0.89					
19 PH 11 10'-30'			2.84					
18 35'-60'			0.12					
17 60'-75'			0.10					
20 PH 12 0'-10'			1.12					

Note: Pulps retained three months.

Rejects retained two weeks.

Form No. 13-C

CAN TEST LTD.

Provincial Assayer

To: NCA Minerals Corporation

P.O. Box 371

Vancouver, B.C.

V6C 2N2

Attention: Attn: Mr. D. McLeod

can test ltd.

1650 PANDORA STREET, VANCOUVER, B.C. V5L 1L6

Telex ... 254 7278 ... 54210

Certificate of Assay

File No. 4575C 3 of 3

Date Feb.26/78

We hereby Certify that the following are the results of assays made by us upon submitted Percussion Drill Cores samples.

Sample Identification	GOLD	SILVER	Tungsten		Specific			
	Ounces Per Ton	Ounces Per Ton	Percent WO_3	Percent	Percent	Percent	Percent	Percent
					Gravity			
Composite 21 PH 12 0'-20'			1.25					
22 PH 13 0'-15'			3.23					
23 PH 14 0'-10'			3.05					
24 PH 17 0'-15"			0.06					
25 15'-30'			0.35					
26 PH 18 5'-20'			0.14					
27 PH 3 70'-75'			0.03					
28 General Sample			0.95					
29 Rock			2.40		2.97			

Note: Pulps retained three months.

Rejects retained two weeks.

Form No 13 C

CAN TEST LTD.

Provincial Assayer

APPENDIX 'C'

PRELIMINARY COST STUDY
GOTCHA SCHEELITE PROPERTY

(1) Ore Reserves - drill indicated only = 16,000 stu[1]

Current market value @ $160.00 Can/S.T.U. = $2,560,000

		75% Rec.	85% Rec.
(2)	Recoverable Value	$1,920,000	$2,176,000
(3)	Less 10% Royalty	192,000	217,600
(4)	Net Value to company	$1,728,000	$1,958,000

(5) Capital Costs

Mill complete		$ 45,000[2]
Power Plant		15,000[3]
Import duty, mill		6,750
Dismantling of mill		11,000
Transport of mill - & crane rentals		12,000
Mill building		25,000
Installation of equipment		15,000
Site Preparation		15,000
Service Vehicle		6,500
	TOTAL	$151,250
Contingencies		23,750
	TOTAL	$175,000

Capital Costs will be amortized in one year which is the expected life of the mine. On the bases, net value of ore to company after return of capital will be -

	75% Rec.	85% Rec.
Value before amortization	$1,728,000	$1,958,400
Less Capital recovery	175,000	175,000
Net after capital rec.	$1,553,000	$1,783,400

Estimated ore to be mined to realize above net = 15,000 tons

Net value per ton = (75% rec.) \$103.53 (85% rec.) \$118.89

(6) <u>Mining Costs</u> (100 tons/day)

Assume 25% of tonnage as open pit and 75% underground

<u>Contract mining costs</u>[4]

Open pit @ \$12.00/ton

Underground @ \$25.00/ton

Total cost of mining 15,000 tons on above ratio

$= 15{,}000 \times \frac{25}{100} \times 12 + 15{,}000 \times \frac{75}{100} \times 25$

$= 45{,}000 + 281{,}250 = \$326{,}250$

Mining cost per ton (average) = \$21.75

(7) <u>Milling Costs</u> (100 tons/day)

1 labour, operators, 2 men/shift, 3 shifts	
= 6 men-shifts/day @ \$60/man/shift with benefits	\$360/day
1 Supervisor @ \$2,000/month	67
1 Mechanic @ \$65/day	65
L Electrician @ \$65/day	65
	\$557/day
Contingencies @ overtime	43/day
Total	\$600/day

	Per Ton
Labour cost per ton milled at 100 tons/day	\$ 6.00
Power - 125 Kw at 75% load @ \$.063/kwh [5]	
Cost/24 hrs. = $25 \times .063 \times \frac{75}{100} \times 125$	141.75
Cost/ton milled = $\frac{141.75}{100}$ = 1.4175	(\$1.45)
Mill maintained and supplies - estimated	0.50
Total direct mill costs	\$ 7.95
Say	(\$8.00)

(8) Total Direct Costs

Mining & Milling = \$21.75 + \$8.00	\$ 29.75
	(\$30.00)

(9) Mine Development

Allow 200' drift, x-cut @ \$100/ft.	\$20,000.00 [6]
200 hrs stripping @ \$50/hr.	\$10,000.00
TOTAL	\$30,000.00

Cost per ton of ore = $\frac{30,000}{15,000}$ = \$2.00

(10) Overhead - Administration, Legal, etc.

Provisionally allow \$5.00/ton	\$ 5.00
Total operating costs/ton exclusive of taxes	
21.75 + 8.00 + 2.00 + 5.00 =	\$ 36.75
Indicated net profit before taxes	
75% rec. basis = 103.53 - 36.75 =	\$ 66.75
85% rec. basis = 118.89 = 36.75 =	\$ 82.14

Indicated profit before taxes for total of 15,000 tons	
75% rec. basis = 66.75 x 15,000 =	$1,001.250.00
85% rec. basis = 82.14 x 15,000 =	$1,232,100.00

Summary of Pre Production Financing

Estimated time required to complete mine preparation, erection of mill, etc. to production start up is 120 days

Pre-production Cash Outlay

Purchase of capital equipment and plant construction	$175,000.00
Pre-production stripping	10,000.00
Engineering, administration, etc. @ $5,000/month	15,000.00
TOTAL	$200,000.00

February 20th, 1978 J. P. ELWELL, P.Eng.

Footnotes
(1) Report dated February 14, 1978
(2) Firm option at this price
(3) Finning Tractor quote
(4) Quote - mining contractors
(5) Finning Tractor quote
(6) Quote by mining contractors

A P P E N D I X   E

BRODIE HICKS ENGINEERING LTD.
SUITE [illegible]
[illegible] WEST PENDER STREET
VANCOUVER, B.C. CANADA [illegible]
TELEPHONE [illegible]

April 5, 1978

Mr. F. Axel Mehrle,
Zentnerstrasse 19,
8000 Munich 40
West Germany

Dear Sir:

<u>Interim Report On Gotcha Property</u>

Further to our telephone conversation and your subsequent telex of March 23, I have proceeded with an investigation of the subject property. This has included a review of existing reports, detailed discussions with Mr. A.D. McLeod, President of NCA Minerals Corporation and with Mr. J.P. Elwell, P. Eng., the company's consulting engineer, and a visit to the property accompanied by both of these gentlemen. I have also arranged for an inspection of a mill in California which is available for the project and have been in touch with the metallurgical consultants who are currently carrying out milling tests.

In general, my opinion is favourable. However there are a number of points which must be cleared up before a firm recommendation can be made. I have discussed these with the Principals of the company who are taking the necessary steps to resolve them. Once this has been done, and if results are satisfactory, a decision to proceed can be made. In the meantime, I am writing this interim report to keep you advised of the current situation.

The following paragraphs answer the specific questions of your telex:

A. <u>Existing Reports</u>

I have reviewed the report of D.L. Cooke, P. Eng., written for Union Carbide Ltd. in March 1973; that of United Mineral Services Ltd., the vendor, written on their own behalf in May 1977, and several reports of J.P. Elwell, P. Eng., consulting engineer to NCA Minerals, the latest dated March 2, 1978. All of these have been carefully written to good engineering standards. While, as will appear below, I am inclined to adopt a somewhat more conservative approach, their conclusions are generally acceptable.

Cont'd . . ./2

B. General Opinion Of Property

My opinion is favourable. The situation is unusual. For the time being, at least, no consideration is being given to the customary long-term mining operation. The plan is to move in with a small production facility, as cheaply as possible, and to mine out the readily accessible mineral quickly. The plan is to operate for part of the year only, shutting down for perhaps four months in the winter to avoid the cost of winterizing buildings. I am in agreement with all of this. It may be that in the course of the work, a potential for increased reserves may appear which would prolong the life of the project but such a possibility is not a factor to be considered at this time.

The physical aspects of the deposit are favourable to minimum capital and operating costs.

C. Potential Economic Recoverable Reserves

Mr. Elwell's estimate shows drill-indicated ore reserves as 8,200 tons grading 2.2% WO_3.

These reserves have not been as completely delineated as one would wish. Normally, additional development would be carried out to up-grade the reserves from the "indicated" to the "proven" category. This is, in fact, recommended by Mr. Elwell in his report of March 2, 1978. However, in view of the high grade of the material, it appears that capital expenditures could be written off over a tonnage less than 50% of the presently indicated reserves. This is almost certainly present and under these circumstances your proposed investment appears reasonably secured.

On the basis of present knowledge, an average grade of 2% WO_3 may be assumed.

D. Economic Aspects

Discussed below.

E. Funds Necessary To Start Mine

In his latest report, Mr. Elwell estimates preproduction cost at $200,000. Without myself undertaking a full feasibility study, it is not possible to criticize this figure in detail but, based on experience, I am of the opinion that it is too low, perhaps by a factor of 50%, so that it would be more conservative, and probably more realistic, to use a figure of

E. Funds Necessary To Start Mine (Cont'd)

$300,000. Further, there has been no provision for working capital which may run to as much as $120,000 per month. Once a saleable concentrate is being produced it should not be difficult to borrow working capital but provision should be made for a minimum of one month before this will become possible. Thus, total preproduction outlay may well become $420,000 and could be higher.

F. Operating Costs

Mr. Elwell estimates operating costs at $37.35 per ton. His figures are reasonably conservative and are acceptable. For general estimating, and to provide a contingency factor, $40 per ton may be used.

Summary

Assuming a 75% recovery factor and various prices per short ton unit of WO_3, the following appears:

	$160/Ton	$140/Ton	$120/Ton
Value Per Ton Ore	$240	$210	$180
Less, Royalty, 10%	24	21	18
	$216	$189	$162
Operating Cost/Ton	40	40	40
Operating Profit/Ton	$176	$149	$122
Tons Required To Write Off $300,00 Capital	1,700	2,000	2,500

The above is an over-simplification, ignoring taxes, interest, and so forth but it serves to indicate that presently estimated capital costs can be written off against less than half of the estimated reserves even at tungsten prices substantially below present figures.

Pre-Financing Requisites

1. Metallurgy. At the present time there is no assurance that the mineral can be processed so as to effect a satisfactory recovery (assumed herein as 75%) or so as to produce a saleable concentrate (assumed as 60% WO_3, subject to rejection for certain impurities). Metallurgical testing is now being done and the laboratory advises that preliminary indications

Pre-Financing Requisites

Metallurgy (Cont'd)

are encouraging. If the ore responds to simple treatment, no difficulty will exist but, if not, a longer period of testing and possibly a more complicated mill flowsheet may be required. In any event, the question must be resolved before any major expenditure is made.

2. Mill. NCA Minerals hold an option on an existing, operating mill in California at a quoted price of $45,000. Capacity is stated to be in excess of 100 tons per day. At my request, this plant has been examined by senior engineers of Kilborn Engineering (B.C.) Ltd. A copy of their report is attached. It will be noted that their general impression is favourable but that no recommendations can be made until the metallurgical testing has been completed.

3. Timing. The Contract between NCA Minerals and the vendors, United Mineral Services Ltd., calls for production to be achieved by November of this year. I do not believe this to be possible. The chief problem is the necessity of satisfying the various government regulatory authorities particularly in such areas as pollution control, reclamation and tailings disposal. The necessary negotiations sometimes require up to a year but in the case of a small mine in a remote area, should require less. However, a number of months may be necessary.

I have discussed this with Mr. MacLeod, of NCA Minerals, who is presently negotiating with the vendors for the necessary extension of time. This is an absolute necessity.

4. Sales Contract. There should be no difficulty in selling the concentrates but negotiations, presently under way, should be pushed to a conclusion. It will be important to know how quickly preliminary payments for shipments will be received in order to be able to estimate more accurately the amount of working capital required.

5. Capital Over-Run

It has been noted that capital requirements plus operating capital may be substantially in excess of your proposed investment. Before proceeding, it will be necessary to ensure that the additional money is provided for. A half-completed mine and mill complex is no value.

Cont'd . . ./5

CONCLUSION

As mentioned above, this is merely an interim report as a final, conclusive report cannot be written until all of the problems are resolved. If you so wish, I will be pleased to keep in touch with the situation and advise you of progress from time to time.

Respectfully submitted,

BRODIE HICKS ENGINEERING LTD.

H. Brodie Hicks, P. Eng., M. Eng.

HBH/sg

encl.

APPENDIX F

BACON, DONALDSON & ASSOCIATES LTD.
Consulting Engineers

117 East 4th Avenue · Vancouver, B.C. V5T 1G4 · Telephone 876-5507

File No: 1714

May 9th, 1978

United Mineral Services Ltd.,
1326 - 510 West Hastings St.,
Vancouver, B. C.
V6B 1L8

Dear Sirs,

Re: Testing of Scheelite Ore

We have carried out metallurgical testing on a sample of tungsten bearing rock which was delivered to our office by R. A. Dickinson on April 19th, 1978.

The sample consisted of a number of pieces of rock less than 10 inches in diameter. The entire sample was crushed to minus 1/4 inch prior to being riffled to obtain test samples.

Tabling was carried out with 8 kilograms of sample which had been ground in a laboratory rod mill to minus 20 mesh. The size analysis of the sample was 32 percent minus 200 mesh and 44 percent plus 100 mesh.

The sample was found to contain 2.87% WO_3. The results of tabling are as follows:

PRODUCT	WT.%	% WO_3	WO_3 DISTRIBUTION
Concentrate	4.1	49.74	71.1
Middlings	7.8	1.42	3.9
Slimes	4.5	2.39	3.8
Tails	83.6	0.73	21.3

The scheelite and gangue minerals were readily distinguished due to a marked color difference. The split between concentrate and middling was made to achieve high recovery with a minimal sacrifice to concentrate grade. The losses to the tailing consists of fine scheelite. Additional recovery could likely be achieved by treating the fines separately or by flotation of the gravity tails. For the small operation being contemplated (i.e. 20 tpd) this does not appear to be warranted.

... 2

May 9th, 1978 File No: 1714

The most critical parameter for maintaining recovery in the plant will be the size distribution of the table feed. The feed should be minus 20 mesh to table effectively but overgrinding should be avoided. The mill must not be oversized unless greater throughput can be accommodated. A 3' x 5' ball mill or rod mill would be adequate for the grinding stage. A peripheral discharge mill would be preferrable but as these may be hard to come by a trommel screen should be provided on the mill discharge to remove oversize material.

We will be pleased to provide additional assistance when the need arises.

Yours truly,

BACON, DONALDSON & ASSOCIATES LTD.

M. Vreugde

M. J. Vreugde, P. Eng.

BACON, DONALDSON & ASSOCIATES LTD.
Consulting Engineers
117 East 4th Avenue · Vancouver, B.C. V5T 1G4 · Telephone 876-5507

File No. 1779

July 12, 1978

United Mineral Services Ltd.
1326 - 510 West Hastings Street
Vancouver, B. C.
V6B 1L8

Dear Sirs,

Re: Flotation Testing of Scheelite Ore

We have carried out additional testing of the scheelite ore which we had previously tested by tabling (report dated May 9, 1978). The present testing was designed to test the amenability to recovery by flotation.

The first test was carried out at a grind of 48.5% minus 200 mesh. A rougher concentrate followed by a scavenger concentrate were removed. The results appended to this report show that although a high recovery was achieved the average concentrate grade is only approximately 10% WO_3. The remainder of the concentrate consisted of fine gangue minerals. The use of cleaner flotation stages should improve the concentrate grade. Also, increased sodium silicate additions should result in improved concentrate grades.

The second test was carried out at a grind of 31.5% minus 200 mesh. The coarse scheelite was removed by jigging prior to flotation. The rougher flotation concentrate was refloated to give a clean concentrate. An improved concentrate grade over the first flotation test was achieved but additional depression of gangue minerals is required. It is not expected that concentrate grades similar to those achieved by tabling (50% WO_3 and better) will be achieved by flotation. Concentrate grades approaching 40% WO_3 should be possible by adequate reagent additions and control of grind however.

Yours truly,

BACON, DONALDSON & ASSOCIATES LTD.

M. Vreugde

M. J. Vreugde, P. Eng.

MJV/gd

Enclosures

TEST NO. 1779-1

Flotation test at 48.5% -200 mesh.

STAGE	TIME (minutes)	ADDITIONS
Grinding	15	1 lb./ton Soda Ash 0.2 lb./ton Sodium Silicate
Condition	3	pH = 9.5 0.1 lb./ton PAMAK C4
Flotation	10	--
Scav. Flotation	8	0.1 lb./ton PAMAK C4

RESULTS

PRODUCT	WEIGHT %	% WO_3	% WO_3 Recovery
Rougher Conc.	18.4	10.16	57.3
Scav. Conc.	11.9	9.64	35.2
Tailing	69.7	0.35	7.5

TEST NO. 1779-2

Jigging and Flotation test at 31.5% -200 mesh.

STAGE	TIME (minutes)	ADDITIONS
Grinding	5	--
Jigging	--	--
Condition	5	0.2 lb./ton sodium silicate Soda Ash to pH = 9.5 0.15 lb./ton PAMAK C4
Rougher Float	10	--
Cleaner Float	3	--

RESULTS

PRODUCT	WEIGHT %	% WO_3	% WO_3 RECOVERY
Jig Conc.	5.8	15.87	34.9
Float Conc.	5.0	24.97	47.3
Cleaner Tail	0.8	11.13	3.4
Tailing	88.4	0.43	14.4

BACON, DONALDSON & ASSOCIATES LTD.
Consulting Engineers

117 East 4th Avenue · Vancouver, B.C. V5T 1G4 · Telephone 876-5507

File No: 1779

August 30th, 1978

United Mineral Services Ltd.,
1326 - 510 West Hastings St.,
Vancouver, B. C.

Attention: Mr. M. McClaren

Dear Sir,

Re: Flotation Testing of Tungsten Ore
Progress Report Number 2

We have carried out an additional flotation test of your scheelite ore. The purpose of this test was to produce a concentrate by flotation only using increased cleaning stages over test 1779-2.

The test procedure and results are appended to this report. A rougher flotation followed by three cleaning stages was carried out. A concentrate assaying 35.92% WO_3 and recovering 86.4 percent of the tungsten was produced. The addition of hydrochloric acid to the concentrate indicated that calcite was a significant impurity.

Two possible methods for reducing the calcite content of the concentrate, and thereby increasing its grade, present themselves. One possibility is to use a depressant such as quebracho in the cleaner flotation circuit. The other possibility is to leach the final concentrate with cold hydrochloric acid. While the addition of depressant to the flotation circuit is a more simple approach the quantity of quebracho used must be carefully controlled. The addition of 0.05 lbs. per ton would be a good starting point. If excessive additions are made, depression of tungsten minerals could result.

A proposed flowsheet based on using the Chaput Mill is also enclosed. The soda ash and sodium silicate should be

... 2

---

added to the ball mill feed. The PAMAK can be added to the classifier overflow. Additional sodium silicate should be added to at least the third cleaner feed.

Yours truly,

BACON, DONALDSON & ASSOCIATES LTD.

M. Vreugde

M. J. Vreugde, P. Eng.

TEST 1779-3

PROCEDURE

STAGE	TIME (Minutes)	ADDITIONS
Grinding	10	1 lb./ton Soda Ash 0.5 lb./ton Sodium Silicate
Condition	2	0.1 lb/ton PAMAK 4 pH = 9.8
Rougher Float	10	-
1st Cleaner	10	-
2nd Cleaner	6	0.05 lb./ton Sodium Silicate
3rd Cleaner	4.5	0.05 lb./ton Sodium Silicate

SIZE ANALYSIS

Flotation feed = 36% minus 200 mesh

RESULTS

PRODUCT	WT.%	%WO_3	WO_3 DISTRIBUTION
Concentrate	6.7	35.92	86.4
3rd Cleaner Tail	3.4	1.87	2.3
2nd Cleaner Conc. (Calc.)	10.1	24.5	88.6
2nd Cleaner Tail	5.8	2.02	4.2
1st Cleaner Conc. (Calc.)	15.9	16.3	92.8
1st Cleaner Tail	2.9	3.24	3.4
Rougher Conc.(Calc.)	18.8	14.3	96.2
Rougher Tail	81.2	.13	3.8
Head (Calc.)	100	2.79	100

OFFICE OF THE CHIEF ANALYST AND ASSAYER

DEPARTMENT OF MINES AND PETROLEUM RESOURCES
VICTORIA

SAMPLE RECEIVED FROM E. W. GROVE

ADDRESS Geological Division

SEMI QUANTITATIVE SPECTROGRAPHIC ANALYSIS

Laboratory No.	19689M					
Submitter's No.	WHOLE ROCK					
Si	>10.0					
Mn	0.4					
Al	7.0					
Mg	0.6					
Pb	0.04					
Ca	>15.0					
Fe	7.0					
V	0.015					
Cu	0.015					
Ag	T					
Zn	0.025					
Na	0.05					
K	T					
Ti	0.15					
Zr	T					
Ni	T					
Co	T					
Sr	T					
Cr	T					
Ba	T					
Traces:	Bi,Ga,Mo					
As	0.04					

Sn	0.02
Be	0.01
W	>1.0

THIS DOCUMENT, OR ANY PART THEREOF, MAY NOT BE REPRODUCED FOR PROMOTIONAL OR ADVERTISING PURPOSES.

DATE July 5, 1978

W. G. Johnson

CHIEF ANALYST AND ASSAYER

APPENDIX G

Kamloops Research & Assay Laboratory Ltd.

B.C. LICENSED ASSAYERS
GEOCHEMICAL ANALYSTS

2095 WEST TRANS CANADA HIGHWAY—KAMLOOPS, B.C. V1S 1A7
TELEPHONE 372-2784 · TELEX 048-8320

CERTIFICATE OF ASSAY

TO United Mineral Services

Certificate No. 1851

Date 11th Dec.

I hereby certify that the following are the results of assays made by us upon the herein described ____ sampl

Kral No.	Marked	GOLD	SILVER	W						
		Ounces Per Ton	Ounces Per Ton	Percent	Percent	Percent	Percent	Percent	Percent	Percent
1889-1	Heads 15-11 4am	Taken from	NOT	.37						
-2	Heads 15-11 4PM	Pan feeder	RELIABLE	.70						
-3	TAILS 4:01			1.61						
-4	Cyclone Discharge	(HEAD)		2.97						
-5	1ST Cl. Conc. 4:00			21.97						
-6	1st Cl. Conc 3:00			43.41						
-7	Conc 4:00			20.80						

NOTE:
Rejects retained three weeks
Pulps retained three months
unless otherwise arranged.

Davis for K. Gibbs
Registered Assayer, Province of British Columbia

B.C. LICENSED ASSAYERS
GEOCHEMICAL ANALYSTS

Kamloops Research & Assay Laboratory Ltd.

2095 WEST TRANS CANADA HIGHWAY—KAMLOOPS, B.C. V1S 1A7
TELEPHONE 372-2784 · TELEX 048-8320

CERTIFICATE OF ASSAY

TO United Mineral Services

Certificate No. 1811

Date 24th Oct

I hereby certify that the following are the results of assays made by us upon the herein described Chips sample

Kral No.	Marked	GOLD	SILVER	W						
		Ounces Per Ton	Ounces Per Ton	Percent	Percent	Percent	Percent	Percent	Percent	Percent
				.97						
1811 - 1	Coarse									
- 2	Fine			1.09						

NOTE:
Rejects retained three weeks
Pulps retained three months
unless otherwise arranged.

Dyas for R&B

Registered Assayer, Province of British Columbia

APPENDIX H

N.T.S. 82M13E

BOULDER PROPERTY

D.D.H. No. 1.

Azimuth: 150°
Declination: -45°
Collar Elevation: 3692'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
0	14	168	29	17.2	? to 24.5	Medium-grained biotite, quartz monzonite. Minor rusty weathering Pegmatite between 13.5' and 14.5'.
14	24.5	126	25	19.8		
24.5	37	150	82	54.6	24.5 to 51	Medium-grained muscovite and minor biotite quartz monzonite showing rusty weathering, some of which is liesegang weathering.
37	42	60	21	35		
42	44	24	24	100		
44	47	36	8	22.2		
47	49	24	2	8.3		
49	51	24	12	50		
51	53.5	30	30	100	51 to 53.5	Coarse-grained muscovite, quartz monzonite showing rusty weathering
53.5	54	6	6	100	53.5 to 62	Fine-grained muscovite and biotite quartz monzonite showing rusty weathering.
54	55.5	18	6	33.3		
55.5	56.5	12	9	75		
56.5	61	54	29	53.7		
61	62	12	8	66.6		
62	64	24	24	100	62 to 64	Pegmatite vein. Contacts at 20° to core.
64	67	36	33	91.6	64 to 92	Fine-grained muscovite, biotite, quartz monzonite showing rusty weathering with unweathered patche of rock. The latter show a distinct boundary with the iron stai ing giving a crude liesegang effe ≃6" pegmatite vein at ≃71.5'. Surface weathering effects stop more or less at base of this intersection.
67	70	36	34	94.4		
70	72.5	30	22	73.3		
72.5	75	30	23	76.6		
75	77	24	22	91.6		
77	80	36	29	80.5		
80	81	12	12	100		
81	82.5	18	13	72.2		
82.5	84	18	15	83.3		
84	86.5	30	16	53.3		
86.5	88.5	24	19	79.1		
88.5	92	42	39	92.8		

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 1.

Azimuth: 150°
Declination: -45°
Collar Elevation: 3962'
Description by: D. L. Cook.

Feet From	Feet To	Inches Int.	Inches Core Recovered	% Core Recovered	Feet Geol. Interval	Description
92	95.5	42	42	100	92 to 98.5	Fine-grained unweathered quartz monzonite with varying amounts of biotite.
95.5	96.5	12	8	66		
96.5	98.5	24	24	100		
98.5	100.5	24	24	100	98.5 to 107	Gneissose or mildly schistose medium-grained, biotite, muscovite, quartz monzonite.
100.5	103	30	30	100		
103	106	36	30	83.3		
106	107	12	12	100		? Schistocity at 45° to core but less evident in the lower part of the intersection.
107	108	12	10	83.3	107 to 108	Medium-grained muscovite quartz monzonite with variable minor biotite.
108	109.5	18	15	83.3	108 to 114.5	Fine and medium-grained muscovite quartz monzonite.
109.5	111.5	24	6	25		
111.5	114.5	36	11	30.5		
114.5	118	42	34	80.9	114.5-127.5	Fine-grained biotite muscovite quartz monzonite. Rusty weathering along occasional strong fractures.
118	119	12	11	91.6		
119	125	72	72	100		
125	127.5	30	30	100		
127.5	132	54	54	100	127.5-134.5	Fine and medium-grained muscovite quartz monzonite with variable minor biotite. Rusty weathering along strong fractures to 131'.
132	134.5	30	30	100		
134.5	214	954	838	87.8	134.5-250	Fine to medium-grained leucocratic muscovite quartz monzonite. Biotit occurs at 140.5' (aligned), 144' to 147', 185', 187', 189.5' to 190.5' 221.5', 222.5', 229' to 235.5', 240' to 243', 245' to 248'.
214	220.5	78	49	62.8		
220.5	222.5	24	20	83.3		
22.5	226	42	18	42.8		
226	250	288	255	88.5		Higher content of muscovite betwee 169' and 196.5' which may indicate increased alteration.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 1.

Azimuth: 150°
Declination: -45°
Collar Elevation: 3962'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
						Thin muscovite "veins" occur spasmodically along this intersection at various obtuse angles to the core. These "veins" are almost certainly alteration along fractures.
						A set of cleavage occurs 20° to 30° to the core between 207.5' and 209.5'.
						The rock is coarser between 237' and 237.5'.

N.T.S. 82M13E

BOULDER PROPERTY

D.D.H. No. 2.

Azimuth: 150°
Declination: -45°
Collar Elevation: 3779'
Description by: D. L. Cook.

Feet From	Feet To	Inches Int.	Inches Core Recovered	% Core Recovered	Feet Geol. Interval	Description
0	20	240	5	2	0-20	Pebbles of fine-grained biotite granodiorite and fine-grained gneissose chlorite granodiorite. Probably represent boulders in the overburden.
20	23	36	18	50	20-30.5	White and grey quartzite or quartz monzonite with some banded diopside ? skarn evident as pebbles after 23'
23	27	48	11	23		
27	30.5	42	21	50		
30.5	35	54	20	37	30.5-42	Biotite quartz schist. Core/schistocity = 85°.
35	42	84	52	62		
42	47	60	9	15	42-61.75	Quartz pebbles probably of a vein or a silicified skarn.
47	61.75	177	7	3.9		
61.75	74.25	150	150	100	61.75-87	Coarse garnet calcite quartz diopsid skarn. No calcite from 66' to 67' and from 76' to 78' and finely and finely banded over these intervals. Quartz from 74.25' to 77'. Core/banding is 70° - 85°.
74.25	77'	33	2	6		
77	87	120	18	15		
87	89	24	2	8.3	87-89	Biotite quartz schist. Core/schistocity = 20°.
89	97	120	2	1.6	89-108	Crushed quartz and clay. Probable fault zone.
97	108	132	7	5.3		
108	111	36	6	16.6	108-117	Coarse tremolite garnet quartz skarn
111	117	72	26	36		
117	122.25	63	22	35	117-122.25	Fine-grained diopside garnet quartz skarn showing some banding at 65° tc core.
122.25	124	21	21	100	122.25-124	Fractured and altered fine-grained quartz (15%) plagioclase (85%) rock.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 2.

Azimuth: 150°
Declination: -45°
Collar Elevation: 3779'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
124	126.5	30	30	100	124-126.5	Coarse garnet diopside quartz idocrase scheelite skarn Scheelite grade 1% between 124.5' and 126.5'.
126.5	132	66	63	95.5	126.5-132	Fine and coarse garnet diopside quartz calcite skarn with some banding of diopside at 55° to core. Calcite also as veins. 3" interval with scheelite at 128.5'. 1" interval with scheelite at 129.5
132	134	24	13	54	132-134	Fractured and altered fine-grained quartz (10%) plagioclase (90%) rock.
134	140	72	53	73.6	134-140	Fine to coarse-grained diopside garnet quartz skarn with some idocrase between 136.5' and 137.5' Diopside banding = 65° to core. 2" interval of high grade scheelite at 138.5'. Interval between 135' and 136' probably a skarnized limey quartzite.
140	144	48	10	20.8	140-144	Crushed garnet skarn and limey clay. Probably a fault zone.
144	147	36	14	38.9	144-147 147	Medium-grained garnet diopside calcite skarn. Calcite also as veins.
147	149	24	12	50	147-149	Epidotized intrusive.

N.T.S. 82M13E

BOULDER PROPERTY

D.D.H. No. 2.

Azimuth: 150°
Declination: -45°
Collar Elevation: 3779'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
149	157.5	102	55	53.9	149-157.5	Schistose (biotite) quartzite.
157.5	159	18	15	83.3	157.5-159	Fine to coarse-grained garnet diopside quartz skarn.
159	164	60	20	33.3	159-164	Quartz biotite schist. Core/ schistocity = 80°.
164	169	60	35	58.3	164-169	Fractured and altered quartz monzonite.
169	171	24	14	58.3	169-171	Massive quartz.
171	173.5	30	13	43.3	171-179.5	Medium-grained muscovite quartz (50%) plagioclase (50%) rock wit minor pyrite.
173.5	174.5	12	4	33.3		
174.5	176.5	24	20	83.3		
176.5	179.5	36	19	52.7		Fracturing and alteration of feldspars and formation of garne and minor molybdenite has occur between 176' and 179.5'.
179.5	188	102	88	86.2	179.5-188	Schistose biotite muscovite ? quartzite with minor garnet.
188	193.3	64	62	97	188-193.3	Quartz biotite schist. Core/ schistocity = 65°.
193.3	198.2	59	54	91.5	193.3-198.2	Schistose biotite quartzite grading into quartzite. Core/ schistocity = 70°.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 3.

Azimuth: 150°
Declination: -45°
Collar Elevation: 3750'
Description by: D. L. Cook.

Feet From	Feet To	Inches Int.	Inches Core Recovered	% Core Recovered	Feet Geol. Interval	Description
0	15	180	2	1.6	0-15	Overburden.
15	19	48	15	31.2	15-19	Fine-grained banded diopside quartz scheelite skarn with minor garnet and calcite. Core/banding = 60° at 15'. Scheelite grade = 2.06% over the interval.
19	27.3	100	100	100	19-27.3	Medium-grained to fairly coarse garnet diopside quartz scheelite skarn. Fractured and silicified from 24' to 27'. Core/banding=85°. Scheelite grade = 2.86% over the interval. Pyritic from 22.5' to 28'.
27.3	28.5	15	15	100	27.3-28.5	Fine-grained, unbanded diopside/ quartz skarn with minor (?) epidote. Pyritic from 28' to 31'.
28.5	32	42	27	64.3	28.5-32	Medium-grained to fairly coarse garnet diopside quartz skarn with massive quartz at each end (3" & 12' respectively). Not banded. Scheelite grade ≃0.1% over the interval.
32	36.5	54	43	79.5	32-37	Quartz biotite schist.
37	41.5	42	42	100	37-41.5	Coarse-grained crystalline rock; quartz (50%) plagioclase (50%). Upper contact irregular and sharp, lower contact transitional.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 3.

Azimuth: 150°
Declination: -45°
Collar Elevation: 3750'
Description by: D. L. Cook.

Feet From	Feet To	Inches Int.	Inches Core Recovered	% Core Recovered	Feet Geol. Interval	Description
41.5	44.5	36	36	100	41.5-44.5	Unbanded diopside quartz ?-skarn with minor coarse-grained crystalline rock (50% quartz and 50% plagioclase) at 42'.
44.5	47	30	30	100	44.5-100.5	Quartz biotite schist. Quartz ? veins at 69' (3" wide), 85'(4" wide), 88' (4" wide), 94.5' wide).
47	55.5	102	95	93.1		
55.5	57	18	7	38.9		
57	59	24	11	45.8		
59	61	24	10	41.7		Core/schistocity = 50° at 46.5'
61	64.5	42	18	42.9		80° " 55'
64.5	69	54	35	64.8		85° " 65'
69	72	36	4	11.1		75° " 84'
72	77	60	8	13.3		50°→70° " 94'
77	79.5	30	7	23.3		80° " 100'
79.5	85	66	16	24.2		
85	88	36	32	88.9		Muscovite-rich rock with minor quartz and K-feldspar between 77' and 79.5'
88	91	36	12	33.3		
91	93.5	90	12	13.3		
93.5	100.5	84	84	100		Large garnet crystals at 94'.
100.5	120.5	240	198	82.5	100.5-120.5	Quartzite, mildly schistose (bioti[illegible] in places. Very minor pyrite. Core/schistocity = 70° at 110'
120.5	125.5	60	40	66.6	120.5-125.5	Quartz biotite schist. Core/schistocity = 80° at 121'; 70° at 125'.
125.5	131	66	12	18.2	125.5-134	Schistose biotite quartzite with very minor pyrite.
134	135.5	18	12	66.6	134-135.5	Fine-grained muscovite quartz monzonite. No contacts recovered in core
135.5	138.5	36	36	100	135.5-138.5	Quartz biotite schist. Core/schistocity = 60° at 135.5'.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 3.

Azimuth: 150°
Declination: -45°
Collar Elevation: 3750'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
138.5	142	42	2	4.8	138.5-142	Quartz (mostly lost).
142	169	324	294	9017	142-169	Intrusive rocks as follows:

1. Fine to medium grained muscovite plagioclase quartz rock (80%) plagioclase, (20% quartz).
2. Fine-grained biotite quartz monzoni
3. Fine-grained muscovite quattz monzon
4. Fine grained muscovite quartz monzo ite with minor garnet.
5. Coarse-grained muscovite granodiori with minor garnet.
6. Coarse-grained muscovite quartz monzonite.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 4.

Azimuth: 173°
Declination: -45°
Collar Elevation: 3840'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
0	10	120	0	0	0-10	Probably all overburden. ? Boulders of intrusive and garnet, tremolite skarn
10	11	12	9	75	10-11	
10	12	24	22	91.6	11-12 "	Green quartzite and fine-grained diopside, quartz skarn. Core/banding = 70° at 11.5'
12	18	72	64	88.9	12-30.5	Tremolite, garnet (coarse) skarn with minor idocrase and diopside.
18	27	108	104	96.3		
27	30.5	42	37	88		
30.5	35.5	60	54	90	30.5-37.5	Fine grained, partly banded diopside, garnet, quartz skarn with minor idocrase. Core/banding = 70° at 35'. Quartz patches 31'-34' showing digestion of skarn. Minor biotite patches at 33'.
35.5	37.5	24	24	100		
37.5	38.5	12	12	100	37.5-38.5	wollastonite Tremolite, garnet (coarse) skarn.
38.5	40	18	18	100	38.5-40	Coarse garnet, fine diopside, coarse idocrase, quartz skarn partly digested by quartz monzonite.
40	42	24	24	100	40-42	Massive quartz and/or quartz monzonite showing digestion in contact region between schist (pyrite patches) and skarn.
42	45	36	35	97.2	42-51	Quartz, biotite schist. Core/banding = 50° at 45.5'.
45	51	72	19	26.4		

N.T.S. 82M13E

BOULDER PROPERTY

D.D.H. No. 4.

Azimuth: 173°
Declination: -45°
Collar Elevation: 3840'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
51	62.5	138	138	100	51-62.5	Fine-grained banded diopside, quartz rock. Probably skarnized and schistose quartzite. 1" vein of quartz monzonite at 53.5'. (Contacts irregular but sharp.) Coarse garnet idocrase skarn patches between 59' and 62.5'. Core/banding = 50° at 56'.
62.5	73.2	129	129	100	62.5-73.2	Tremolite garnet (coarse) idocrase skarn with calcite predominant over tremolite last 30". Minor banding between 66' and 66.5'. Core/banding = 50° at 66'.
73.2	74.5	16	16	100	73.2-74.5	Fine-grained diopside garnet idocrase skarn.
74.5	75	6	6	100	74.5-75	Quartz biotite schist. Core/schistocity = 55°.
75	75.5	6	6	100	75-75.5	Coarse-grained quartz monzonite with contacts at 55° to core and sharp.
75.5	79	42	42	100	75.5-79	Coarse-grained diopside idocrase garnet quartz skarn with tremolite garnet idocrase skarn between 77' and 78.5'.
79	81.75	33	33	100	79-81.75	Coarse-grained quartz monzonite. Upper contact sharp and 45° to core, lower contact irregular and transitional.
81.75	82.5	9	9	100	81.75-82.5	Coarse diopside idocrase garnet quartz skarn.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 4.

Azimuth: 173°
Declination: -45°
Collar Elevation: 3840'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
82.5	88.5	72	72	100	82.5-88.5	Fine to coarse tremolite garnet, diopside skarn.
						Partly schistose (6") diopside garnet band between 85' and 87.5' Core/schistocity = 50°.
88.5	91	30	24	80	88.5-91	Fine-grained banded diopside quartz skarn.
						3" vein of quartz monzonite at 89.75' with irregular, sharp contacts.
91	93	24	24	100	91-93	Tremolite garnet idocrase skarn
93	97	48	19	39.6	93-105.5	Fine-grained banded diopside quart skarn with minor coarse garnet and minor pyrite.
97	105.5	102	102	100		
						Core/banding = 60° at 98.5'.
						Zones of intrusive between 96.5' and 98'. Contact at 98' = ≃60°, at 103.5' and 104.5' = 55° and transitional at 96.5'.
105.5	109.5	48	48	100	105.5-109.5	Coarse garnet idocrase quartz ska
						Hornblende (?) at 107'.
109.5	114.25	57	57	100	109.5-114.25	Fine-grained/banded diopside qua skarn with extensive silification between 110.5' and 113.5'. Contac transitional. Core/banding =55° all
114.25	117	33	33	100	114.25-117	Coarse garnet idocrase diopside, quartz skarn.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 4.

Azimuth: 173°
Declination: -45°
Collar Elevation: 3840'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
117	119	24	24	100	117-119	Tremolite garnet (coarse) skarn
119	122.75	45	45	100	119-122.75	Coarse garnet idocrase quartz skarn. Minor fine-grained banded diopside skarn at each end. Core/banding = 70° at 122'.
122.75	124	15	15	100	122.75-124	Tremolite garnet skarn
124	132	96	96	100	124-132	Coarse diopside garnet idocrase quartz skarn with minor banding. 4" vein of pyritic coarse-grainec biotite quartz monzonite at 128.5'. Core/banding = 55° at 132'. Sharp contact at 90° to core.
132	134.5	30	30	100	132-134.5	Coarse grained quartz monzonite a quartz. Both contacts transition
134.5	139	54	54	100	134.5-139	Quartz biotite schist
139	141	24	24	100	139-141	Tremolite garnet skarn with 5" intersection of coarse garnet id crase skarn.
141	143.5	30	30	100	141-143.5	Diopside garnet idocrase quartz skarn. Banded (?schistose) and pyritic between 141.3' and 142.3 Core/banding = 65°.
143.5	145	18	18	100	143.5-145	Tremolite garnet skarn.
145	150.5	66	66	100	145-150.5	Fine to coarse garnet idocrase diopside ?chlorite quartz skarn. Not banded. Major strain cleava at 146' = 15° to core.
150.5	153.5	36	36	100	150.5-153.5	Fine-grained banded diopside qu skarn with some coarse garnet. Core/banding = 60° at 153.5'

N.T.S. 82M13E

BOULDER PROPERTY

D.D.H. No. 4.

Azimuth: 173°
Declination: -45°
Collar Elevation: 3840'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
153.5	156.5	36	36	100	153.5-156.5	Medium-grained quartz monzonite. Major strain cleavage at 20° to core.
156.5	163	78	78	100	156.5-163	Fine to coarse garnet idocrase diopside quartz skarn extensively silicified and with chlorite.
163	164.5	18	18	100	163-164.5	Fine-grained banded diopside garnet biotite quartz schistose skar Core/banding = 60°.
164.5	169	54	54	100	164.5-169	Fractured coarse-grained quartz monzonite with minor pyrite
169	172.5	42	42	100	169-172.5	Schistose and skarnized rock (probably originally limey, argillaceous quartzite).
172.5	177	54	54	100	172.5-177	(?) Transitional contact.
177	179	24	24	100	177-179	Biotite quartz monzonite. Lower contact = 65° to core.
179	181.5	30	30	100	179-181.5	Pyritic leucocratic quartz monzonite. Upper contact =65° to co
181.5	188	78	78	100	181.5-188	Leucocratic quartz monzonite.
188	190	24	24	100	188-190	Biotite quartz monzonite.
190	192.5	30	30	100	190-192.5	Pyritic fine-grained banded diopside rock (skarnified) merging into quartz biotite schist. Core banding = 65°.
192.5	201.5	108	108	100	192.5-201.5	Biotite quartz monzonite; well fractured.
201.5	203.5	24	24	100	201.5-203.5	Leucocratic, quartz monzonite. Well fractured.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 4.

Azimuth: 173°
Declination: -45°
Collar Elevation: 3840'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
203.5	207.5	48	48	100	203.5-207.5	Biotite quartz monzonite; well fractured to 205'.
207.5	225	210	210	100	207.5-225	Leucocratic quartz monzonite. 5" of biotite quartz monzonite at ≈215.5' and ≈217'. Small garnets between 215.5' and 219'.
225	235.5	126	126	100	225-235.5	Pegmatite.
235.5	256	234	234	100	235.5-256	Variable fine to pegmatitic leucocratic quartz monzonite. Medium-grained garnets at 238' and between 241.5' and 244'. 1" vein of pyrite 55° to core at 237.5'. One sharp contact noted between 2 of the above intrusives at 241.5' at 60° to core.
256	257.5	18	18	100	256-257.5	Fine-grained leucocratic quartz monzonite.
257.5	267.5	120	120	100	257.5-267.5	Fine to medium-grained biotite quartz monzonite with minor muscovite banding at 30° to core. Fine-grained leucocratic quartz monzonite between 261' and 263.5'.
7.5	270.5	36	36	100	267.5-270.5	Digested and fractured transitional zone between the above intrusive an the following schist.
270.5	277.5	84	84	100	270.5-277.5	Quartz biotite schist. Core/schistosity = 40° at 272.5'.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 4.

Azimuth: 173°
Declination: -45°
Collar Elevation: 3840'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
277.5	285	90	90	100	277.5-285	Schistose biotite quartzite with minor pyrite. Core/schistocity = 55° at 284.5'.
285	290	60	60	100	285-294	Quartz biotite schist with small intersections of quartzite throughout interval. Core/schistocity = 65° at 294'.
290	294	48	20			
294	300	72	9	12.5	294-303	Crushed and gouged schist. Probably a faulted area.
300	303	36	36	100		
303	307.5	54	54	100	303-307.5	Quartz biotite schist and quartzite
307.5	320	150	150	100	307.5-320	Mildly schistose muscovite quartzit Core/schistocity =40° at 312'.
320	321	12	12	100	320-321	Schistose biotite quartzite.
321	324	36	10	27.8	321-324	Brecciated quartzite with calcite cement.
324	334	120	14	11.6	324-336	Quartzite showing strong strain cleavage 20° to core.
334	336	24	9	37.5		
336	341.5	66	64	97	336-350	? Epidote quartz monzonite. Most ?epidote between 337' and 341.5'.
341.5	344.5	36	36	100		
344.5	347.5	36	25	69.4		
347.5	348	6	1	16.6		
348	350	24	24	100		Brecciated with calcite cement at 342'.
350	350.1	1	1	100	350-350.1	Quartz biotite schist. Core/schistocity = 60° at 350'.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 5.

Azimuth: 170°
Declination: -45°
Collar Elevation: 3765'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
?0	?22.5	270	15	5.5	?-?22.5 15.9	Medium-grained biotite quartz monzonite-
22.5?	30	90	84	93.3	?.22.5-36.5 9.9	Mainly limey garnet idocrase, diopside, tremolite skarn. (garnet and idocrase coarse). Longest tremolite interval between 35.5' to 36.5'. Calcite absent from: 22.5' to 24' 26' to 26.5' 33' to 36.5'
30	33	36	36	100		
33	35.5	30	10	33		
35.5	36.5	12	12	100		
36.5	40	42	35	83.3	36.5-40 3.5	Banded diopside, quartz, garnet skarn becoming schistose last 2'. Banding at 65° to core.
40	45	60	56	93.3	40-45 5	Siliceous skarn and banded diopside skarn. Minor coarse garnet. Banding at 65° to core.
45	47.5	30	26	86.6	45-47.5 2.5	Limey diopside, garnet, idocrase skarn.
47.5	49	18	18	100	47.5-49 1.5	Banded diopside, quartz skarn. Banding at 55° to core.
49	50.25	15	15	100	49-50.25 1.25	Limey diopside, garnet idocrase skarn.
50.25	52.75	30	30	100	50.25-52.75 .5	Banded diopside, quartz skarn with coarse garnet and idocrase at each end. Tremolite last 2". Banding at 70° to core.
52.75	63.5	131	116	88.5	52.75-63.5 9.75 23.96	Banded diopside, quartz skarn with unbanded coarse garnet, idocrase patches and the following:

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 5.

Azimuth: 170°
Declination: -45°
Collar Elevation: 3765'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
						Tremolite garnet skarn between 61.5' and 62' and between 62.5'and 63'. Limey diopside, garnet, idocrase skarn between: 52.75' and 53' 56.25' and 57' 61' and 61.5' 62' and 62.5' Banding at 60° to core at 60'.
63.5	66.5	36	36	100	63.5-66.5 3	Fine to coarse-grained leucocratic quartz monzonite. Contact with skarn is fairly sharp at 45° to cor
66.5	68.5	24	24	100	66.5-68.5 2 3.5	Medium-grained biotite quartz monzonite. Contact with leucocratic quartz monzonite is gradational and at 45° to core. The leucocratic quartz monzonite appears to be digesting the biotite quartz monzonit
68.5	70.5	24	17	70.8	68.5-70.5 2	Cavernous limey garnet, idocrase, diopside skarn. (garnet and idocrase coarse). Minor scheelite at 70'.
70.5	72.25	21	21	100	70.5-72.25 1.75	Coarse garnet, diopside skarn.
72.25	74.5	27	18	66.6	72.25-74.5 2.25	Quartz
74.5	76.3	22	22	100	74.5-80 5.5	Medium to coarse-grained leucocratic quartz monzonite with fragments and small intervals of coarse garnet, diopside, idocrase skarn. These latter show evidence of digestion at their margins. Contact o the quartz monzonite with the previously described interval (quartz) is 60° to the core. Schistocity at 79' =65°.
76.3	80	44	26	59		

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 5.

Azimuth: 170°
Declination: -45°
Collar Elevation: 3765'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
80	83.5	42	33	78.5	80-83.5 3.5	Quartz-biotite schist. Schistocit at 40° to core at 83'.
83.5	87.5	48	42	87.5	83.5-87.5 4	Medium grained leucocratic and biotite quartz monzonite with small intervals of biotite schist and banded ? diopside ? skarn. The margins of the metasedimentar rocks show digestion by the quart monzonite. Banding at ≈30° to core suggestir disorientation of the metasedimer by floating in the intrusive melt
87.5	91.5	48	37	77	87.5-91.5 3	Banded diopside, quartz, biotite ?skarn with fine-grained garnet, diopside, idocrase, quartz skarn between 89.5' and 90.25'. Banding at 55° to core.
91.5	92.5	12	10	83.3	91.5-92.5 1	Quartz-biotite schist. Schistoci at 50° to core.
92.5	100	90	81	90	92.5-100 7.5	Medium to coarse-grained leuco-cratic quartz monzonite. Upper contact 45° to core.
100	102.25	26	26	100	100-102.25 2.25	Massive coarse pyroxene or amphi bole with minor pyrite.
102.25	111	99	98	98.9	102.25-111 8.75	Medium-grained biotite quartz mo zonite. Upper contact 80° to co Bands of biotite last 2' probabl represent last stage of digestio of a xenolith.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 5.

Azimuth: 170°
Declination: -45°
Collar Elevation: 3765'
Description by: D. L. Cook.

Feet From	Feet To	Inches Int.	Inches Core Recovered	% Core Recovered	Feet Geol. Interval	Description
111	116	60	50	83.3	111-116 5	Fine to medium-grained grey muscovite quartz monzonite.
116	131.75	177	174	98.5	116-131.75 15.75	Coarse quartz-rich garnet, diopsic idocrase skarn. Vuggey and ferruginous from 116' to 120'. Minor limey patches. Coarse scheelite >0.5% between 116.3' and 116.5'. Banded diopside, quartz skarn between 127.75' and 128.25'. Ferruginous and gossany between 119' and 120'.
131.75	141	117	117	100	131.75-141 9.25	Quartz-biotite schist. Schistocity to core at 137.5' = 65°.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 6.

Azimuth: 173°
Declination: -45
Collar Elevation: 3802'
Description by: D.L. Cook.

Feet From	Feet To	Inches Int.	Inches Core Recovered	% Core Recovered	Feet Geol. Interval	Description
0	9.5	66	6	9.1	0-9.5	Overburden with schist, pegmatite and quartz monzonite boulders.
9.5	18	102	102	100	9.5-18	Medium-grained muscovite quartz monzonite. Strong fracturing at 20° to core.
18	25.5	90	90	100	18-25.5	Medium-grained quartz monzonite with coarse muscovite. Coarse biotite between 22' and 23'.
25.5	33	90	90	100	25.5-33	Porous and rusty medium-grained muscovite biotite quartz monzonite.
33	37	48	42	87.5	33-60	Muscovite quartz feldspar pegmatite. Rusty.
37	39	24	12	50		
39	49	120	120	100		
49	56.5	90	78	86.6		
56.5	60	42	39	92.8		
60	64.5	54	54	100	60-64.5	Leucocratic fine-grained quartz monzonite with variable muscovite.
64.5	71	78	78	100	64.5-71	Medium-grained muscovite-biotite quartz monzonite. Rusty along strong fractures
71	72.6	20	20	100	71-72.6	Pegmatite
72.6	78.25	68	54	79.4	72.6-78.25	Medium-grained muscovite-biotite quartz monzonite. Rusty along strong fractures.
78.25	80.75	30	19	63.3	78.25-80.75	Pegmatite

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
80.75	104	279	261	93.5	80.75-104	Medium-grained leucocratic quartz monzonite with minor muscovite and minor patches of biotite. High muscovite last 2'. Rusty along strong fractures.
104	109	60	14	23.3	104-118	Medium to coarse-grained muscovite rich quartz monzonite showing fracturing, crush and gouge increasing in intensity along the interval.
109	112	36	11	30.5		
112	114	24	14	58.3		
114	116	24	24	100		
116	118	24	19	79.1		
118	119	12	12	100	118-119	Medium-grained leucocratic quartz monzonite. Rusty along strong fractures.
119	144	300	294	98	119-144	Medium-grained biotite quartz monzonite. Rusty staining and weathering of biotite between 134' and 144'.
144	146.5	30	30	100	144-146.5	Medium-grained biotite-rich granitic. Probably xenolithic materia
146.5	160.5	168	168	100	146.5-160.5	Medium to coarse-grained biotite quartz monzonite with patches of biotite-rich material and patches of schistose, mica-rich material which are probably xenolithic.
160.5	168	90	90	100	160.5-168	Medium to coarse-grained muscovite quartz monzonite. Irregular lower contact at 25° to core.
168	170	24	24	100	168-170	Medium-grained biotite quartz monzonite.
170	173	36	36	100	170-173	Medium to coarse-grained quartz monzonite with variable biotite (some of which shows alignment) and muscovite. Probably xenolithic Rust staining and alteration along strong fractures.
[illegible]	177	48	42	87.5	173-189.5	Medium to coarse-grained biotite quartz monzonite. Highly fractured and rust stained between 173' to 175' & between 177' to 182'.
177	179	24	18	75		
179	181	24	18	75		
181	182	12	11	91.6		
182	185	36	32	88.8		
185	189.5	54	48	88.8		

Hole No. 6.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
189.5	191.5	24	11	45.8	189.5-191.5	Mylonite, crush and gouge of a fault.
191.5	196	54	48	88.8	191.5-196	Fine to coarse-grained muscovite quartz monzonite.
196	197	12	8	66	196-212	Quartz-biotite schist with some grey clay at 196', 197' and 207'. The intervals 199' to 206' and 211' to 212' show distinct digestion of the schist by quartz-rich intrusive and leucocratic quartz monzonite. Minor pyrite at 204.5'. Schistocity at 198' = 75° to core. Schistocity at 209' = 30° to core.
197	206	108	108	100		
206	212	72	60	83.3		
212	249	444	434	98.1	212-249	Medium to coarse-grained leucocratic quartz monzonite showing considerable fracturing with muscovite, milky quartz and siderite along the fractures. Strongest fractures at 25° to core.

N.T.S. 82M13E

BOULDER PROPERTY

D.D.H. No. 7.

Azimuth: 150°
Declination: -85°
Collar Elevation: 3779'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
0	17.5	214	3	1.4	0 to 17.5	Overburden with pegmatite and quartz monzonite boulders.
17.5	19	18	15	83.3	17.5-19	Coarse garnet diopside idocrase skarn showing some banding. (40° to core). Clay at 19'.
19	20	12	12	100	19-20	Banded green siliceous rock (? quar zite) (35° to core) with massive quartz between about 19.25' to 19.7
20	21	12	6	50	20-30.5	Banded limey diopside garnet skarn Banding at 30° to core. Minor siliceous patches.
21	23	24	18	75		
23	27	48	48	100		
27	30.5	42	34	81		
30.5	32.5	24	24	100	30.5-36	Banded siliceous rock (?quartzite).
32.5	36	42	6	14.3		
36	38	24	17	71	36-39.5	Quartz-biotite schist. Schistocity at 45° to core.
38	39.5	18	16	89		
39.5	43	30	13	43.3	39.5-43	Crush and gouge of fault
43	45	24	4	16	43-46.25	Quartz/biotite schist Schistocity at 45.5' = 40° to core
45	46.25	15	15	100		
46.25	48	21	21	100	46.25-48	Medium-granined leucocratic quartz monzonite. Contacts//compositional banding as follows: 1. Upper Contact 40° to core. 2. Lower Contact 10° to core.
48	51	36	36	100	48-51	Biotite quartz schist.
51	53	24	24	100	51-53	Chloritized and mildly pyritized medium-grained quartz monzonite. Upper contact transgresses compositional banding at 75° to core.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No.7.

Azimuth: 150°
Declination: -85°
Collar Elevation: 3779'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
53	60	84	77	91.6	53-60	Pegmatite with minor graphic granite at 59.5'.
60	65	60	60	100	60-65	Fine-grained biotite ?-granodiorite. Both contacts irregular.
65	67.5	30	30	100	65-67.5	Pegmatite
67.5	69.5	24	24	100	67.5-69.5	Medium-grained biotite quartz monzonite
69.5	72	30	30	100	69.5-72	Coarse-grained leucocratic quartz monzonite with minor biotite.
72	83	132	132	100	72-83	Fine-grained biotite ?-granodiorite.
83	91	96	96	100	83-91	Fine-grained leucocratic quartz monzonite with soft green mineral (? Chlorite) in fractures.
91	130	468	468	100	91-130	Fine-grained biotite ?-granodiorite with following variations: 1. Fracture at 94' filled with ?siderite and at 5° to core. Biotite absent from the intrusive to either side of the fracture. 2. Thin calcite veins at 93.5'& between 123' and 126' 3. Chlorite-filled fractures between 114' and 115'. 4. Four small grains of molybdo-scheelite at 93.5'.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 8.

Azimuth: 167°
Declination: -45°
Collar Elevation: 3939'
Description by: D. L. Cook.

Feet From	Feet To	Inches Int.	Inches Core Recovered	% Core Recovered	Feet Geol. Interval	Description
0	22	264	6	2.2	0-22	Overburden with ? boulders of intrusives.
22	27	60	48	80	22-27	Tremolite, garnet skarn with some idocrase and minor green calcite. Silicified first and last 12".
27	29	24	5	21	27-34	Garnet, quartz, idocrase diopside skarn. Mostly banded at 70° to core.
29	31	24	14	58.3		
31	34	36	27	75		
34	47.5	162	162	100	34-47.5	Tremolite, garnet, green calcite, skarn with some idocrase. Banded diopside, garnet, quartz skarn betwee 39' and 41'. Banding at 60° to core.
47.5	60	150	150	100	47.5-60	Banded grey quartzite. Banding at 51' = 65° to core.
60	66.75	81	81	100	60-66.75	Schistose grey quartzite. Schistocity at 61' = 60° to core.
66.75	92.5	309	309	100	66.75-92.5	Banded and schistose diopside, quartz biotite ?-skarn or ?-hornfels with occasional patches of garnet and idocrase. Banding at 69'=70°, at 81' = 60°. Narrow calcite veins at 76' and 82.5' at about 15° to core.
92.5	103	126	126	100	92.5-103	Quartzite with dark biotite banding. Banding at 101' = 75° to core.
103	128	300	288	96	103-128	Mainly banded diopside biotite ?chlorite quartz ?-skarn or ?-hornfels with the following variations:

1. Silification mainly between 103.5' & 105.5'; 112' and 113'.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 8.

Azimuth: 167°
Declination: -45°
Collar Elevation: 3939'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
						2. Garnet at 103.5', 106.5', 114.5' 117' - 118' and 124'. (idocrase at 124').
						3. Calcite filling veins and fractures from 107' to 112', from 113 to 121' and from 124' to 126'.
						4. Schist between: 106'; 108.5'- 109.5'; 122'- 174'. Compositional banding at 122' =8 to core.
128	161.5	402	390	97	128-161.5	Mainly biotite schist with the following variations:
						1. Calcite in veins from 154.5' to 157.5'.
						2. Limestone from 139' to 140', 134' to 135' and at 161.5'.
						3. Quartz monzonite vein at 141'.
						4. Banded diopside, garnet, quartz skarn from: 137' to 140'; 143' to 145.5'; 149' to 150'
						5. Silicified at 155'. Banding at: 132' = 70° to core; 143' = 75° " "; 152' = 80° " "

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 8.

Azimuth: 167°
Declination: -45°
Collar Elevation: 3939'
Description by: D. L. Cook.

Feet From	Feet To	Inches Int.	Inches Core Recovered	% Core Recovered	Feet Geol. Interval	Description
161.5	165	42	42	100	161.5'-165'	Medium-grained quartz monzonite with minor biotite and some calcite veining.
165	170.5	66	66	100	165-170.5	Mainly quartz/biotite schist with calcite veining from 165' to 166'. Yellow calcite and silification at 168.5'. Compositional banding at 169.5' = 50° to core.
170.5	171.5	12	12	100	170.5-171.5	Medium to coarse-grained quartz monzonite. Contacts very irregular distinctly showing digestion of the schist.
171.5	174.5	36	36	100	171.5-174.5	Biotite/quartz schist with yellow calcite at 171'.
174.5	181.5	84	67	79.7	174.5-181.5	Medium-grained leucocratic quartz monzonite. Fault crush and gouge between 178.5' and 181'.
181.5	191.25	117	117	100	181.5-191.25	Mainly diopside, calcite, quartz, garnet skarn showing indistinct and irregular banding. Almost completely silicified between 186' and 188.
191.25	210	105	105	100	191.25-210	Quartz biotite schist. Compositiona banding as follows: 193' = 75° to core 203' = 85° to core 205' to 207' - highly contorted. 210' = 50° to core.
210	218	96	96	100	210-218	Banded quartzite. Compositional banding at 215' = 35° to core.
218	227	108	108	100	218-227	Quartz biotite schist. Compositiona banding at 225' = 40° to core.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 8

Azimuth: 167°
Declination: -45°
Collar Elevation: 3939'
Description by: D. L. Cook.

Feet From	Feet To	Inches Int.	Inches Core Recovered	% Core Recovered	Feet Geol. Interval	Description
227	230	36	36	100	227-230	Kaolinization of quartz muscovite schist. Pyrite patches.
230	232.5	33	33	100	230-232.5	Fractured quartz. Pyrite patches. Kaolinization in fractures.
232.5	240	90	90	100	232.5-240	Quartzite. Biotite-rich and schistose from 232.5' to 236'. Pyrite veining at 239'. Compositiona banding at 235' = 60° to core.
240	263.5	282	282	100	240-263.5	Biotite quartz schist with the following variations: 1. Muscovite and kaolinization from 240' to 244'. 2. Pyrite as patches in the banding an as a vein (30° to core) at 244.5'. 3. Pyrite in banding at 257'. Compositional banding at 245' =75° and at 255' = 45° to core.
263.5	270	78	78	100	263.5-270'	Quartzite showing faint compositiona banding. Banding at 265' = 75°tocor
270	271	12	12	100	270'-271'	Coarse-grained leucocratic quartz monzonite. Both contacts//banding as follows: 1. Upper contact 60° to core. 2. Lower contact 55° to core.
271	275	48	48	100	271-275	Schistose biotite quartzite. Compositional banding at 273.5' = 45° to core.
275	279	48	48	100	275-279	Medium-grained leucocratic quartz monzonite. Contacts not evident.

N.T.S. 82M/13E

BOULDER PROPERTY

D.D.H. No. 8.

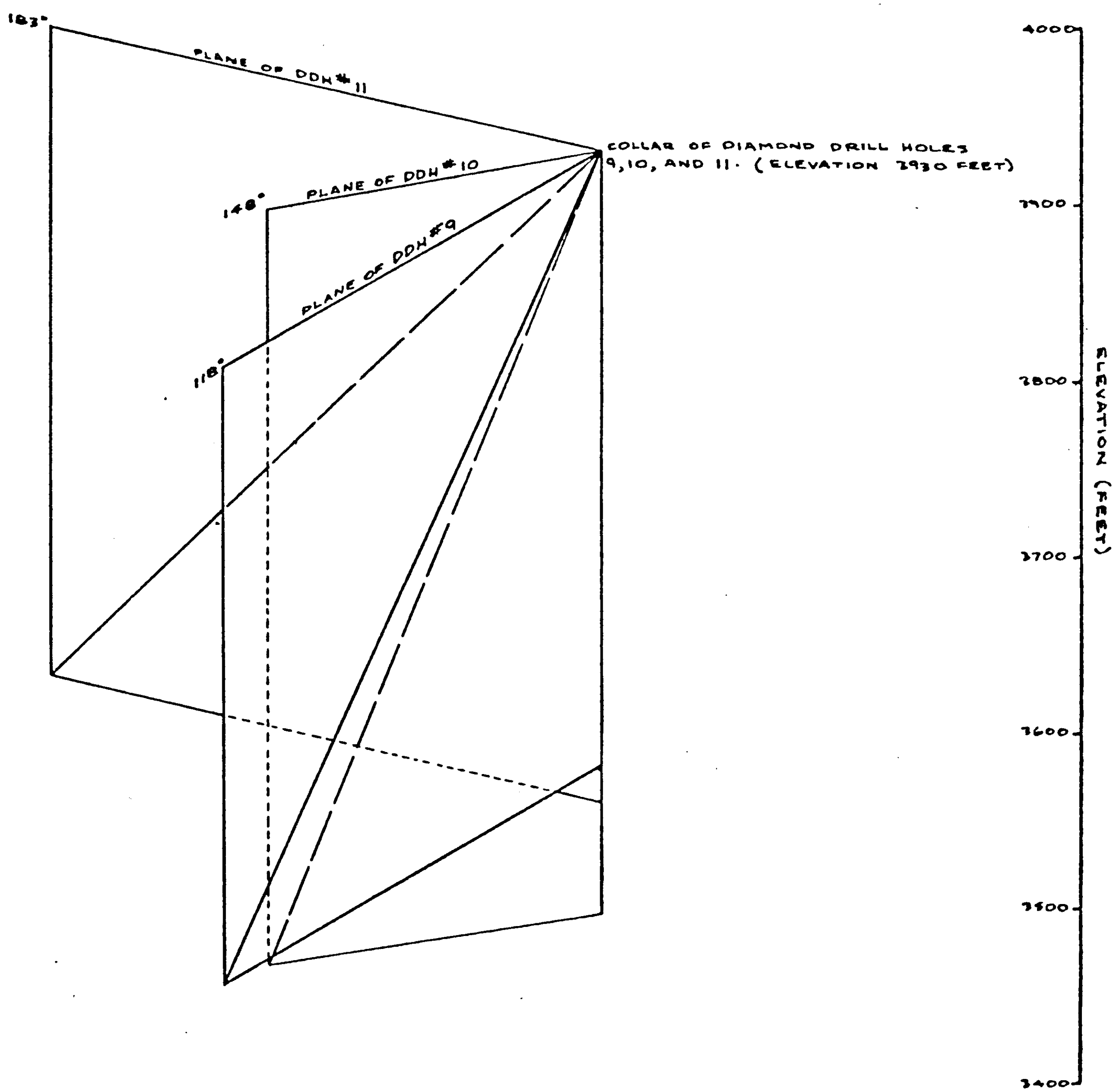
Azimuth: 167°
Declination: -45°
Collar Elevation: 3939'
Description by: D. L. Cook.

Feet		Inches			Feet	
From	To	Int.	Core Recovered	% Core Recovered	Geol. Interval	Description
279	281	24	24	100	279-281	Schistose quartzite. Compositional banding at 279' ≈ 50° to core. Some evidence of invasion by the quartz monzonite at 281'.

DIAGRAM 4

LAYOUT OF DIAMOND DRILL HOLES - 19[illegible]

BOULDER PROPERTY



DRILL HOLE INFORMATION:

HOLE NUMBER	9	10	11
AZIMUTH	118°	148°	183°
PLUNGE	53°	65°	46°
HOLE DEPTH (FT.)	430	476	530

DIAGRAM 6

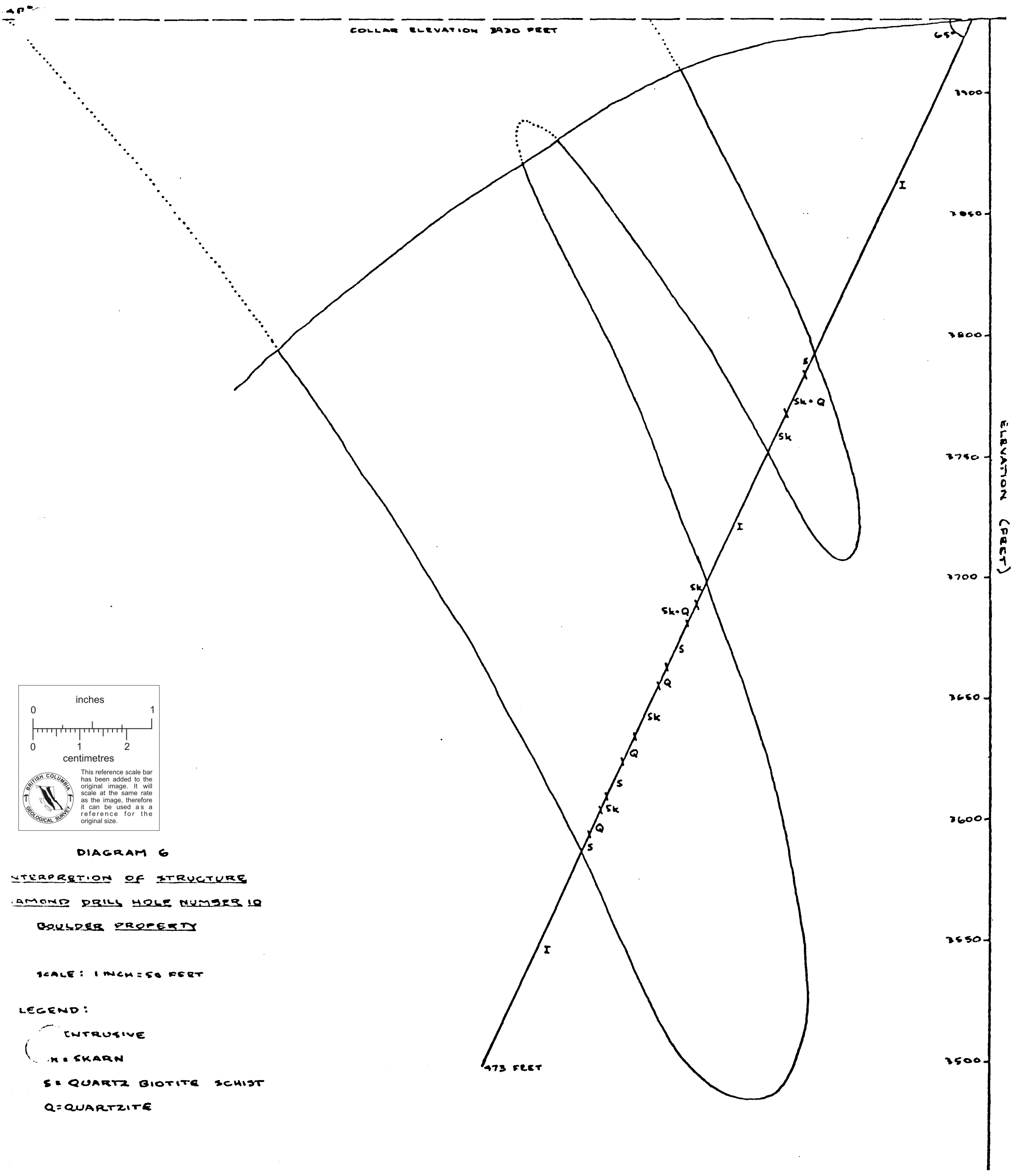


DIAGRAM 6

NTERPRETION OF STRUCTURE

AMOND DRILL HOLE NUMBER 10

BOULDER PROPERTY

SCALE: 1 INCH = 50 FEET

LEGEND:

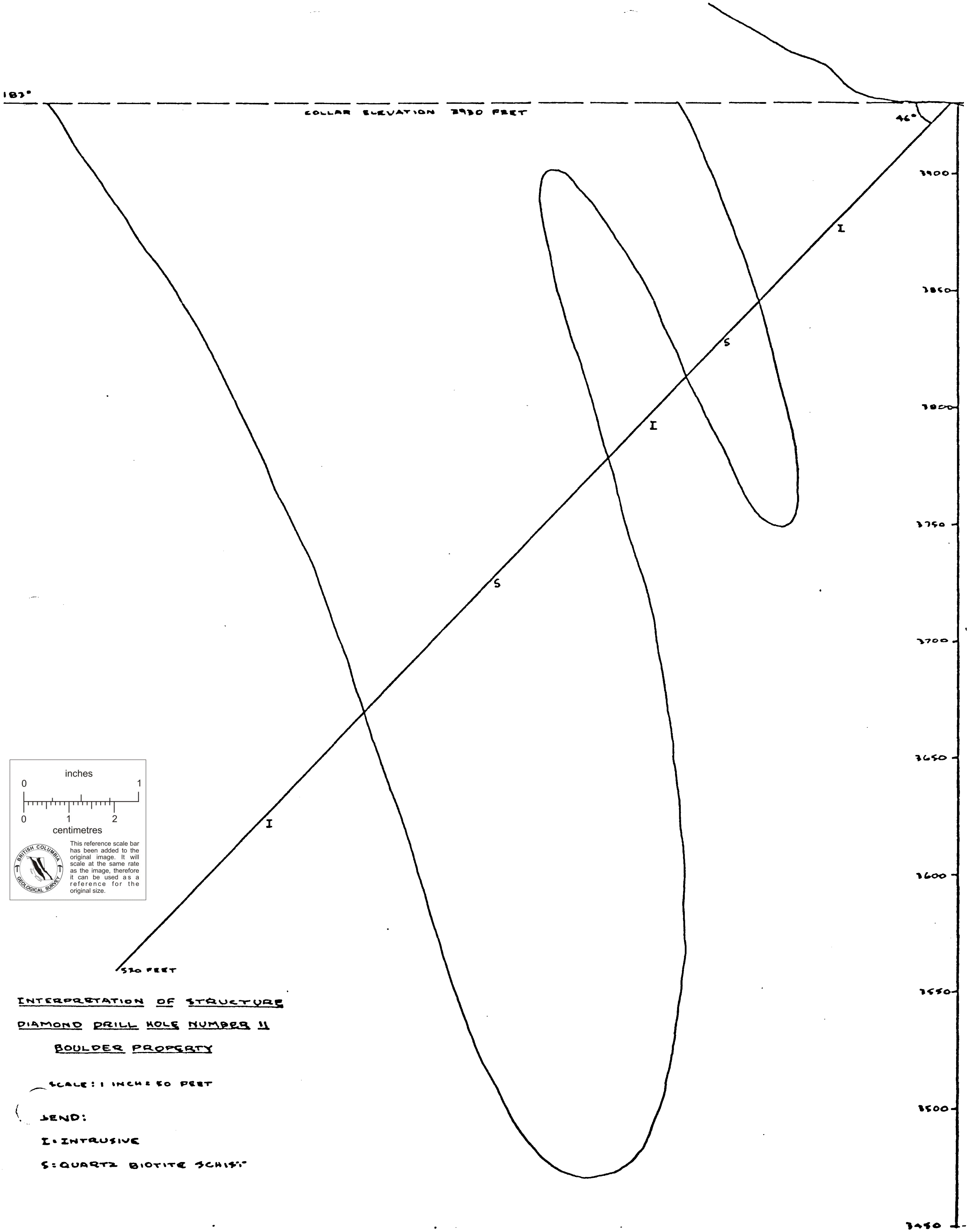
INTRUSIVE

K = SKARN

S = QUARTZ BIOTITE SCHIST

Q = QUARTZITE

DIAGRAM 1



INTERPRETATION OF STRUCTURE

DIAMOND DRILL HOLE NUMBER 11

BOULDER PROPERTY

SCALE: 1 INCH = 50 FEET

LEGEND:

I: INTRUSIVE

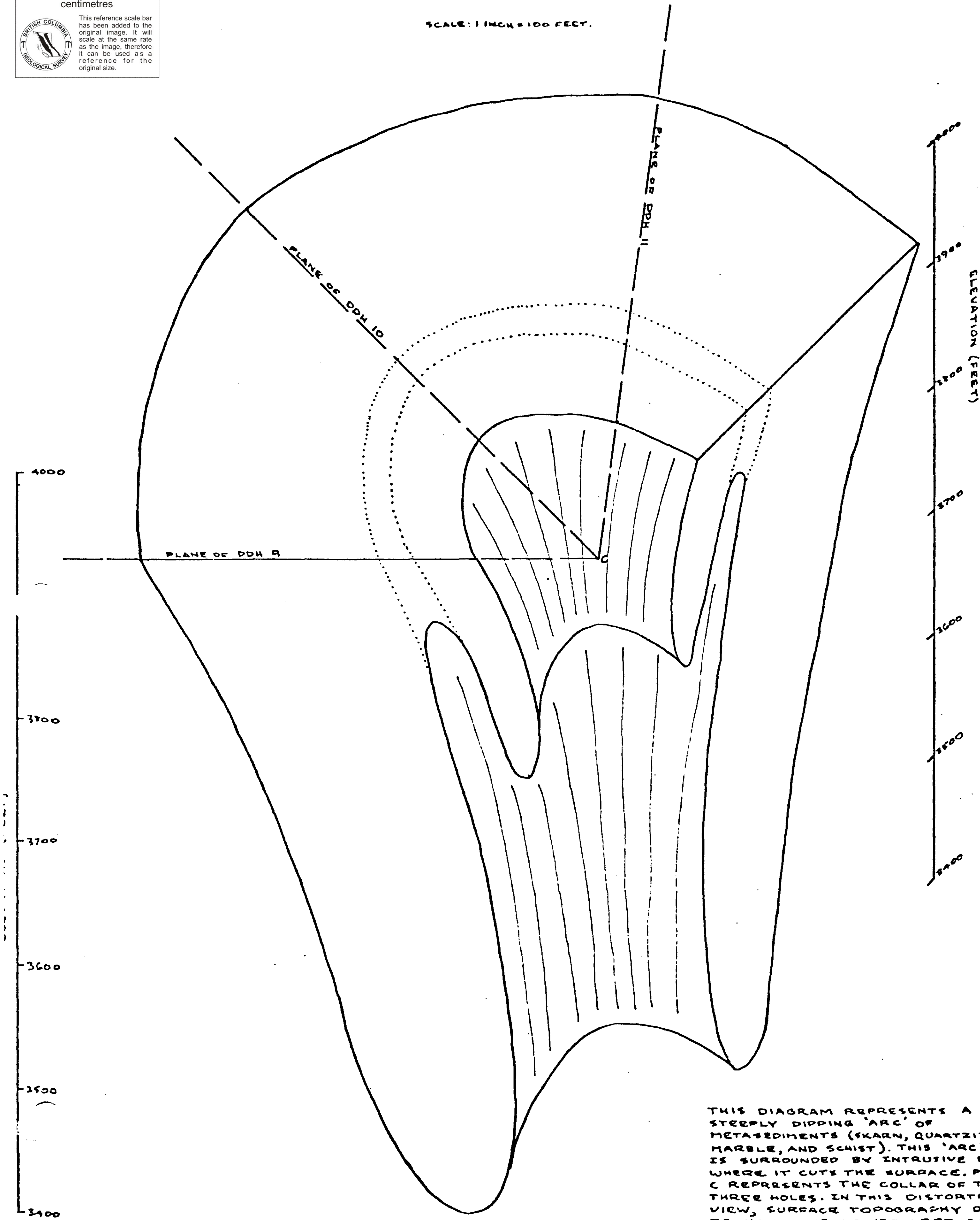
S: QUARTZ BIOTITE SCHIST

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

DIAGRAM 8

FOLD HYPOTHESIS FOR THE EXPLANATION OF INFORMATION FOUND IN DIAMON DRILL HOLES 9, 10, AND 11. BOULDI PROPERTY - 1973.

SCALE: 1 INCH = 100 FEET.

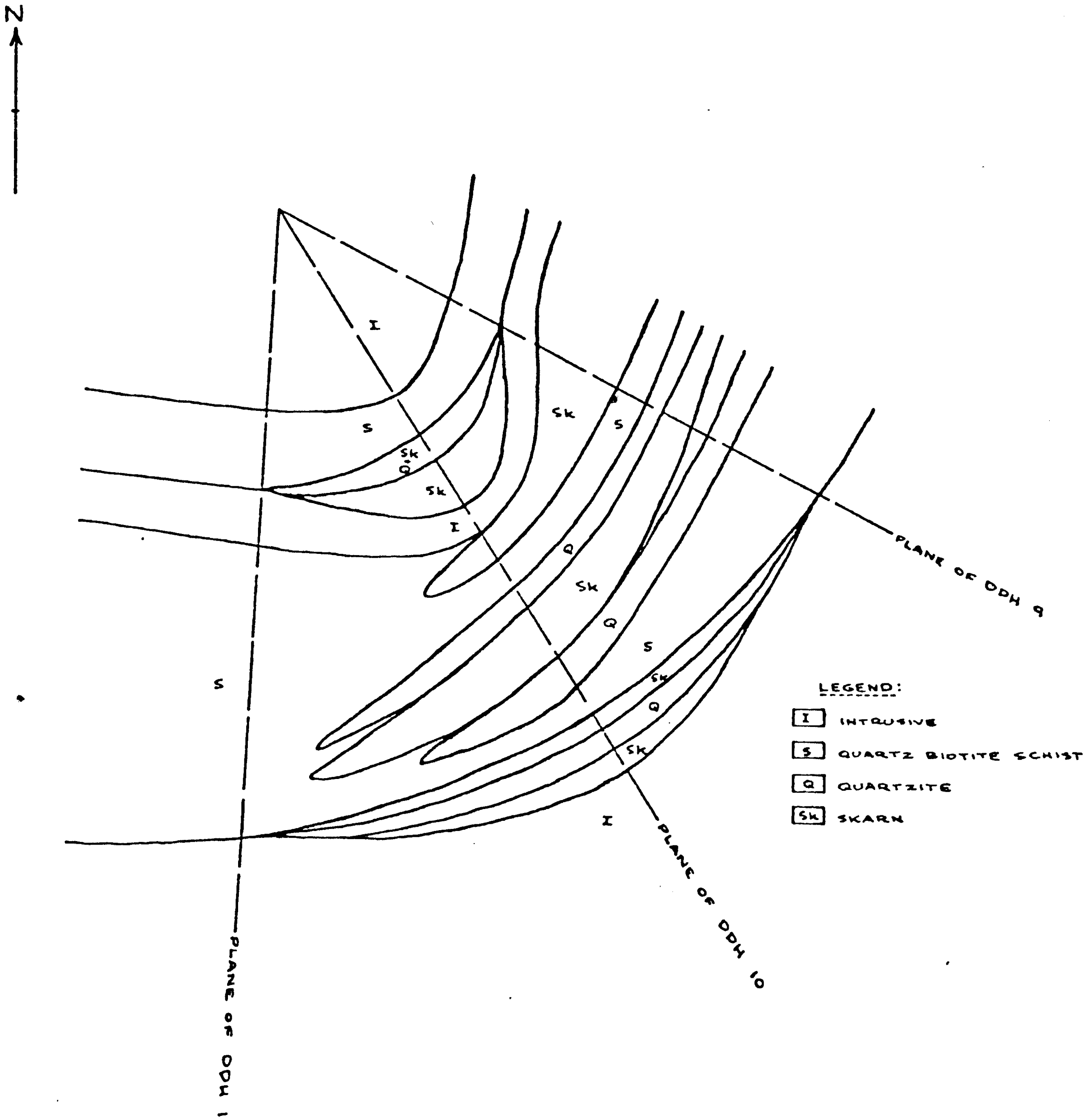


THIS DIAGRAM REPRESENTS A STEEPLY DIPPING 'ARC' OF METASEDIMENTS (SKARN, QUARTZITE, MARBLE, AND SCHIST). THIS 'ARC' IS SURROUNDED BY INTRUSIVE EXCEPT WHERE IT CUTS THE SURFACE. POINT C REPRESENTS THE COLLAR OF THE THREE HOLES. IN THIS DISTORTED VIEW, SURFACE TOPOGRAPHY DIPS TOWARDS THE LOWER LEFT CORNE OF THE PAPER. GRADIANT OF THE SLOPE IS APPROXIMATELY 20°-25°

DIAGRAM 9

A POSSIBL ACIES CHANGE EXPLAINING THE D ERENCES IN LITHOLOGY BETWEEN DRILL HOLES 9, 10, AND 11.

SCALE: 1 INCH = 100 FEET.



THIS IS A HORIZONTAL SECTION CONSTRUCTED AT THE 2920 FOOT ELEVATION i.e. COLLAR ELEVATION. THE STRUCTURE IS BASED ON THE 'FOLD' HYPOTHESIS (DIAGRAM 8). NOTE THAT ONLY HORIZONTAL FACIES CHANGES ARE CONSIDERED. INSUFFICIENT DATA IS AVAILABLE TO CONSIDER VERTICAL CHANGES IN FACIES ALTHOUGH THESE MOST CERTAINLY EXIST.

DIAGRAM 10

ALTERNATE HY HESIS ('THREE BAND HYPOTHESIS) FO XPLANATION OF INFORMATION FOUND IN DRILL HOLES 9, 10, AND 11.

SCALE: 1 INCH = 100 FEET.

THIS HORIZONTAL SECTION IS CONSTRUCTED AT THE 3930 FOOT ELEVATION ie AT COLLAR ELEVATION.

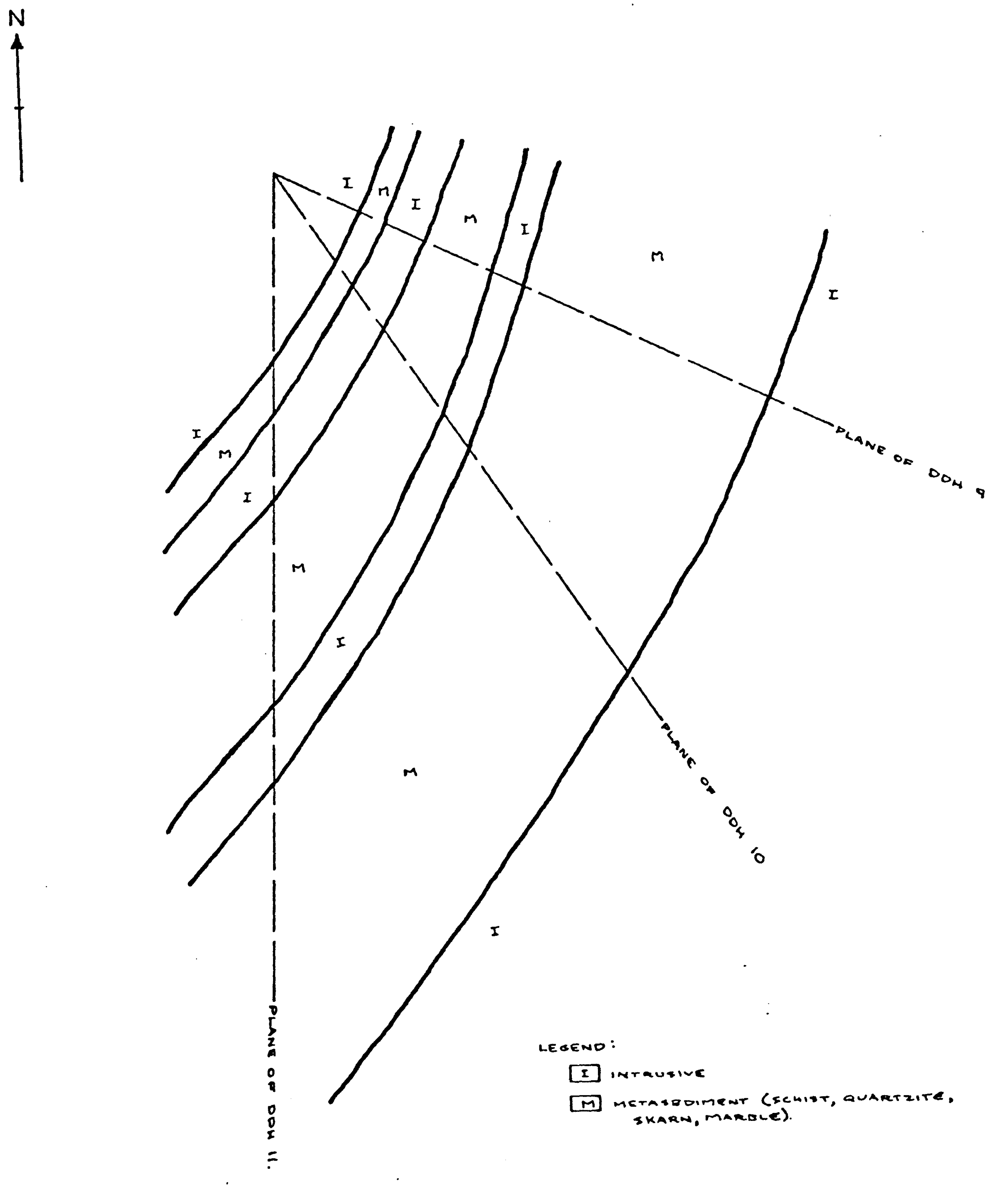
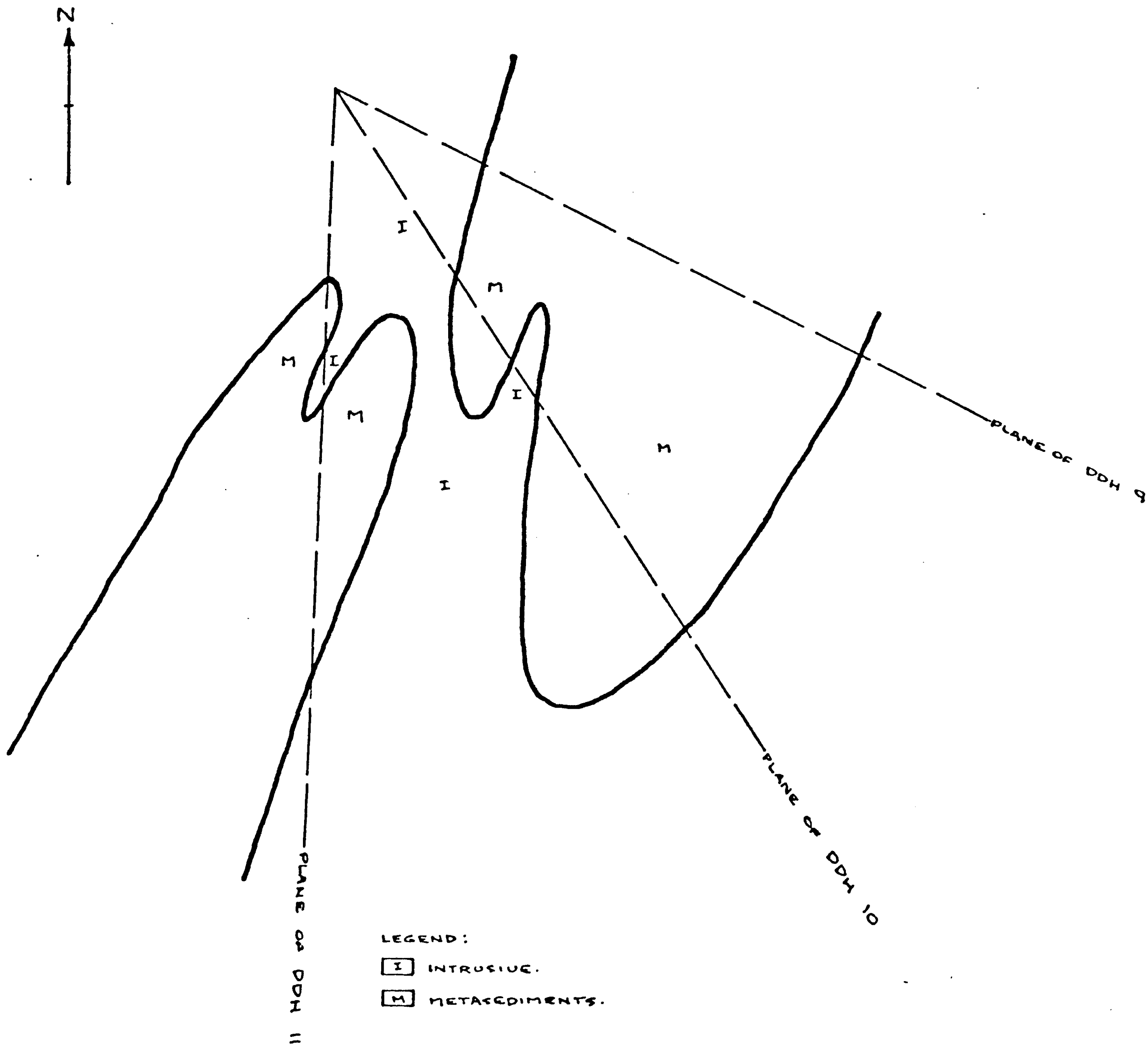


DIAGRAM 11

ALTERNATE ("DISCONTINUOUS BAND") HYPOTHESIS OR EXPLANATION OF INFORMATION FOUND IN DRILL HOLES 9, 10, AND 11.

SCALE: 1 INCH = 100 FEET

THIS HORIZONTAL SECTION IS CONSTRUCTED AT THE 2930 FOOT ELEVATION i.e. AT COLLAR ELEVATION.



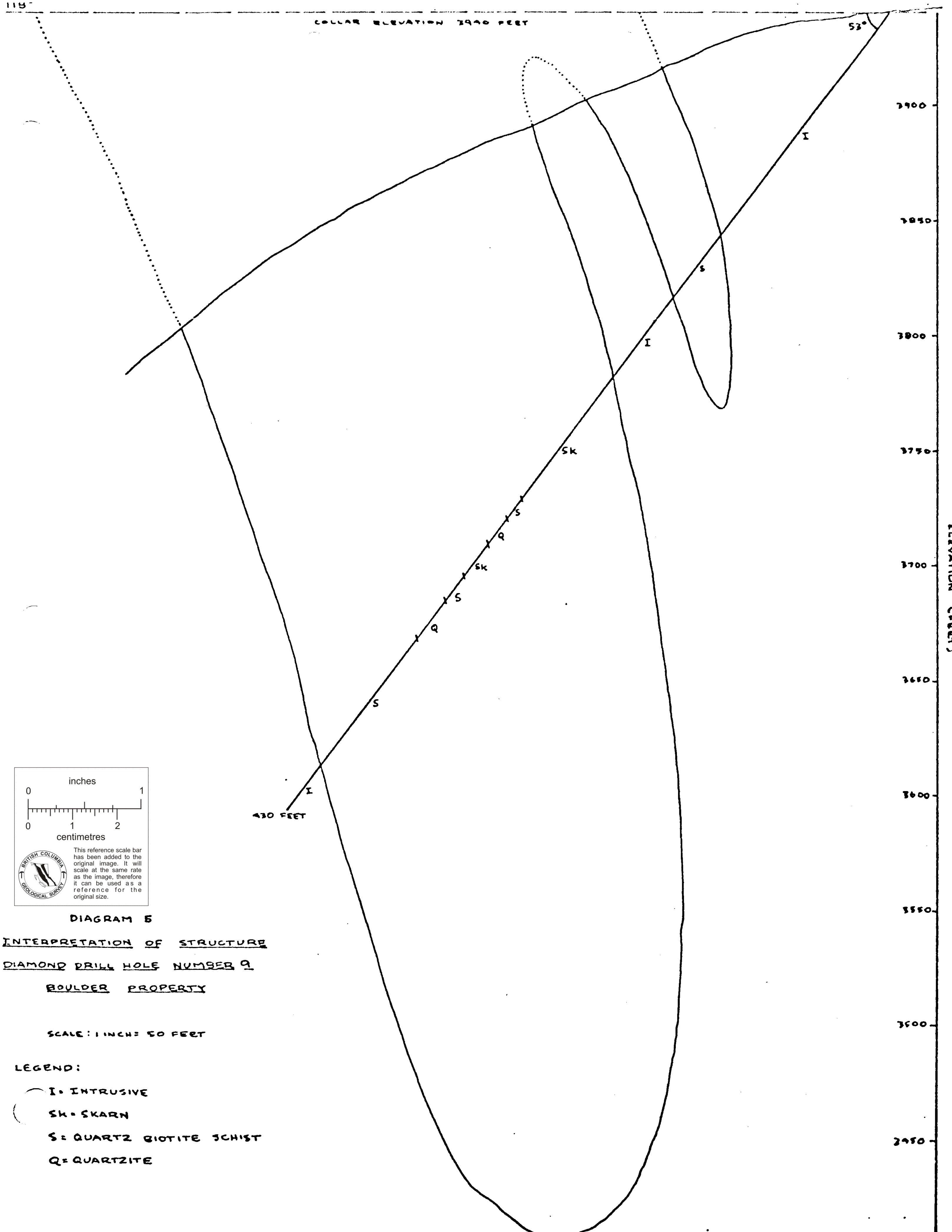


DIAGRAM 5

INTERPRETATION OF STRUCTURE

DIAMOND DRILL HOLE NUMBER 9

BOULDER PROPERTY

SCALE: 1 INCH = 50 FEET

LEGEND:

I = INTRUSIVE

SK = SKARN

S = QUARTZ BIOTITE SCHIST

Q = QUARTZITE

DIAGRAM 15.

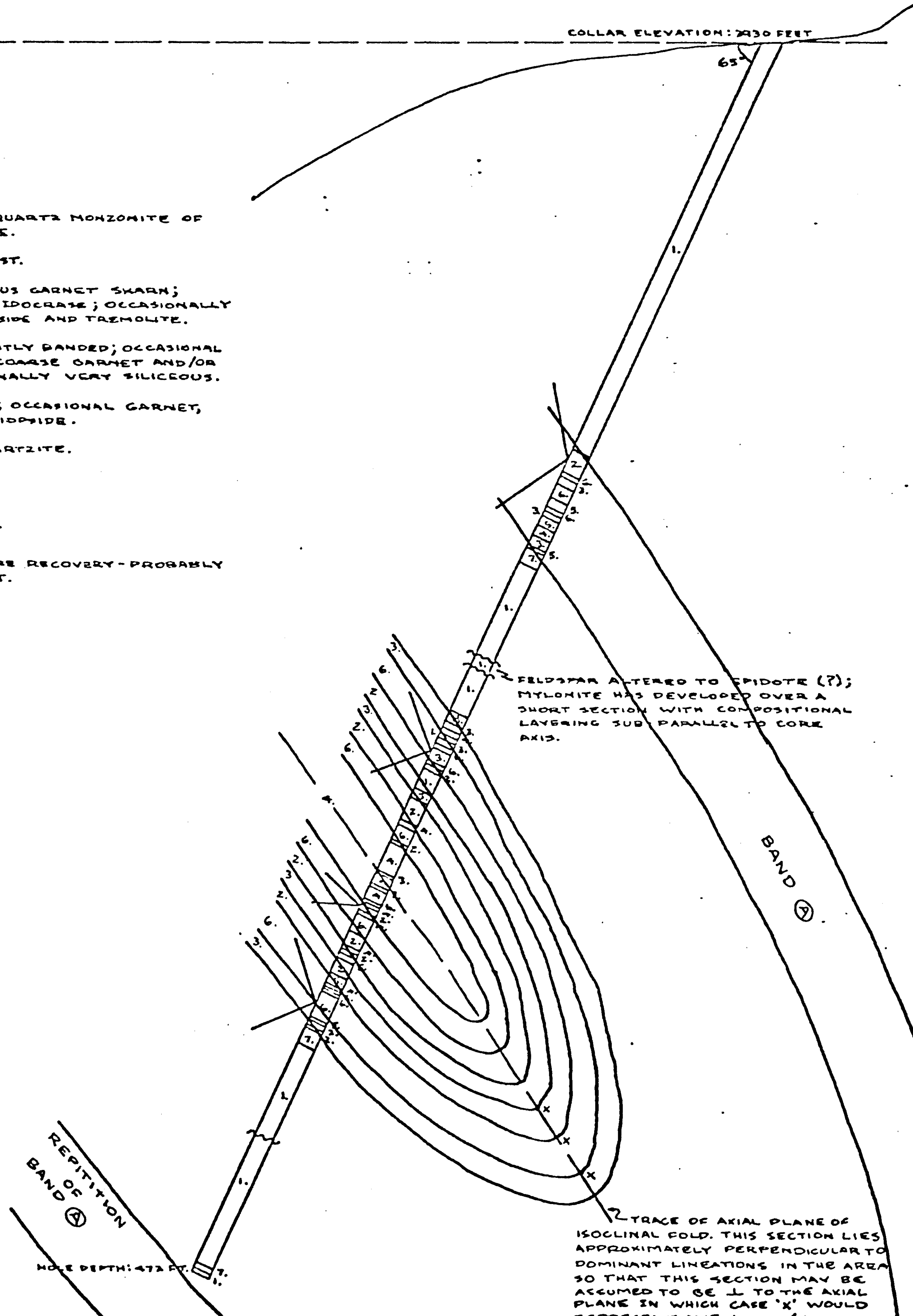
REPITITION OF LITHOLOGY AS APPLIED TO AN ISOCLINAL FOLD HYPOTHESIS IN DIAMOND DRILL HOLE NUMBER 10.

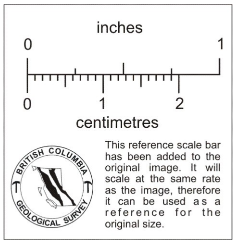
SCALE: 1 INCH = 50 FEET.

LEGEND:

- 1. INTRUSIVE; MAINLY QUARTZ MONZONITE OF VARIABLE GRAIN SIZE.
- 2. QUARTZ BIOTITE SCHIST.
- 3. COARSE, VERY SILICEOUS GARNET SKARN; OCCASIONAL COARSE IDOCRASE; OCCASIONALLY BANDED; MINOR DIOPSIDE AND TREMOLITE.
- 4. WHITE MARBLE; SLIGHTLY BANDED; OCCASIONAL SHORT SECTIONS OF COARSE GARNET AND/OR IDOCRASE; OCCASIONALLY VERY SILICEOUS.
- 5. TREMOLITE SKARN; OCCASIONAL GARNET, IDOCRASE, AND/OR DIOPSIDE.
- 6. BANDED, LIMEY QUARTZITE.
- 7. PEGMATITE.
- 8. QUARTZ.

GOUGE OR POOR CORE RECOVERY - PROBABLY INDICATES A FAULT.



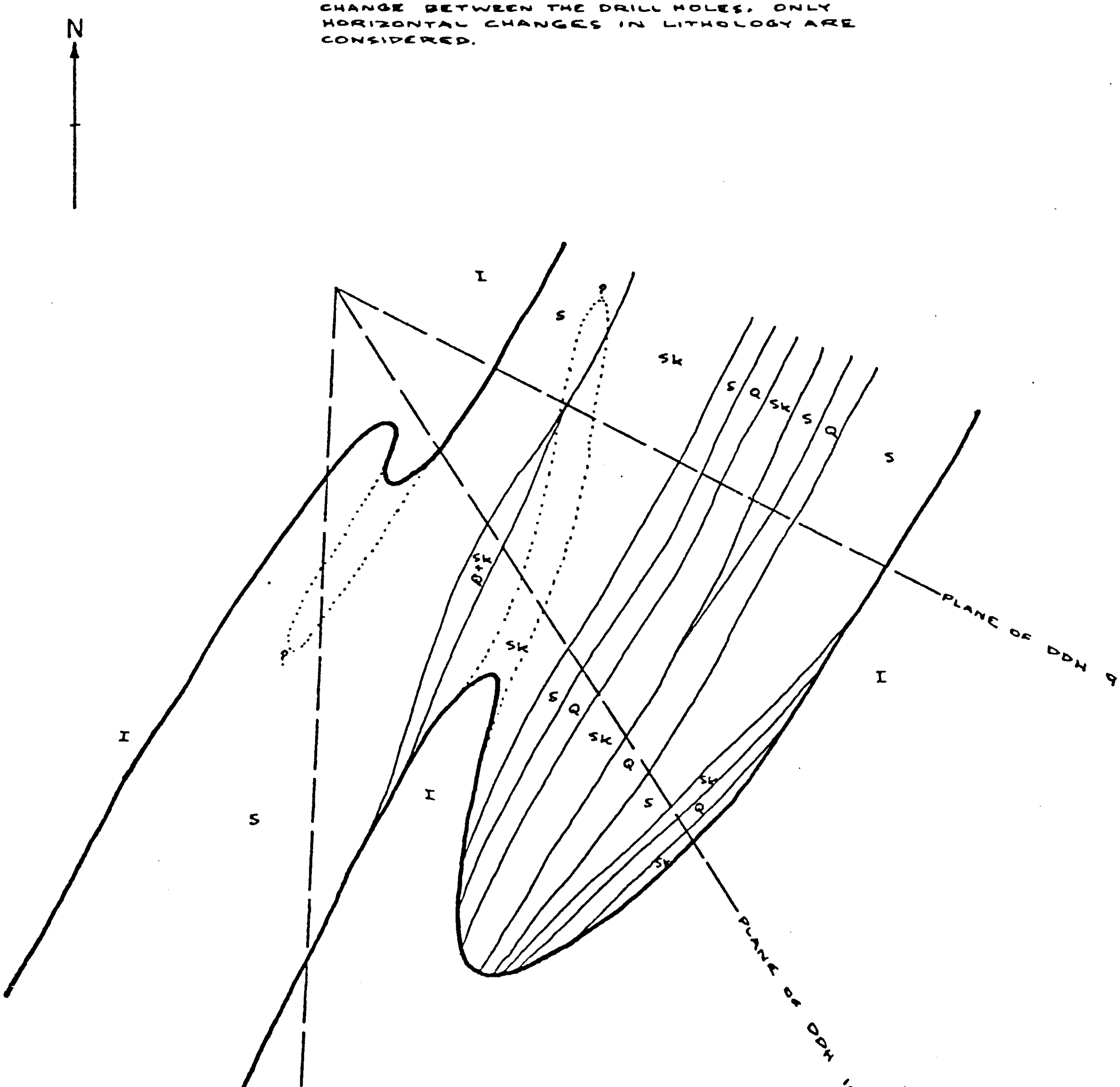


ALTERNATE ('ONE BAND') HYPOTHESIS FOR EXPLANATION OF INFORMATION FOUND IN DRILL HOLES 9, 10, AND 11.

SCALE: 1 INCH = 100 FEET.

THIS HORIZONTAL SECTION IS CONSTRUCTED AT THE 3930 FOOT ELEVATION i.e. AT COLLAR ELEVATION.

THIS DIAGRAM ALSO SHOWS A POSSIBLE FACIES CHANGE BETWEEN THE DRILL HOLES. ONLY HORIZONTAL CHANGES IN LITHOLOGY ARE CONSIDERED.



LEGEND:

I INTRUSIVE.

S QUARTZ BIOTITE SCHIST.

Q QUARTZITE.

SK SKARN

.... REPRESENTS APPROXIMATE POSITION OF THE TWO 'GROOVES' IN THE BAND WHERE THEY CUT THE DRILL HOLES. THE POSITION OF THE 'GROOVE' IS PROJECTED UP-DIP TO THE 3930 FOOT LEVEL ie THIS WOULD REPRESENT THE POSITION OF THE 'GROOVE' AT THE 3930 FOOT LEVEL IF IT DID NOT FADE OUT WITH INCREASE IN ELEVATION (IT IS ASSUMED HERE THAT THE GROOVE FADES OUT WITH INCREASE IN ELEVATION).

A P P E N D I X I

The Red Rose Mine*

British Columbia's First Tungsten Producer

By H. M. WRIGHT†

Acknowledgment

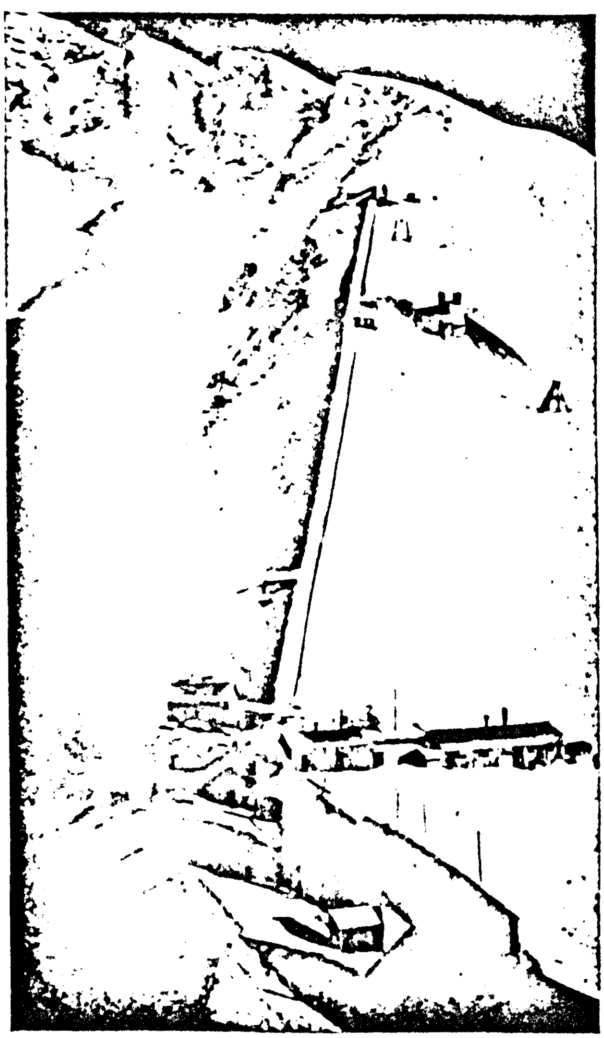
THE author acknowledges with thanks the kind permission of Mr. R. W. Diamond, Vice-President and General Manager of Consolidated, to publish this article on the Red Rose Mine. Mr. Stanley Gray, Superintendent of Outside Mills for Consolidated supplied much of the data incorporated in this article and to him, as well as to the Editorial Committee of the Company, appreciation is herewith expressed.

Acknowledgement is also made to the British Columbia Department of Mines and Dr. J. S. Stevenson for information regarding geology and history of the Red Rose as portrayed in Bulletin No. 10 by Dr. Stevenson.

Foreword

Among the many adjustments necessitated by the war was the desire to satisfy Canada's tungsten requirements from production within the Dominion. In order to assist with this program Consolidated optioned the Red Rose Mine in 1939. This was followed by an active development period in conjunction with ore testing. Test work was conducted in the laboratory of the Sullivan Concentrator at Chapman Camp, B. C., under the supervision of Mr. Gray.

This work was augmented by several jig tests by Denver Equipment Company in Denver and the results obtained by the two laboratories evidenced that satisfactory metallurgical results could be obtained. The test work indicated that over 80% recovery could be expected by employing gravity concentration with a grade in the neighborhood of 65% WO_3. It was determined that the greater proportion of the recovery could be made with a jig at a coarse grind and tables to raise the overall recovery. On this basis a 25 ton gravity mill was designed and installed by Mr. Gray in 1941, and operations commenced in February, 1942. In order to satisfy increasing demands shortly thereafter, the original mill was increased in capacity by equipment changes and at the same time flotation was added to increase the overall recovery. At this stage the Red Rose Mill handled 75 tons per day and produced gravity concentrates suitable for direct charging to the steel furnaces and a flotation concentrate which was shipped for chemical treatment. Operations continued on this basis until November, 1943, at which time the property was closed down owing to lack of market and critical wartime labor shortage.



The Red Rose tungsten mine.
—Photo by Angus W. Davis

Location

The Red Rose Mine is located on the west side of Rocher de Boule Mountain, seven miles south of Hazelton, British Columbia. The mine is reached by 30 miles of road from Ha-

*Reprinted with the permission of Deco Trefoil.
†Northwestern Engineer, Denver Equipment Co.

The mine and mill are connected by nearly a mile of gravity aerial tram.

zelton by way of Skeena Crossing which is 15 miles down the Skeena River from Hazelton. Hazelton is on the northern line of the Canadian National Railway 175 miles northeasterly from Prince Rupert. Hazelton is 820 miles from Vancouver, B. C., by automobile.

The Red Rose Mill is located on Red Rose Creek at timberline, at an elevation of 4000 feet. It is one mile distant from the mine camp which is at an elevation of 5600 feet. The mine workings are at elevations up to 6360 feet and the mine and mill are connected with a 5000 foot gravity aerial tram. The mine camp and mine workings are connected with a surface tram which is closed in on account of the severe weather conditions experienced at this elevation. There is also a tractor road connecting the mill camp and mine camp over which heavy supplies are hauled. This road courses up the steep slope over talus slides and rock bluffs.

History

The Red Rose property was first staked in 1913 on a gold-silver showing. Preliminary work was carried out in 1914 and again in 1916 along the shear where the discovery was made. In all, three adits were driven at 5150 feet, 5450 feet and 5690 feet elevations. The results of this work were inconclusive.

In 1923 tungsten was reported from a quartz vein that outcropped 700 feet higher in elevation than the upper workings on the gold-silver zone. Other than assessment work, however, little was accomplished until Consolidated drilled the tungsten showing in 1940, and this was the commencement of the present history which was confined to the tungsten zone.

Geology

The rocks in the vicinity of the Red Rose consist of sediments in various stages of alteration to hornfels. The hornfels are intruded by sill-like masses of diorite and diorite porphyry. East of the tungsten workings, some 750 feet, is the contact with a large mass of granodiorite. The formations in this area have a general strike of north-east and dip north-west. The tungsten occurs in a quartz fissure vein in diorite that strikes north 45 degrees west and dips from 55 to 60 degrees south-west. The vein is lenticular in character and pinches and swells from several inches up to 11 feet in width. Vein matter consists of quartz with feldspar, apatite, biotite, tourmaline, scheelite, ferberite, hematite and very small amounts of chalcopyrite, molybdenite and arsenopyrite. The copper carbonates, malachite and azurite, are also present.

Mining

The upper mine portal is at an elevation of 6130 feet and this connects with the mine camp by surface tram. The No. 300 cross-cut from the upper mine portal intersects the vein at 230 feet from the portal and at about 150 feet of depth. A raise connects the No. 300 level with the No. 200 level which is a drift on the vein. The 600 cross-cut from the surface is 200 feet vertically below the 300 level and this intersects the vein at 300 feet. At the elevation of the mine camp and 260 feet vertically below the 600 level another cross-cut from the surface has been driven towards the vein.

The upper three levels are connected with raises and the mine has been developed with drifting and stoping on the vein. The No. 600 is the main level and the upper terminal of the aerial tram is located at its portal.

Water and Power

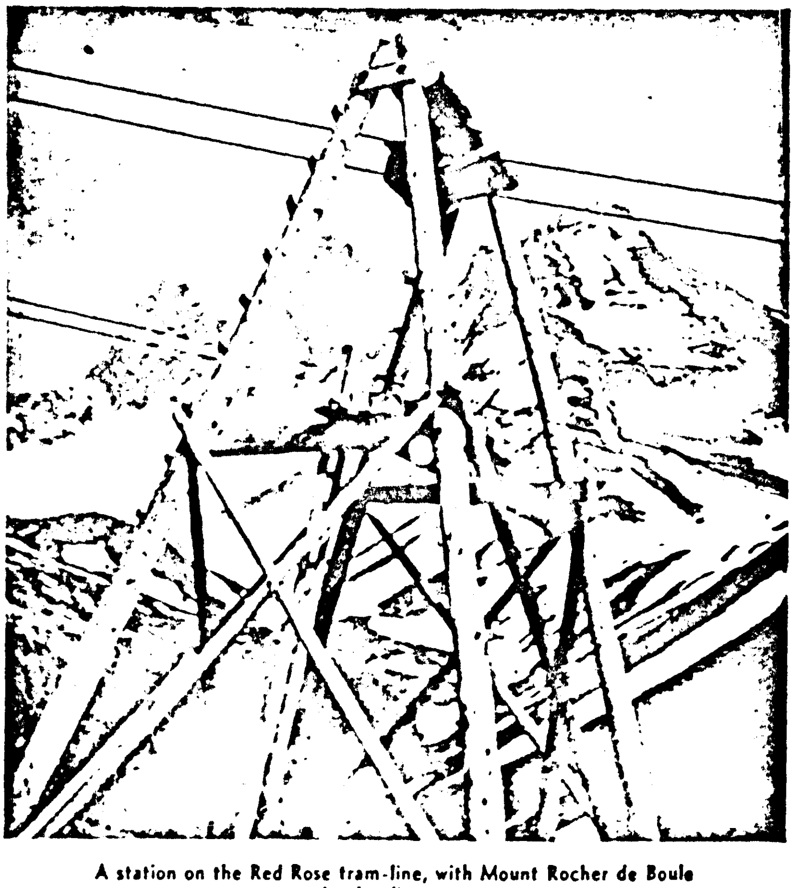
Water for the mill camp is obtained from Red Rose Creek. This stream flows westerly into Juniper Creek, which in turn flows into the Skeena River at Skeena Crossing.

The main power plant is situated at the mill camp. Power is developed at 440 volts by a Vivian diesel, driving an electric generator. There is a separate power unit for the mine, situated at the mine camp, as well as diesel driven compressor unit.

Milling

Crushing

Ore is transported from the mine to the mill by a 5000 foot gravity tram which discharges to the coarse ore bin. Coarse ore is fed over a one inch grizzly to an 8" x 12" jaw crusher



A station on the Red Rose tram-line, with Mount Rocher de Boule in the distance.

set at one inch. The crusher discharges directly to the fine ore bin along with the grizzly undersize.

Grinding

The grinding circuit consists of a 8' x 6' Denver Rod Mill with a V to flat drive through clutch from a 25 hp. motor. It is in closed circuit with a 16" x 24" Denver Duplex Jig with 1½ hp. motor drive and a 2' x 4' Tyrock Screen with 2 hp. motor drive. The circuit is fed with a 12" x 6' Denver Adjustable Stroke Belt Ore Feeder which controls the feed rate to the mill. The Tyrock Screen is decked with a 10 mesh cloth. The rod mill is equipped with a spiral screen with ¼" cloth. The plus ¼" material along with the plus 10 mesh from the Tyrock returns to the rod mill for regrinding. The minus ¼" product from the rod mill is handled with a 2" Wilfley pump to the jig.

The rod mill discharges at 65% solids and the grind is controlled to produce a minimum of minus 200 mesh material. Liberation of the tungsten mineral is fairly complete at 20 mesh.

Rod Mill Feed		Rod Mill Discharge	
Mesh	Wt. %	Mesh	Wt. %
+ 20	83.0	+ 20	15.0
+200	7.8	+200	60.0
—200	9.2	—200	25.0

Jigging

The 16"x24" Duplex Denver Jig in the closed grinding circuit is responsible for the major proportion of the gravity recovery of tungsten in the Red Rose Mill. It operates as a rougher unit and the feed to the jig pumped from the rod mill is at 50% solids. The stroke of the jig is varied between ⅛" to ½", depending on

Rocher de Boule Mountain, in the Omineca mining division.

grade. A water pressure of 19 pounds is maintained with a constant head tank. The jig compartments are equipped with 2 mm. opening wedge wire screens with each screen carrying about 1500 grammes of jig shot. The jig produces a concentrate running about 50% WO_3.

The Duplex rougher jig discharges at intervals as the hutches are filled to a 12"x18" Denver Duplex Jig which operates as a cleaner unit. The feed to the cleaner jig is at 70% solids and the same water pressure of 19 pounds is used. The stroke of the cleaner jig is usually $\frac{1}{16}$" longer than the stroke on the rougher machine. The product of this jig grades from 72 to 73% WO_3 and amounts to 65% of the gravity concentrate recovered. This amounts to 0.9 up to 1 ton per day. The cleaner jig tailing returns by gravity to join the rod mill discharge where it is returned to the rougher jig.

The final jig concentrate is, on an average, made up of 77% scheelite, 8% ferberite, 4% hematite and 11% silicates and oxides. The specific gravity is 5.73.

Mesh Analysis—Final Jig Concentrate

Mesh		% Wt.	Cumulative %
	+ 28	8.1	8.1
—28	+ 35	15.9	24.0
—35	+ 48	19.4	43.4
—48	+ 65	25.4	68.8
—65	+100	25.3	94.1
	—100	5.9	100.0
		100.0	

Tables

The minus 10 mesh product from the Tyrock Screen flows by gravity to the table section. This circuit consists of 2 Wilfley and 3 Deister Plate Tables. The middling products from the table is returned with a 1" Denver Vertical Sand Pump to the discharge of the rod mill while the tailing passes to the flotation circuit.

The concentrates from the table amount to 35% of the total gravity concentrate at 72 to 73% WO_3 grade. Tonnage is in the neighborhood of 0.4 to 0.5 tons per day. These concentrates join the cleaner jig concentrate for subsequent retreatment.

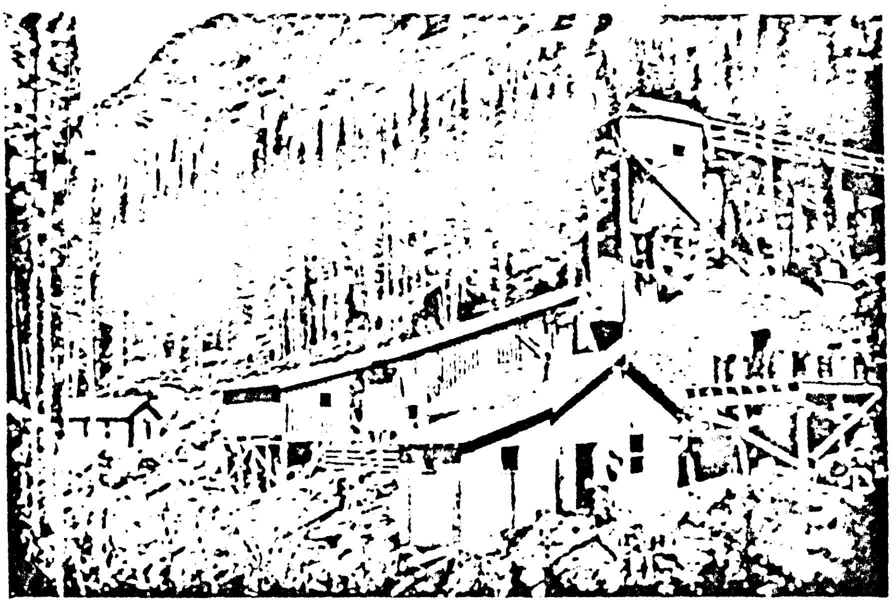
Screen Analysis Table Concentrate

Mesh		Wt. %	% Cumulative
	+ 65	7.3	7.3
— 65	+100	21.2	28.5
—100	+150	28.7	57.2
—150	+200	18.7	75.9
	—200	24.1	100.0
		100.0	

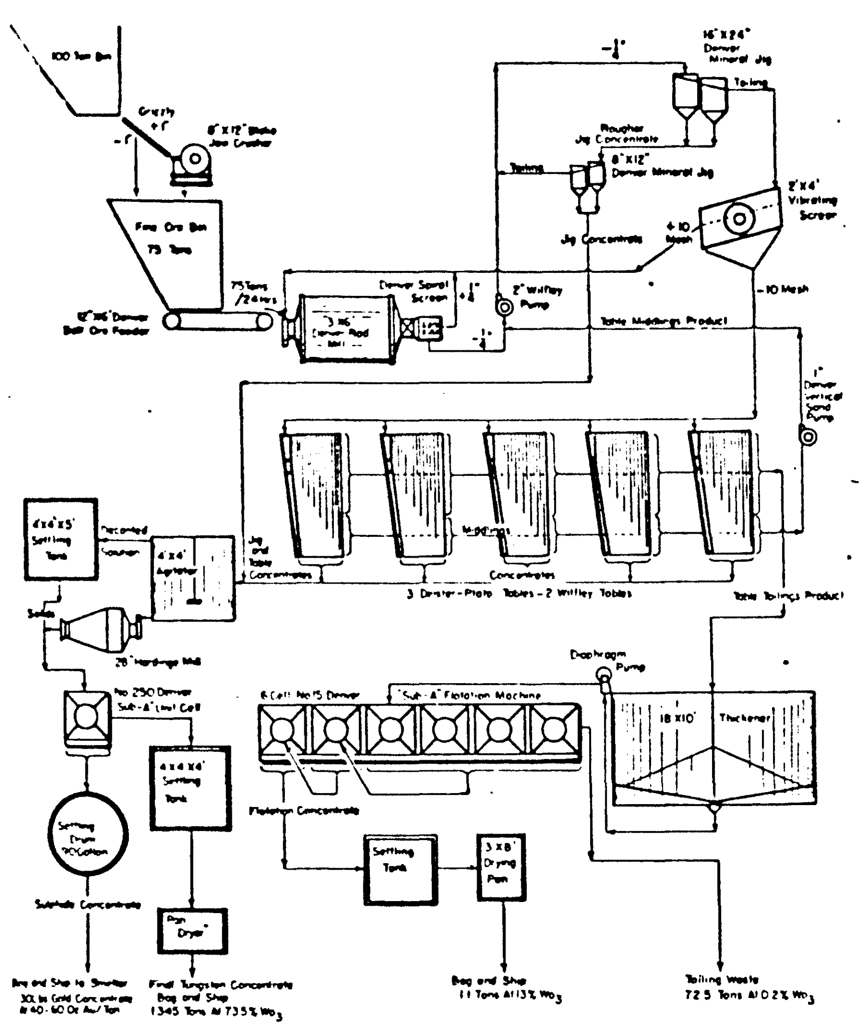
Flotation Circuit

Table tailing is thickened to 35% solids in 18' by 8' Dorr Thickener and pumped with a 2" Diaphragm Pump to a 6 cell No. 15 Denver 'Sub-A" Flotation Machine for production of a flotation tungsten concentrate. The final flotation concentrate carries 14% WO_3 and amounts to 1.1 tons per day.

Flotation feed enters No. 3 cell. The rougher concentrates from cells 3 to 6



The Red Rose tungsten mill.



are cleaned in No. 2 cell and recleaned to final grade in No. 1 cell. The tailing from flotation amounting to 72.5 tons per day at 0.2% WO_3 passes to waste.

Flotation Reagents

The temperature of flotation pulp is maintained at 25°C. The reagents orso, emulsol, soda ash and sodium silicate are used as indicated in the following table.

Reagent	Pounds per Ton	Point of Addition
Orso	1.50	Rougher Flotation
Emulsol	0.15	Rougher Flotation
Soda Ash	1.00	Rougher Flotation
Sodium Sil	0.10	Cleaner Flotation

Flotation Concentrate Dewatering

Final Flotation concentrates pass to a settling tank where overflow passes to waste and underflow to a 3' x 8' Drying Pan. After drying, the concentrates are bagged and shipped to Salt Lake, where they are later subjected to chemical treatment to produce required grade.

Retreatment of Gravity Concentrates

Retreatment Agitation: The cleaner jig concentrates plus the table concentrates usually carry phosphorus, copper and arsenic in excess of shipping requirements. These concentrates are acid treated in ton lots in a 4' x 4' agitator. Agitation is carried out for a period of two hours at a dilution of 1:1 using 125 pounds of 39% HCL per ton of solution. During this cycle the bulk of the phosphorus and copper carbonate is removed. Following the two hour agitation period the pulp is allowed to settle for two hours when the acid solution is siphoned off and sent to a small settling tank 4' x 4' x 5' in size. The concentrate in the agitator is then given two separate fresh water washes for a total period of 1½ hours.

Retreatment Regrind: The acid treated concentrate is fed to a 28" Hardinge ball mill in open circuit where it is reground to approximately 50% minus 200 mesh. The mill discharges to a No. 250 Denver "Sub-A" Unit Flotation Cell where solids from the solution settling tank are also added.

Retreatment Flotation: Unit cell flotation is carried out with a pulp density of 60% solids in an alkaline circuit for removal of excess copper and arsenic sulphides. About 30 pounds of concentrates are produced each day. The concentrates, largely sulphide carrying 40 to 60 ounces in gold, are settled in a 90 gallon drum, then bagged and shipped to the smelter at Trail, B. C.

Retreatment Reagents:

Reagent	Pounds per Ton
Soda Ash	2-3
Z-9	0.1
Aerofloat 25	0.06
Pine Oil	0.1

Final Gravity Concentrate (Unit Cell Tailing): The tailing from the Unit cell, which is the WO_3 residue after leaching and flotation passes to a 4' x 4' x 4' settling tank. The settled solids are dried on a pan dryer and this product, amounting to 1.34 tons per day at 73.5%, WO_3, is bagged and shipped.

Leaching and Flotation Analysis

	Wt. %	% WO_3	% P	% S	% Cu
Leach Sol.		1.25	1.55	Grammes	Litre
Flot. Conc.	1.0	21.5	.25	17.9	11.0
WO_3 Residue	99.0	73.4	.05	.03	.05
	100.0	73.1	.35	.21	.17

	% As	Oz. Au	% Sb	% Sn	% Mn	% Mo
Leach Sol.	(1.9 sol: 1 solids)					
Flot. Conc.	9.8	58.4	.04	.04	.1	Nil
WO_3 Residue	.06	.10	.02	.01	.1	.05
	.16	.69	.02	.01	.1	.05

Distribution (Leaching and Flotation)

	% WO_3	% P	% S	% Cu	% As	% Au
Leach Sol.	.33	84.5	.....	.....		.....
Flot. Conc.	.30	.7	85.6	69.0	62.4	84.8
WO_3 Residue	.06	.10	.02	.01	.1	.05
	100.00	100.0	100.0	100.0	100.0	100.0

Mill Metallurgy

	Tons	% WO_3	Tons WO_3	Recovery
Feed	75.	1.71	1.281	100.0
Gravity Conc.	1.345	73.0	.98185	76.7
Flot. Conc.	1.1	14.0	.15400	12.0
Tails	72.555	.2	.14510	11.3
			1.28095	

Tungsten Milling and Current Metallurgy at Canadian Exploration Limited

*By R. J. McLEOD**

(Annual Western Meeting, Vancouver, November, 1956)

(Transactions, Volume LX, 1957, pp. 79-84)

Introduction

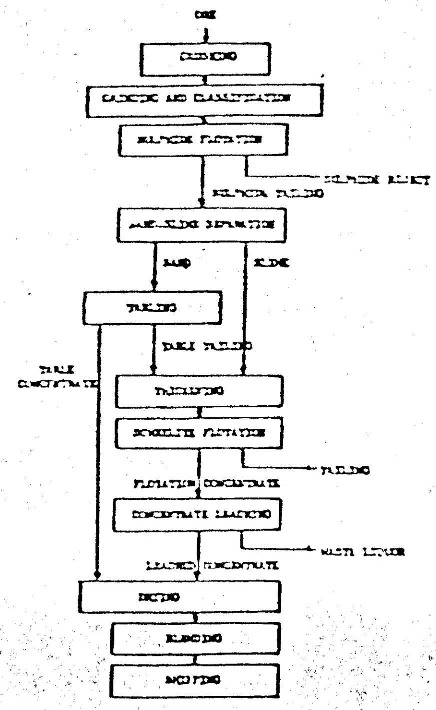
THIS PAPER outlines the milling method and metallurgy in the tungsten mill of Canadian Exploration Limited, Salmo, B.C.

The history of the property, mine development, and the underground crusher-conveyor system have been dealt with in earlier papers (see references at end of paper). It is sufficient here to note the developments in tungsten milling from the erection of the present mill.

In 1951 the first section of the tungsten mill was built by Canadian Exploration Limited for the Federal government to treat ore from two government-owned blocks of ground. Operation of this mill commenced in November, 1951, with a rated capacity of 250 tons per day. Further exploration by the Company resulted in the discovery of another tungsten deposit outside the government-owned blocks. To treat ore from this deposit, a second section, a duplicate of the first, was added to the mill by the Company and completed in April, 1952. In September, 1952, the Company purchased the government's interests in the mine and the mill, and operated the whole as one unit.

There have been no further additions to the mill but, by small alterations and better understanding of the metallurgy, the milling rate has been increased steadily to the present 700 tons per day.

The mill is situated a short distance from the 3,800-foot level of the Emerald mine, which is about 2,000 feet above the valley floor. The mill and property is serviced by two all-weather roads. Supplies are hauled up, and the concentrate down, these roads by local and long-distance trucking firms. Ample water is obtained by a pipe line from Lost creek. Hydro-electric power is supplied by the West Kootenay Power and Light Company.



Tungsten mill flow-sheet.

Mineral Associations

The ore treated at the tungsten mill may be referred to as a normal tacite (locally called 'skarn') ore. The ore mineral is scheelite with trace amounts of powellite. The principal gangue minerals are calcite, pyrrhotite, pyrite, apatite, quartz, mica, siderite, garnet, hornblende, and pyroxene, with very minor amounts of fluorite and molybdenite.

Crushing

Most of the ore is crushed in the underground crushing plant, described in an earlier paper (2). The original crushing plant for the mill, on surface, still remains in place, and a small amount of ore from the lower levels of the mine is still crushed in this plant. At present, this ore is about 10 per cent of the whole and, on completion of the inclined shaft now being sunk, it will be diverted to the underground crushing plant.

The underground crushing plant operates one shift per day on tungsten ore, at a rate of 150 tons per hour. The equipment consists of a 3,000-ton coarse-ore pocket, 60-in. pan-feeder, 36 in. by 48 in. jaw-crusher set for 4½-inch product, and a No. 760 Allis Chalmers hydrocone set at ⅝ in. The hydrocone crusher is in closed circuit with a Tyler-Tyrock, two-deck screen delivering a 1¼-inch product to the mill fine-ore bins.

The mill crushing plant, capacity 40 tons per hour, operates one shift per week and comprises a 200-ton coarse-ore bin, an 18 in. by 32 in. Telsmith Wheeling jaw-crusher, a 3 ft. standard Symons cone crusher, and a 3 ft. by 8 ft. two-deck Denver Dillon screen. The jaw-crusher discharge is conveyed to the screen. The oversize is crushed in the cone-crusher, the product joining the screen-undersize and being conveyed to the mill fine-ore bins.

Two fine-ore bins receive the ore from the underground crusher and the mill crusher. They are made from 2 in. by 12 in. B.C. fir, nailed on the flat. The bins are flat-bottom, 32 ft. high by 24 ft. diameter, and octagonal in shape. Each bin has a theoretical capacity of 1,000 tons; the live capacity is approximately 600 tons each. The bins are discharged through Hardinge constant-weight feedometers, two per bin.

Grinding

The grinding is done in a two-stage circuit, a rod-mill and ball-mill combination, with the rod-mill

*Mill Superintendent, Canadian Exploration Limited, Salmo, B.C.

TABLE I.—SIZE ANALYSES OF GRINDING CIRCUIT, SEPTEMBER, 1956

		Mill Feed		R.M. Disc.		Prim.CL.Sand.		B.M.Disch.		Sec.CL.Sand.		Prim.CL.O'Flow.		Sec.CL.O'Flow.		Combined Class O'Flows	
		% On	Cum	% On	Cum	% On	Cum	% On	Cum	% On	Cum	% On	Cum	% On	Cum	% On	Cum
	+.742"	16.1	16.1														
-.742"	+.525"	24.9	41.0														
-.525"	+.371"	10.6	51.6														
-.371"	+.263"	8.8	60.4														
-3	+4	5.0	65.4														
-4	+6	4.8	70.2														
-6	+8	2.4	72.6														
-8	+10	2.7	75.3	6.4	6.4	11.2	11.2										
-10	+14	2.9	78.2	8.4	14.8	14.3	25.5										
-14	+20	2.2	80.4	10.8	25.6	18.2	43.7	3.4	3.4	7.4	7.4						
-20	+28	1.6	82.0	9.1	34.7	13.7	57.4	4.6	8.0	8.1	15.5						
-28	+35	1.9	83.9	8.9	43.6	12.2	69.6	10.4	18.4	15.3	30.8	2.4	2.4	4.0	4.0	3.5	3.5
-35	+48	1.4	85.3	7.1	50.7	7.9	77.5	12.8	31.2	16.5	47.3	4.4	6.8	7.4	11.4	6.6	10.1
-48	+65	1.3	86.6	6.3	57.0	5.1	82.6	14.5	45.7	15.7	63.0	6.8	13.6	12.0	23.4	10.5	20.6
-65	+100	1.6	88.2	6.2	63.2	3.7	86.3	13.5	59.2	13.0	76.0	8.7	22.3	13.3	36.7	12.8	33.4
-100	+150	1.3	89.5	5.1	68.3	2.5	88.8	9.4	68.6	7.1	83.1	8.6	30.9	12.0	48.7	10.9	44.3
-150	+200	0.9	90.4	3.7	72.0	1.5	90.3	6.1	74.7	3.9	87.0	6.9	37.8	8.8	57.5	7.8	52.1
-200		9.6	100.0	28.0	100.0	9.7	100.0	25.3	100.0	13.0	100.0	62.2	100.0	42.5	100.0	47.9	100.0
	Total	100.0		100.0		100.0		100.0		100.0		100.0		100.0		100.0	
% Solids				76.4				65.0				34.7		47.3		41.5	
Calculated proportion of Mill Feed												30%		70%			
Weighted average of two classifier o'flows, @ 30:70 ratio (% solids)																41.5	

in open circuit with a classifier as the first stage. The second stage is the ball-mill in closed circuit with a classifier. Approximately one-third of the feed leaves the circuit in the overflow of the rod-mill classifier.

The classifier sand is conveyed to the second stage by a screw-conveyor, which discharges to the scoop-box of the ball-mill. The remaining two-thirds of the feed leaves the circuit in the overflow of the secondary classifier. It is necessary to return a portion of the primary classifier overflow through the rod-mill to facilitate the entrance of new feed and assist the flow of coarse ore through the rods. This has raised the capacity of the rod-mill from 600 tons per day to 700 tons.

The equipment used in the grinding circuit consists of two 6 ft. by 12 ft. Dominion mills, overflow type, with combination scoop and drum feeders and driven by 200 h.p. motors and speed reducers. The mills are equipped with single-wave type manganese liners. The classifiers used in the grinding circuit are two 48 in. by 28 ft. high-weir Akins machines.

The rod-mill is charged with 3 in. diameter by 11 ft. 3 in. rods occupying 45 per cent of the mill volume, and the ball-mill with 2 in. quick-quenched cast-steel balls occupying about 50 per cent of its volume. Each mill draws maximum horsepower.

The original speed of both mills was 18 r.p.m., which is 55.7 per cent of critical speed. Two speed changes were made in the mills. At the first change, the ball-mill was speeded up to 24.4 r.p.m., 75.5 per cent of critical, raising the capacity to about 575 tons per day.

At the second change, the speed of both mills was increased, the rod-mill to 24.4 r.p.m., 75.5 per cent of critical, and the ball-mill to 26.9 r.p.m., 83.3 per cent of critical, resulting in a capacity of 700 tons per day.

Rod consumption is 0.70 lb. per ton of ore and ball consumption 0.90 lb. per ton. The densities of the various parts of the circuit are as follows: rod-mill discharge, 76-80 per cent solids; primary classifier overflow, 45 per cent solids; ball-mill density, 70 per cent solids; and secondary classifier overflow, 45 per cent solids.

The size analyses of grinding circuit products are given in Table I.

SULPHIDE FLOTATION

Sulphur occurs in the ore mainly as pyrrhotite; there is little pyrite. The head assays 4 to 7 per cent sulphur. The first step in the flotation process is to remove as much sulphide mineral as possible, in order to keep the final scheelite concentrate below 0.5 per cent sulphur. It is necessary also to reduce the heavy load of sulphide, which would interfere with tabling in the gravity section. The sulphide which does not float readily is depressed in the scheelite flotation.

A bank of No. 24 Denver *Sub A* flotation machines, with fourteen rougher cells and four cleaner cells, is used for sulphide flotation. Conical-disc impellers and wear-plates are used. The cells were equipped originally with a 10 h.p. motor dual drive. This has been replaced by 15 h.p. motors, and the impeller speed has been increased from 275 to 320 r.p.m. The peripheral speed of the impellers, which was approximately 1,400 ft. per min., is now 1,800 ft. per min. A great improvement was noted when the speed was increased; the aeration of the cells was doubled, which increased the froth-making ability of the machine. This resulted in increased efficiency and a saving of frothing and collecting reagents.

No conditioners and no conditioning agents are used in the flotation of the sulphides. A bulk float is made and cleaned once. The cleaned sulphide concentrate is discharged to waste, while the pulp from the tailing end of the machine is pumped to the de-sliming circuit. The assay of the sulphide concentrate is 0.10 per cent WO_3, and represents a loss of 1.5 to 2.0 per cent of the scheelite in the ore. The flotation is carried on at *pH* 8.4, the natural *pH* of the ore, and at a density of 40 per cent solids.

The reagents used per ton of ore are: pentasol xanthate, 0.20 lb.; Aerofloat 25, 0.133 lb.; copper sulphate, 0.05 lb.; and amyl alcohol, 0.02 lb.

At one time, sulphuric acid was used in the flotation of the sulphides. The floatability of pyrrhotite was improved, but the acid was suspected of having a depressing effect on scheelite in the subsequent scheelite flotation, and was discontinued. At times, a large amount of calcium sulphate was formed, which buffered the pulp in the scheelite flotation circuit, and the desired *pH* could not be obtained; also, the consumption of soda-ash was high in the scheelite circuit, and xanthate consumption was excessive in the sulphide circuit. The corrosive effect of the acid has required much replacement and repair to tanks and mechanisms in the sulphide flotation cells.

Tabling

The purpose of tabling is to obtain a high-grade gravity concentrate that will meet specifications without further treatment. The recovery of scheelite from the circuit at this point reduces the future handling and treatment in the form of flotation concentrate.

The amount of gravity concentrate produced depends on the fineness of the scheelite in the ore, the fineness of grinding, and the efficiency of the de-sliming circuit. The recovery here varies between 25 and 45 per cent of the WO_3 in the head. Efficient tabling requires a de-slimed feed, hydraulically sized.

The equipment used in the de-sliming circuit is three 12 in. by 20 deg. Dorrclones, and a Dorrco *Type EX*, five-compartment sizer, together with the necessary pumps.

The Dorrco sizer is a Fahrenwald-type hydraulic classifier in which the sizing is effected by controlling the densities in the different pockets with a rising current of water, and regulating the rate of discharge of pulp. The heavier the density, the coarser the screen-size of discharge. The spigot discharge is regulated by hydrostatic pressure operating through a 'Pressuretrol' and motors.

Following the sulphide flotation the feed is pumped to two primary 12-in. Dorrclones in parallel. The overflows from these by-pass the tables and flow directly to two 40-ft. thickeners; the underflows are further cleaned in one secondary 12-in. Dorrclone, and finally in the sizer. The overflows from the secondary Dorrclone and sizer can be closed-circuited or can flow to the thickeners. A sized product is discharged from each of the five compartments of the sizer into revolving distributors, which feed the tables in groups.

The tables are No. 6 Deister super-duty diagonal deck, with linoleum deck and hardwood riffles. They are pitched both longitudinally and transversely according to the size of feed being tabled. On the coarse tables, the longitudinal pitch is 1/4 in. per ft., and the transverse 1 1/8 in. per ft. The fine tables are pitched 1/8 in. and 3/4 in. per ft. The speed of the tables is 314 strokes per minute and the length of the stroke varies from 3/8 to 3/4 in.

In the operation, eighteen tables are used for roughing, four for middling, and four for final concentrate clean-up. The rougher tables are grouped according to sizing. The feeds to the middling tables and to the final cleaning tables are sized hydraulically by small sizers.

As the pulp flows over the rougher tables, the scheelite and heavy minerals progress toward the concentrate end; the tailing is discharged along the tailing edge to a launder, and is pumped to the thickeners for flotation. At the concentrate end of the rougher tables, a scheelite concentrate is cut off and pumped to the final cleaning tables. A middling cut is taken also, de-watered in a 12-in. Dorrclone, and flows to the middling tables. The concentrate from the middling tables is pumped to the final cleaning tables. The middling table tailing is de-watered in a 12-in. Dorrclone and flows to the grinding section for re-grinding. Finished concentrate assaying 76 per cent WO_3, 0.12 per cent sulphur, and 0.02 per cent phosphorus, is cut from the final cleaning table. The tailing from the latter is pumped ot the middling tables.

The concentrate from the rougher and middling tables contains a small amount of pyrrhotite and pyrite. Before this product goes to the cleaning tables these minerals are removed by a Dings H.M. Crockett separator and a small flotation circuit of two rougher cells and two cleaner cells. The rejects are returned to the grinding circuit.

The size analysis of the de-sliming circuit and tabling is given in Tables II and III.

Thickening

Two 40-foot thickeners receive the table tailing, the slime fraction of the ore, Dorrclone overflows from middling circuit, and miscellaneous wash water. The purpose is to thicken the pulp for the scheelite flotation to follow. Thickener overflow water is used in the de-sliming circuit and tabling circuit.

TABLE II.—SIZE ANALYSES OF DE-SLIMING CIRCUIT, SEPTEMBER, 1956

MESH	SULPHIDE FLOT'N. TAIL		TABLE TAIL		SLIME FEED TO THICKENER		PRIMARY DORRCLONE u'flow		PRIMARY DORRCLONE o'flow		SECONDARY DORRCLONE u'flow		SECONDARY DORRCLONE o'flow	
	% On	Cum	% On	Cum	% On	Cum	% On	Cum	% On	Cum	% On	Cum	% On	Cum
+35	2.5	2.5	3.1	3.1			3.4	3.4			3.2	3.2		
-35 +48	5.5	8.0	8.2	11.3			7.3	10.7			7.3	10.5		
-48 +65	9.6	17.6	15.5	26.8			13.4	24.1			14.2	24.7		
-65 +100	12.8	30.4	17.7	44.5	2.0	2.0	16.3	40.4	0.8	0.8	18.3	13.0	0.5	0.5
-100 +150	11.2	41.6	16.3	60.8	0.8	2.8	14.2	54.6	0.6	1.4	16.1	59.1	0.8	1.3
-150 +200	8.5	50.1	10.8	71.6	1.2	4.0	10.0	64.6	0.7	2.1	11.1	70.2	1.6	2.9
-200	49.9	100.0	28.4	100.0	96.0	100.0	35.4	100.0	97.9	100.0	29.8	100.0	97.1	100.0
TOTAL	100.0		100.0		100.0		100.0		100.0		100.0		100.0	
% SOLIDS	24.4		29.3		9.2		60.0		10.3		64.9		11.5	

MILL FEED RATE 660 tons per day. Feed containing some oxidized surface ore.

TABLE III.—SIZE ANALYSES OF DORRCO SIZER PRODUCTS, SEPTEMBER, 1956

MESH	NO. 1 SPIGOT % On	NO. 1 SPIGOT Cum	NO. 2 SPIGOT % On	NO. 2 SPIGOT Cum	NO. 3 SPIGOT % On	NO. 3 SPIGOT Cum	NO. 4 SPIGOT % On	NO. 4 SPIGOT Cum	NO. 5 SPIGOT % On	NO. 5 SPIGOT Cum
+35	4.5	4.5	4.2	4.2	1.6	1.6				
-35 +48	9.9	14.4	10.2	14.4	7.9	9.7	1.8	1.8		
-48 +65	17.0	31.4	17.7	32.1	17.6	27.3	8.6	10.4		
-65 +100	19.2	50.6	19.5	51.6	20.6	47.9	19.2	29.6	1.7	1.7
-100 +150	15.6	66.2	15.5	67.1	16.3	64.2	20.6	50.2	12.5	14.2
-150 +200	10.3	76.5	9.2	76.3	9.5	73.7	14.0	64.2	18.4	32.6
-200	23.5	100.0	23.7	100.0	26.3	100.0	35.8	100.0	67.4	100.0
TOTAL	100.0		100.0		100.0		100.0		100.0	
% SOLIDS	45.0		52.3		47.4		49.0		24.3	
% DISTRIBUTION OF SPIGOT PRODUCTS	19.7		22.1		18.3		27.0		12.9	

MILL FEED RATE 660 Tons per day. Feed containing some oxidized surface ore.

A settling agent, either Separan 2610 or Aerofloc 3000, 0.005 lb. per ton, is used to help thickening. The use of these reagents does not affect the scheelite flotation.

SCHEELITE FLOTATION

The scheelite is floated with fatty acid as the collector. A separation is made of scheelite and other cacium minerals in a highly dispersed pulp.

Prior to flotation, the thickened pulp is conditioned for 16 minutes in four 5 ft. by 6 ft. Denver conditioners. The conditioning reagents are added here. The alkalinity of the circuit is carefully controlled and kept at a *pH* of 10.1 to 10.2, with a view to controlling the conversion of oleic acid to soap. An automatic *pH* control system, with the electrode assembly immersed in No. 2 conditioner, is used to control the amount of soda-ash fed. The reagents used, in lb. per ton, and their function, are as follows: soda ash, 2.5, alkalinity control; sodium silicate, 0.25, depressant for quartz and silicates; quebracho, 0.80, depressant for calcite; sodium cyanide, 0.15, depressant for sulphides. All of these reagents are dispersants. Oleic acid is the collector and is stage fed, about 0.40 lb. per ton being used.

The feed is kept at about 45 per cent solids and at a temperature between 16 and 20 deg. C. If the temperature drops much below 15 deg.

TABLE IV.—ELUTRIATION OF TUNGSTEN FLOTATION TAILINGS
Week composite, July 16-22, 1956

		WEIGHT % Frog.	WEIGHT % Cum.	Assay % WO3	% Distribution of WO3 Frog.	% Distribution of WO3 Cum.
+48	Mesh	10.4	10.4	0.10	9.8	9.8
-48 +65	Mesh	11.7	22.1	0.10	11.1	20.9
-65 +100	Mesh	13.5	35.6	0.08	10.2	31.1
-100 +325	Mesh	35.3	70.9	0.05	16.7	47.8
-325 + 26	Microns	13.2	84.1	0.08	10.0	57.8
-26 + 13	Microns	5.1	89.2	0.09	4.3	62.1
-13	Microns	10.8	100.0	0.37	37.9	100.0
TOTAL		100.0		0.10	100.0	

Notes:

325 mesh aperture 42 microns.
Elutriation sizes are for quartz.

it is difficult to maintain a froth on the machines; if it is above 20 deg., too much froth is produced. A 100 h.p. boiler is used to provide steam for heating the pulp, the steam being introduced into No. 1 conditioner.

No. 24 Denver *Sub A* machines are used for scheelite flotation. They are equipped with 15 h.p. dual drives. Sixteen rougher and four cleaner cells are used and are arranged for three cleaning stages.

The average WO_3 content of the head to the scheelite flotation circuit is 0.50 per cent. The concentrate produced is 80 per cent WO_3, with 0.24 per cent sulphur and 3.5 per cent phosphorus, and varying amounts of calcite, silicates, and occasionally fluorite. It is filtered on an American-type disc filter to about 14 per cent moisture. The average flotation tailing is 0.10 per cent WO_3.

A size analysis of the flotation tailing is given in Table IV.

LEACHING

The flotation concentrate must be up-graded to at least 60 per cent

T/ V.—METALLURGICAL BALANCE, SEPT. 1/ MAY 31/66

	WEIGHT TONS	% WO3	WO3 UNITS	% DIST. WO3
Mill Feed	150,125.00	0.71	106,137.54	100.00
Iron Concentrate	12,294.10	0.12	1,475.30	1.39
Table Concentrate	470.77	75.03	35,319.58	33.28
WO3 Flotation Feed	137,360.13	0.50	69,342.66	65.33
Leached Concentrate	822.57	64.14	52,758.72	49.71
Leaching Loss			1,076.71	1.01
Leach Feed	1,998.35	26.94	53,835.43	50.72
WO3 Flotation Tailing	135,361.78	0.11	15,507.23	14.61
Total Tailing	148,831.46	0.12	18,059.24	17.01

Total Marketable Recovery 82.99%

WO_3 for commerce, and the specifications must be met for phosphorus and other impurities. This is effected by dissolving the calcite and apatite with hydrochloric acid. If silicates or fluorite are present they are unaffected by this treatment and dilute the final concentrate.

The equipment used in the leaching process must be acid-proof. The leaching is done in 8 ft. by 9 ft. B.C. fir stave tanks, painted inside and out with Quigley acid-resisting paint. The agitating mechanisms are rubber-covered ship-type propellers and shafts supplied by the Denver Equipment Company. Two tanks 10 ft. diameter by 10 ft. high, similar in construction to the leach tanks, but without mechanisms, are used as decant tanks.

The leached concentrate is filtered in a 42 in. Shriver press with 42 chambers, of the open delivery and washing type, equipped with a 'Hydro Kloser'. Dynel filter cloth and maple-wood plates and frames are used. The acid-treated concentrate is pumped to the press by a Shriver 45 g.p.m. rubber-lined diaphragm pump. A Durichlor pump of high-silicon iron alloy is used for pumping solution. Two Brooks Rotameters are used to indicate the rate of flow of acid to the leaching tanks. The leaching and decant tanks are hooded, and the vapours are exhausted by a ventilating system. The acid storage consists of three rubber-lined steel tanks, 9 ft. by 25 ft. 6 in. One tank is located at the railway at Salmo and the other two are at the mill site. The total storage is 171 tons of acid.

The leaching is carried on as a batch process in order to obtain close control. The hydrochloric acid is fed slowly to give a selective dissolution of the gangue minerals. If a concentration of calcium chloride is built-up at the start of the leach, the order of dissolution will be calcite, apatite, scheelite. The concentration of calcium chloride will retard the dissolution of scheelite.

Approximately five tons of flotation concentrate is charged into one leach tank. The concentrate is repulped to about 25 per cent solids with water. Hydrochloric acid, 20° Baume, is fed at a varying rate for several hours. As the acid is fed, the charge will effervesce with the liberation of CO_2 during the calcite leach. The effervescence subsides when all the calcite is dissolved. Agitation of the charge is continued until the phosphorus is below 0.05 per cent; then the charge is pumped to the filter press.

The filtrate from the press is discharged to a decant tank to recover possible losses, and is allowed to settle for eight hours before being discarded. The cake in the press is blown with compressed air for thirty minutes to remove phosphoric acid, in order to prevent precipitation of phosphorus in the cake on washing. It is next washed with water for an hour, with the wash-water going to the second tank. The cake is then blown with compressed air for about three hours, when it will average 4 per cent moisture. The press is discharged manually.

A close metallurgical control is kept of the process and losses are less than 2 per cent.

DRYING AND BLENDING

The concentrate from the tabling section averages 10 per cent moisture, and the leached concentrate 4 per cent. These concentrates must be dried to less than 0.5 per cent moisture for blending and shipping.

The table concentrate is dried on two flat, stationary driers, 8 ft. by 21 ft. The drier consists of a steam coil covered by a 35-mesh screen, forming a tray on top of steel bins. One hundred infra-red lamps are suspended in a rack over each drier. The wet table concentrate is sampled and spread on the drier, and falls through the screen when dry. Occasional stirring will hasten the drying. The concentrate remains in the bin until required for blending.

The drier for the leached concentrate was manufactured by the Conveyor Company, of Los Angeles. It is a paddle-type doublescrew conveyor 26 ft. long, screw diameter 9 in., equipped with seventy-five 375-watt infra-red lamps, suspended over the screws. The drier is equipped with a feed hopper.

It is planned to replace the lamps with radiant heating panels. These heating panels are considered superior for light-coloured concentrate because of the longer wave-length of the infra-red emission.

The leached concentrate from the filter press is shovelled into a hopper on the drier. The dried concentrate is discharged continuously to barrels on the floor below, where it is sampled and stored in batches until assays for impurities and grade are available.

A small flat-hearth roaster, complete with bag-type dust collecting system, is installed and may be used when necessary for an additional drier or to reduce the sulphur content of the concentrate.

The blending of the table and the leached concentrates is done in a 24-in. horizontal screw mixer, 18 ft. long, manufactured by the United Steel Corporation, and driven by a 25 h.p. motor.

The blending is done to meet the buyer's specification. The concentrate is shipped in steel cans, content 150 lb. net weight. The canned concentrate is handled and stored on pallets by a Yale Fork Lift truck.

ACKNOWLEDGMENT

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Development of Tungsten Ore Dressing Practice

At Canadian Exploration Limited, Salmo, B.C.

*By H. H. KIPP**

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Introduction

THE DEVELOPMENT of today's ore dressing practice at the tungsten concentrator of Canadian Exploration Limited dates back in 1943, when the Federal government was responsible for the erection of a 300-ton mill that produced 182 tons of high-grade scheelite concentrate in a production period of six weeks. The property was then closed down due to easing of the tungsten requirements for war production.

In 1947, Canadian Exploration Limited took over the property and production was resumed in June of that year, using the same concentrator. However, the operation was terminated in the closing days of 1948 as a result of low tungsten prices. The mill was modified to treat lead-zinc ore and it has operated as a base-metal producer since that time.

In the meantime, the Korean conflict created a renewed demand for tungsten, for immediate use and for stockpiling. In early 1951, the Company commenced design and construction of a new concentrator for the government, and it was placed in operation before the end of the year. The design of the new mill, of 250 ton per day capacity, was based on the experience gained during one and a half year's operation of the old mill.

This paper outlines the changes that have been made in ore dressing practice since the present concentrator commenced to operate. The following description of the ore should assist the reader to understand the ore-dressing problems with which we are concerned.

Ore Character and Mineral Properties Affecting Treatment

The specification limits for marketable scheelite concentrate are as follows:

Tungsten trioxide (WO_3) (minimum)	60.00%
Sulphur	0.50%
Phosphorus	0.05%

The tungsten ore occurs in limestone and skarn country rock, heavily mineralized with pyrrhotite, and with some quartz replacement of limestone. The mill feed averages 0.7 per cent WO_3 and 5 per cent sulphur.

The ore mineral is scheelite, $CaWO_4$, sp. gr. 6.0, friable. It may be concentrated by soap flotation, and is only slightly soluble in dilute hydrochloric acid. The purity of the scheelite is not uniform. It fluoresces under the ultra-violet lamp in many shades of blue, white, and yellow. All mill samples show a mixture of these fluorescent colours, and intermediate mill products show no evidence of segregation by fluorescent colour. Crystal growth of the scheelite has been good, with the result that most of it is liberated at about 65 mesh; however, locked scheelite may be found in particles down to 200 mesh. In a coarsely ground product, *e.g.*, 10 mesh, free scheelite occurs as coarse particles up to 20 mesh size. Most of the free scheelite grains in mill products have the characteristic octahedral form or are tabular, but many flat, plate-like splinters break off during grinding. These flat shapes tend to lower recovery in the tabling section. Scheelite is seen to be more finely disseminated in the pyrrhotite than in its other associations. The grain boundaries are strongly knit, with the result that many scheelite grains carry minute to fractional scabs and smears of sulphide and carbonate minerals. In a similar manner, particles of gangue minerals, both sulphide and non-sulphide, carry scabs or smears of scheelite.

The principal sulphide mineral is pyrrhotite, Fe_nS_n, sp. gr. 4.6, magnetic. Pyrrhotite is difficult to remove completely by flotation ahead of tabling, it interferes with tabling scheelite, but it is easily depressed with cyanide in scheelite flotation. Magnetic attractability of the pyrrhotite varies over a wide range. Some specimens show a decided lack of response to a hand magnet placed at close range; others respond strongly to the same magnet. It was found that devices employing magnetic separation were not effective as a simple means of cleaning a finished concentrate. Medium-intensity magnetic fields were needed to remove pyrrhotite of low magnetic attractability, with the result that the smallest scab or inclusion of pyrrhotite on an otherwise pure grain of scheelite would cause the scheelite to report in the magnetic fraction. Sheelite losses then became prohibitive.

Pyrite, FeS_2, sp. gr. 5.1, occurs in lesser amount than pyrrhotite. It is readily removed by flotation ahead of tabling and is depressed by cyanide in scheelite flotation.

Apatite, $(CaF)Ca_4(PO_4)_3$, sp. gr. 3.2, is objectionable because of its phosphorus content. It may report in the table concentrate as 'fines', it is not depressed effectively in the scheelite flotation circuit, but it is soluble in dilute hydrochloric acid, permitting separation from scheelite by acid leach.

Calcite, $CaCO_3$, sp. gr. 2.7, is easily rejected in gravity concentration, difficult to depress in scheelite flotation, but readily soluble in dilute hydrochloric acid, permitting separation from scheelite by acid leach.

Quartz and silicates, sp. gr. 2.6 to 3.2, are depressed by sodium silicate in scheelite flotation, and are readily rejected in tabling, except garnet, which may have a specific gravity as high as 4.2.

Fluorite, CaF_2, sp. gr. 3.2, is not a problem in table concentration, but floats similarly to scheelite. It is not soluble in dilute hydrochloric acid, so small amounts in the mill feed can be a serious diluent in the final concentrate.

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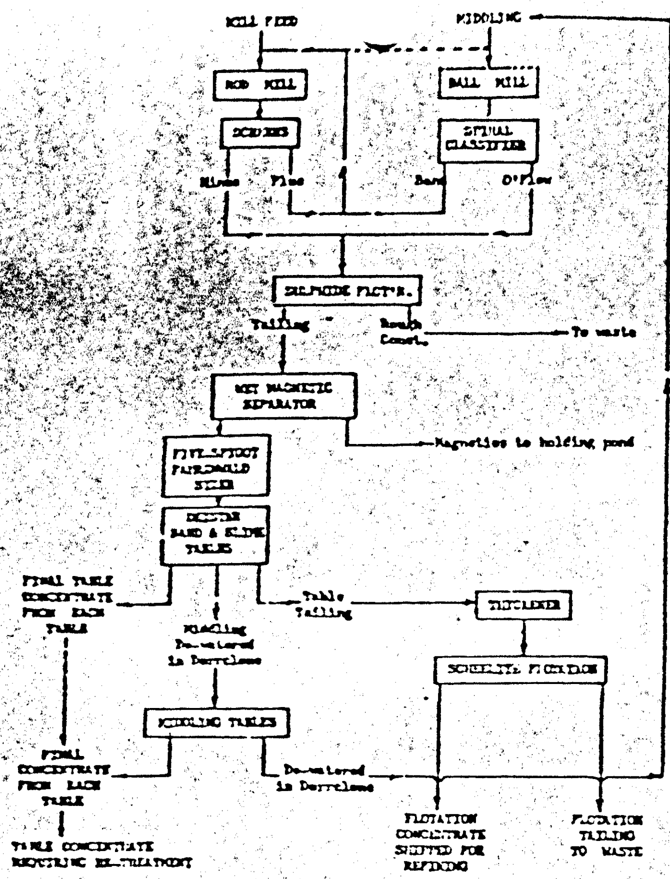

Figure 1.—Original flow diagram of new mill.

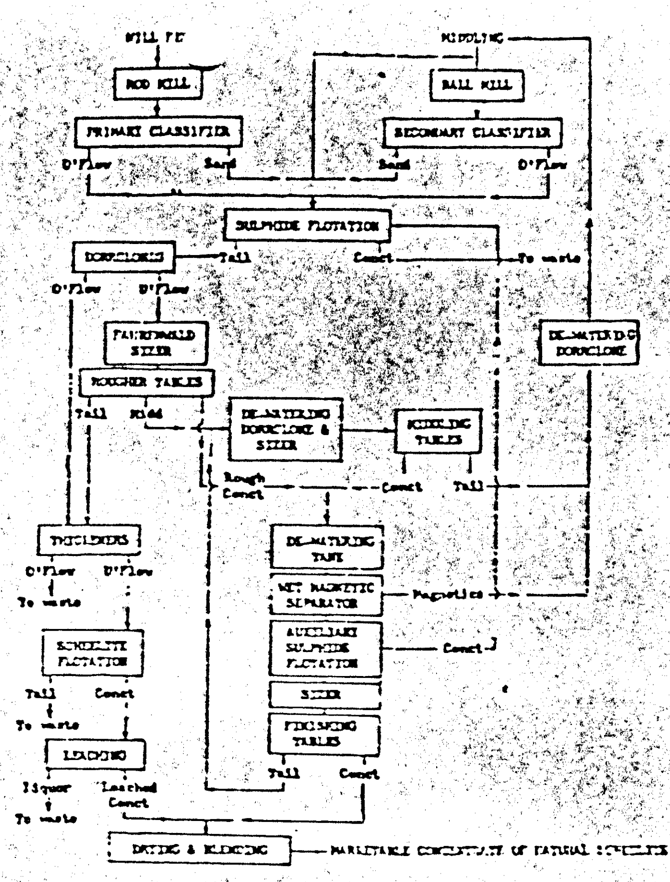

Figure 2.—Flow diagram of present mill.

Original Treatment Plan of the New Mill

Grinding

The original treatment plan of the new mill is illustrated in Figure 1. Both rod-mill grinding and screen-sizing were adopted, to reduce over-grinding of scheelite. The spiral classifier was installed as a stand-by unit for use when the screens were shut down for repair or cleaning. It could be used also to wash fines from the screen oversize before returning this product to the rod-mill.

A ball-mill was used to re-grind table middling. Provision was made to divert part or all of the classifier sand to either the rod-mill scoop box, or the ball-mill scoop box, as required.

Sulphide Removal

In preparation of the feed for tabling, pyrrhotite and pyrite were removed from the pulp by flotation and wet magnetic separation. Complete removal was aimed for, but was not achieved.

Tabling

The original treatment plan called for the separation of a marketable concentrate from each rougher table and each middling table. This objective was not reached. Residual sulphide in the rougher table feed interfered with the separation of scheelite, making it necessary to admit some of the sulphide into the concentrate if a reasonable recovery were to be made. This produced a concentrate satisfactory in grade, but above specification in sulphur. The concentrate from the middling tables was even higher in sulphur, and lower in grade, because of the large proportion of sulphide and garnet in the feed to these tables.

Removal of sulphur was all that was required to make the rougher table concentrate marketable. Several months' production was dried and re-treated in a tray-type low-intensity magnetic separator. The non-magnetic product from this operation was marketable, but the magnetic product, which assayed about 80 per cent WO_3, had to be shipped to a refinery in New York for recovery of the scheelite content. A second lot of several months' production was trucked to a local custom smelter, where the sulphur was removed by roasting, leaving a marketable product. Finally, for a period of ten months, a coarser cut was taken from the rougher tables to improve recovery, making a low-grade high-sulphur concentrate, which was combined with the middling table concentrate and shipped to the refinery in New York.

Scheelite Flotation

The flotation section of the mill was conventional in plan. A single conditioning tank preceded the rougher flotation cells. Two-stage cleaning was employed, with a single cell in each stage. The concentrate was filtered and shipped in drums to the refinery in New York.

Results of Original Treatment Method

In the first year of operation, scheelite recovery was 41 per cent in the table concentrate and 29 per cent in the flotation concentrate. These concentrates were not marketable, and required further treatment at the refinery, involving some loss in the process.

Present Treatment Plan of the New Mill

After five years of operation, during which patient experimentation in the plant, and investigation in the laboratory, were carried on, the present-day treatment plan was developed as shown in Figure 2.

Grinding

The use of single-stage rod grinding, in closed circuit with screens, was abandoned in favour of conventional two-stage grinding with a rod-mill in open circuit with a classifier, the sand going to a ball-mill in closed circuit with another classifier.

Experience has indicated that although there is preferential fine grinding of the scheelite in this system, the high proportion of tailing loss that occurs in the minus-five-micron size range owes its origin to natural processes in the orebody rather than to over-grinding of the mill feed.

Sulphide Flotation

The sulphide flotation circuit is still required to remove a large proportion of the pyrrhotite from the pulp along with the pyrite and the minor sulphides. The tabling section has been made to handle a reasonable amount of pyrrhotite, while extracting a clean, marketable scheelite concentrate at a reasonable recovery. Consequently, the urgent need for maximum recovery of pyrrhotite in the sulphide circuit has been circumvented, with the result that the circuit now uses only 20 per cent of its former reagent consumption, requires much less operating attention, and still manages to keep scheelite losses in the sulphide concentrate at a minimum.

Tabling

Research data indicated that the table concentrate being shipped to the refinery for up-grading and removal of sulphur could be brought to market specifications by re-tabling, but an outlet was needed for disposing of the garnet, coarse calcite, and pyrrhotite that was rejected in the process. In adopting the re-tabling system in the plant, this outlet was provided by the middling table circuit, to which the tailing from the re-treatment (retabling) tables was directed. The tailing that was discharged from the middling tables was returned to the main circuit *via* the secondary grinding mill. Later, magnetic separation and auxiliary sulphide flotation were added to the re-treatment circuit.

The following points are significant in the present tabling system:

(1) Sulphide flotation removes most of the sulphide from the whole of the mill feed, but complete removal of pyrrhotite is not attempted.

(2) A small part of the remaining pyrrhotite reports in the concentrate from the rougher and middling tables. This is step-eliminated in the re-tabling circuit by magnetic separation, flotation, and re-tabling. The remainder of the pyrrhotite reports in the middling table tailing and is returned to the grinding circuit.

(3) The re-tabling process is similar in function to the cleaner step in a flotation circuit, in that a high-grade fraction is removed from a rougher concentrate, and the remainder is returned to the main circuit as 'cleaner tailing'.

(4) The sulphide removed in the step-elimination processes is all returned to the main circuit, and eventually reaches either the main sulphide flotation concentrate or the final tailing.

Scheelite Flotation

A conventional flotation circuit continues to be used for floating scheelite, but important changes have been made, as follows:

(1) Stage feeding of oleic acid has been extended to four points of addition.

(2) Three-stage cleaning has replaced two-stage cleaning.

(3) Steam is used to raise the temperature of the pulp from 4°C. to between 16° and 18°C.

(4) The use of the detergent frother *Oronite D*-40 has been discontinued. The soap produced by the reaction between oleic acid and soda-ash is the only source of froth in the present circuit.

(5) Small reagent feeders have been replaced by large disc-and-cup feeders by which changes in feed rate can be made only by adding or taking away standard size cups.

(6) The impeller speed of each cell has been increased by 28 per cent.

(7) An automatic *pH* control system has been installed.

These changes have been effective in lowering scheelite losses in the tailing. The circuit is stable, most fluctuations therein arising from changes in the character of the mill feed.

The flotation concentrate is upgraded to market specification by a simple acid leach to remove calcite and apatite. Fluorite has been troublesome for short periods, producing low-grade leached concentrate, but in time this has been blended with higher-grade concentrate for marketing.

Testing on a pilot-plant scale has indicated that an all-flotation process may be as efficient as the present gravity concentration and flotation process.

Results of Present Treatment Plan

All of the concentrate now produced at the Canadian Exploration Limited tungsten concentrator is marketable. The metallurgical data shown in the following table cover nine months of operation, ending May 31st, 1956.

PRESENT METALLURGICAL RESULTS

Tons milled	150,125
Average mill head, WO_3	0.71%
Table concentrate, tons	470.77
Table concentrate, WO_3	75.03%
Leached flotation concentrate, tons	822.58
Leached flotation concentrate, WO_3	64.14%
Mill tailing, WO_3	0.12%
Recovery in table concentrate	33.3%
Recovery in leached flotation concentrate	49.7%
Recovery, total	83.0%

SUMMARY

The design of the new tungsten mill, over five years ago, was based on the concept that the dressing of a scheelite ore was a proposition for gravity concentration. Scheelite flotation assumed an auxiliary role through which some of the fine scheelite that escaped the gravity section could be scavenged from the gravity tailing before it was sent to the tailing pond.

Some time elapsed before judicious use of existing equipment brought the gravity section to the point where it was able to produce a marketable grade of concentrate from a difficult pyrrhotitic ore.

In the meantime, there developed a better understanding of the process of scheelite flotation. Operation of the scheelite flotation circuit continued to improve until there occurred a reversal of the roles originally planned for the two systems. Today, scheelite flotation is considered to be the important process in tungsten ore-dressing practice at Canadian Exploration Limited.

ACKNOWLEDGMENTS

The writer wishes to thank the management of Canadian Exploration Limited for permission to present and publish this paper. The invaluable assistance of Mr. H. A. Steane, General Mill Superintendent and Chief Metallurgist, and Mr. R. M. Wigglesworth, Metallurgical Consulatnt, in the preparation of the paper is gratefully acknowledged.

(Reprinted from The Canadian Mining and Metallurgical Bulletin, March, 1957)

Printed in Canada

APPENDIX J

UNION CARBIDE CORPORATION
METALS DIVISION
Bishop, California 9351

SCHEDULE FOR PURCHASING SCHEELITE CONCENTRATES WHICH ARE COMPLETELY AMENABLE TO OUR PROCESS*

WO_3 Content		Per Short Ton Unit WO_3 f.o.b. Upper Scheelite Near Bishop, California
Less than	15.00	No Payment
15.00 to	19.99	103.75
20.00	24.99	105.00
25.00	29.99	106.25
30.00	34.99	107.50
35.00	39.99	108.75
40.00	44.99	110.00
45.00	49.99	111.25
50.00	54.99	112.50
55.00	59.99	113.75
60.00	AND UP	115.00

This schedule is not an offer to purchase tungsten concentrates. Do not ship concentrates unless we issue an order to purchase.

Prices are subject to change without notice.

For materials which originate from foreign sources, Seller/Shipper must arrange to pay applicable U. S. Duty and submit evidence of such payment.

*Based on five-pound sample submitted by Seller to the above address. Materials shipped must conform to sample submitted for evaluation. Please mark sample, "Tungsten Sample".

We require approximately three weeks to conduct test work on samples submitted.

Prices apply to Lot deliveries of one dry ton or more. Deductions as show below will be made from regular purchase schedule for Lots of material delivered in quantities of less than one (1) dry ton:

		Deduct
Less than	2000 lbs. (Dry)	\$ 1.00 Unit WO_3
	1500	1.50
	1000	2.00
	500	2.50

or \$ 50.00, whichever is greater

Concentrate particle size must be less than one-quarter (1/4) inch.

Concentrates which contain excessive moisture (generally in excess of 5% H_2O) are not acceptable.

Deliveries are limited from 8:00 a.m. to 3:00 p.m., Monday through Friday, except holidays.

Revised: March 4, 1978

<u>ATTACHMENT A</u>

SAMPLING AND ANALYTICAL PROCEDURE FOR TUNGSTEN BEARING CONCENTRATES AND RESIDUES FROM OTHER PRODUCERS/SUPPLIERS

Settlement will be based on Union Carbide Corporation's sample, weights and moisture determinations by standard practice (i.e., individual cans/ sacks will be "thief" sampled upon arrival; a portion of such sample will be used for moisture determination and another to provide for sample pulps on which analyses will be based).

Material received at one time will be sampled as a lot unless the quantity exceeds a reasonable amount.

One pulp will be mailed to producer/supplier within ten (10) days after delivery of the material. In case of disagreement on assays as to any constituent of the material, an umpire shall be selected in rotation from a list of umpires approved by Union Carbide Corporation, whose assays will be final if within the limits of the assays of the two parties; if not, the assay which is nearer to that of the umpire shall prevail. The party whose assay is further from that of the umpire shall pay the cost of the umpire's assay for the constituent of the concentrate which is in dispute. In the event that the umpire's assay is equally distant from the assay of each party, costs will be split equally. In the case of the Seller's failure to make or submit assays, the Buyer's assays shall govern. After sampling, the concentrates may be placed in process, commingled or otherwise disposed of by the Buyer.

Deliveries must be made Monday through Friday, holidays excepted, between the hours of <u>8:00 a.m. and 3:00 p.m.</u>

Settlement for material delivered will be made within 7 to 14 days.

LAW:gp

Revised: March 15, 1977

C.N. Forster

UNION CARBIDE CORPORATION
METALS DIVISION
shop, California 93514

f.i.e

SCHEDULE FOR PURCHASING SCHEELITE CONCENTRATES WHICH ARE COMPLETELY AMENABLE TO OUR PROCESS*

WO_3 Content		Per Short Ton Unit WO_3 f.o.b. Upper Scheelite Near Bishop, California
Less than	15.00	No Payment
15.00 to	19.99	93.75
20.00	24.99	95.00
25.00	29.99	96.25
30.00	34.99	97.50
35.00	39.99	98.75
40.00	44.99	100.00
45.00	49.99	101.25
50.00	54.99	102.50
55.00	59.99	103.75
60.00	AND UP	105.00

MAR 23 1979

This schedule is not an offer to purchase tungsten concentrates. Do not ship concentrates unless we issue an order to purchase.

Prices are subject to change without notice.

For materials which originate from foreign sources, Seller/Shipper must arrange to pay applicable U. S. Duty and submit evidence of such payment.

*Based on five-pound sample submitted by Seller to the above address. Materials shipped must conform to sample submitted for evaluation. Please mark sample, "Tungsten Sample".

We require approximately three weeks to conduct test work on samples submitted.

Prices apply to Lot deliveries of one dry ton or more. Deductions as shown below will be made from regular purchase schedule for Lots of material delivered in quantities of less than one (1) dry ton:

		Deduct
Less than	2000 lbs. (Dry)	\$ 1.00 Unit WO_3
	1500	1.50
	1000	2.00
	500	2.50

or \$ 50.00, whichever is greater

Concentrate particle size must be less than one-quarter (1/4) inch.

Concentrates which contain excessive moisture (generally in excess of 5% H_2O) are not acceptable.

Deliveries are limited from 8:00 a.m. to 3:00 p.m., Monday through Friday, except holidays.

Revised: February 5, 1979

ATTACHMENT A-1

AFFIDAVIT OF OWNERSHIP OF TUNGSTEN ORE CONCENTRATES

State of California)
) ss.
County of ________)

________________________________, being duly sworn, hereby
Name of Affiant
deposes and says:

1. That he is the owner of all right, title and interest in and to those certain tungsten ore concentrates to be delivered pursuant to ____
Date of Letter Offer

2. That he has not entered into any presently binding agreement with any third party which would in any way restrict him from offering such tungsten ore concentrates to Union Carbide, and such offer is valid and without condition or restriction.

3. That he agrees to indemnify Union Carbide Corporation against and hold it harmless from any breach of any representation or warranty.

IN WITNESS WHEREOF, ____________________________ has hereunto
Name of Affiant
set his hand and seal this ______________________ day of

______________________, 19 ___.

____
Signature

Sworn to before me this

_________________________ day of _____________________, 19 __.

____
Notary Public

(SEAL)

ATTACHMENT A

SAMPLING AND ANALYTICAL PROCEDURE FOR TUNGSTEN BEARING CONCENTRATES AND RESIDUES FROM OTHER PRODUCERS/SUPPLIERS

Settlement will be based on Union Carbide Corporation's sample, weights and moisture determinations by standard practice (i.e., individual cans/sacks will be "thief" sampled upon arrival; a portion of such sample will be used for moisture determination and another to provide for sample pulps on which analyses will be based).

Material received at one time will be sampled as a lot unless the quantity exceeds a reasonable amount.

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Deliveries must be made Monday through Friday, holidays excepted, between the hours of 8:00 a.m. and 3:00 p.m.

Settlement for material delivered will be made within 7 to 14 days.

LAW:gp

Revised: March 15, 1977

(CHECK ONE)

2		UCC CREDIT MEMO
3		UCC DEBIT MEMO
4	X	UCC SUBSTITUTE FOR VENDOR'S INVOICE

UCC COMPONENT NAME AND ADDRESS

Union Carbide Corporation
Metals Division
Bishop, California 93514

DATE May 8, 1978

VENDOR (NAME AND ADDRESS)

Mina Minerals, Inc.
Box 126
Mina, Nevada 89422

SHIPPED TO (IF DIFFERENT FROM VENDOR'S NAME AND ADDRESS)

Cedar Chest mine
Mineral County nev

ACCOUNT CLASSIFICATION	DESCRIPTION	AMOUNT
	Tentative Settlement on Tungsten Ore Concentrates received 4/19/78, Our Lot No. 4-6.	

Gross Weight	Tare Weight	Net Wet Weight	%H2O	Net Dry Weight	%WO3	Units WO3
12880	6300	6580	0.11	6573	44.73	147.01

147.01 Units WO3 @ $110.00/Unit — $16,171.10

Request For Umpire must be submitted within 30 days of this Tentative Settlement date.

UC 103-32G (10-76) TRIPLICATE-FORWARD TO CORPORATE ACCOUNTS PAYABLE DEPT. WITH INVOICE TRANSMITTAL FOR TRANSMITTAL TO VENDOR WITH REMITTANCE

GOTCHA MINERAL CLAIMS

INFORMAL REPORT B. Ryan 1/6/79

INTRODUCTION

This report is a preliminary assessment of some of the accumulated information at hand. This information includes: (1) Geologic map 50' to 1"; (2) Drill logs for eleven Union Carbide diamond drill holes; (3) Union Carbide report for the Boulder Group 1973 (area since restaked as the Gotcha Claims); (4) Geologic report dated 1978 by J. P. Elwell discussing 18 percussion holes.

GENERAL GEOLOGY

Mineralization in the form of coarse grained scheelite is contained in two bands of skarn. The surrounding metasedimentary rocks are mapped as part of the Shuswap Metamorphic Complex. These rocks are complexly folded and isoclinal folds can be expected in the area. The skarn was probably originally a calc silicate in this assemblage but has now been intruded by a post kinematic Cretaceous stock. Faulting in part post dating the Cretaceous stock is evidenced in the area by gouge and intense fracturing found in some outcrops.

Important controls on mineralization are probably (1) faulting; (2) folding; (3) lithology; (4) proximity to Cretaceous intrusion; (5) type of intrusion. There are probably others but the above five will be discussed in the light of what can be interpreted from the information at hand.

FAULTING

There is evidence for faulting on surface and in the DD holes. The surface evidence suggest an east-west rather than north-south trend but gives no information about possible displacements. Two faults have been postulated for one working hypothesis. Evidence for these is as follows. D.D.hole 2 intersects a considerable thickness of skarn, more than appears to outcrop updip, this could be explained by a fault causing the band to be intersected twice in the same hole. (See section for D.D.Holes 2 and 7). The most probable trend for the fault is east - northeast and it may be responsible for the absence of skarn in D.D.Hole 1. A second fault is postulated to explain the non-correlation of skarn from D.D.Hole 2 to D.D.Hole 7 (see section for D. D. Holes 2 and 7). These faults would be normal south dipping with the south side down dropped and they could cause the skarn bands to be repeated in outcrop to the northeast of D. D. Holes 2 and 7. If this is the case it is not apparent as a boulder train or in soil sample results.

FOLDING

The area is located in the Shuswap Complex and the rocks therefore are almost certainly complexly folded with the earlier folds isoclinal. The presence of a least one phase of folding is suggested by tectonic lineations and the occasional tight minor fold. Foliation measurements in the vicinity of the mineralization are reasonably consistent in orientation ruling out the presence of large open folds in the area. Any major folds in the area would probably be isoclinal and therefore best identified by mirror repetitions in the stratigraphy. This can best be checked by looking at the sections for D.D. Holes 1 to 11. There appears to be a 3 member lithologic succession composed of skarn (calc silicates), schist, quartzite. The succession may have quartz monzonite (sill?) above or below it. If this lithologic succession is real then there is no evidence to suggest that the two skarn bands are separate limbs of the same fold; nor is there any evidence to suggest that they each represent a single isoclinal fold.

LITHOLOGY

Other than the broad scale lithologic succession outlined above not much can be done with the metamorphic rock types. The skarn, whether it is isochemical or not is certainly high temperature, low pressure as indicated by the presence of wollastonite, diopside and idocrase. These minerals are more prevalent to the east of the area adjacent to the leucocratic quartz monzonite. Scheelite is found in conjunction with idocrase and or diopside and in fine grained or coarse grained skarn. It seems to be restricted to the east to within about 250 ft. of the major contact with the leucocratic quartz monzonite.

INTRUSIVE CONTACTS AND INTRUSIVE ROCK TYPES

It is important to differentiate between synkinematic sills and post kinematic stocks. This is best done in the field in part using the presence or absence of tectonic foliation in the rock. The leucocratic quartz monzonite (post tectonic Cretaceous ?) is intersected by D.D. Hole 1 and probably as sills in D.D. Holes 2, 7 and 3. Most of the intersections of intrusive rock in the D.D. Holes probably represent sills though some represent intersections with discordant intrusive phases. This is most clearly the case in D.D. Hole 6. It is possible to interpret intrusive contacts as representing positive dipping topography (i.e. not overturned) and to contour the contacts. This is done for the major contacts and a picture of a north-south embayment or pendant emerges, this pendant is deflected or kinked east-west in the region of the scheelite mineralization. (See accompanying overlay).

ORE POTENTIAL

The engineering report that discusses the results of the percussion drill hole data outlines "drill indicated", "probable" and "possible" ore tonnages and assay values for the two skarn bands.

LOWER SKARN BAND

A drill indicated tonnage of 5200 tons with an assay value of approximately 1.7% is outlined for the lower band. Probable and possible tonnages are 2010 tons. These are assigned an assay value of 2% (high?). If the lower skarn band is considered to have a strike length of about 200 ft. and to dip northwest towards an east-west striking steep fault or intrusive contact then if a 2 metre thickness is assumed for the band it has a total tonnage of 12,000 tons. About 7000 tons are already accounted for, this leaves 5000 tons. If an assay value of (.66%) less than 1% is assigned to this tonnage there are possible additional reserves of 4000 s.t.u.

UPPER BAND

The upper band can be treated in the same way that is projected to the north and the surface area of the slab calculated. The area of the slab already considered as drill indicated is subtracted and the remaining area assigned a thickness of 2 metres and assay value (.66%) less than 1% to give 4000 s.t.u.

SUMMARY

Projecting the strike length of the two skarn bands northwards to an east-west trending intrusive contact or fault provides an additional possible reserves of 8000 s.t.u. over and above those outlined as drill indicated, probable and possible based on the percussion hole data. Further ore reserves should be looked for at deeper depths and to the north of diamond drill holes 2 and 7. In this region the skarn bands may wrap round the side of the intrusion or be repeated by normal faulting.

TO: UNITED MINERAL SERVICES

FROM: A. Magill, Milling Consultant

Subject: MILLING OF UNITED MINERAL SERVICES SCHEELITE ORE

For the purpose of this test, United Mineral Services obtained the use of a flotation, grinding and crushing plant at Lumby, B.C.

The amount of ore treated was approximately 300 ton. During the entire operation, we were plaqued with problems which had a deleterious effect on the effectiveness of the project. The two most serious of these problems was:

1) the lack of sufficient water (shutdowns were required at roughly six hour intervals for a period of three hours);

2) inability to supply a constant feed due to poor design of bins and transfer chutes. (There being no accurate methods of measuring tonnage, all tonnages and reagent additions are approximate.)

The ranges which reagents were fed were soda ash 1.5 to 4.5 lb. per ton. This reagent was fed in widely varying doses as the flotation appeared very sensitive to P.H. change. The optimum range seems to be 1.5 to 2.5 with a P.H. between 9.5 to 10. Sodium silicate .5 to 1 lb. per ton. Pamak was fed between .1 and .2 lb. per ton. Quebracho at .05 to .1.

P.H. control on this ore is quite critical. A low P.H. would appear to cause poor recovery while too high a P.H. appears to be harmful to grade. It would also appear that availability of water addition points on flotation would also be desirable. At times when the grade of conc. was poor the addition of fresh water in moderate quantities to the cleaner and/or concentrate cells was sufficient to increase con grade substantially.

Grinding of the ore appears to be hard so liberation of mineral particles will be a consideration in any full scale operation.

On a whole, I found this ore very manageable and can foresee no serious problems in the flotation of the scheelite.

After observing a natural concentration of scheelite in the corners of launderers, etc., I decided to do a rough test of gravity seperation. By nailing 1" strips of wood on a piece of plywood at 1' intervals, a rough riffle board was formed. Setting this at approximately 12 deg., I washed two shovelsful of material containing approximately 2% WO_3 *and ground to 60% - 200. Visually, the seperation was quite emphatic and substantiated under the mineral light.*

In view of the lack of such things as controlled feed, insufficient water and assay facilities, I have not made any attempt to assess recoveries or economics etc. My conclusions, based on my own ability and my 17 years' experience in mineral seperation, are that providing the remainder of the ore is similar to the + or - 300 ton that was treated by me, there should be no serious problem in achieving a satisfactory seperation.

6 June 1979.

CHEMEX LABS LTD.

212 BROOKSBANK AVE.
NORTH VANCOUVER, B.C.
CANADA V7J 2C1
TELEPHONE [redacted] 984-0221
AREA CODE 604
TELEX: 043-52597

• ANALYTICAL CHEMISTS • GEOCHEMISTS • REGISTERED ASSAYERS

CERTIFICATE OF ASSAY

CERTIFICATE NO. 65295
INVOICE NO. 30486
RECEIVED May 30, 1979
ANALYSED June 12, 1979

TO: United Mineral Services
~~Malabar Mines Ltd.~~
1326 510 W. Hastings St.
Vancouver, B.C.
V6B 1L8
ATTN: Robert A. Dickinson

SAMPLE NO. :	% WO_3	% P_2O_5	Wt %
1 A	9.74	0.94	.41
1 B	8.81	0.89	.39
1 C	8.06	0.77	.34
2	21.8	2.94	1.28

CTA MEMBER CANADIAN TESTING ASSOCIATION

B. Twaites
REGISTERED ASSAYER, PROVINCE OF BRITISH COLUMBIA