W.A. NO. .....

NAME Geol Rpts

SUBJECT .....

.....

.....

GOTCHA

2 OF 2

82M 123

DOCUMENTATION OF PHYSICAL WORK
UNDERTAKEN BY
UNITED MINERAL SERVICES LTD.

ON THE

GOTCHA, GOTCHA 2, MAX 1 AND MAX 2
MINERAL CLAIMS

SITUATED NEAR MAXWELL CREEK,
KAMLOOPS MINING DIVISION

for

PROPERTY FILE

MINISTRY OF MINES AND PETROLEUM RESOURCES
(MINERAL EXPLORATION INCENTIVE PROGRAM)
CONTRACT NO: 16

TO AUGUST 31, 1978

ROBERT A. DICKINSON MURRAY McCLAREN

UNITED MINERAL SERVICES LTD.
1326 - 510 West Hastings Street
Vancouver, B. C.
V6B 1L8

INTRODUCTION

The purpose of this interim report is to document physical work undertaken on the Gotcha Tungsten Property to August 31, 1978. Work consisted of road building, overburden stripping and rock trenching which are part of the "Approved Exploration Activities" of Contract 16, Mineral Exploration Incentive Program.

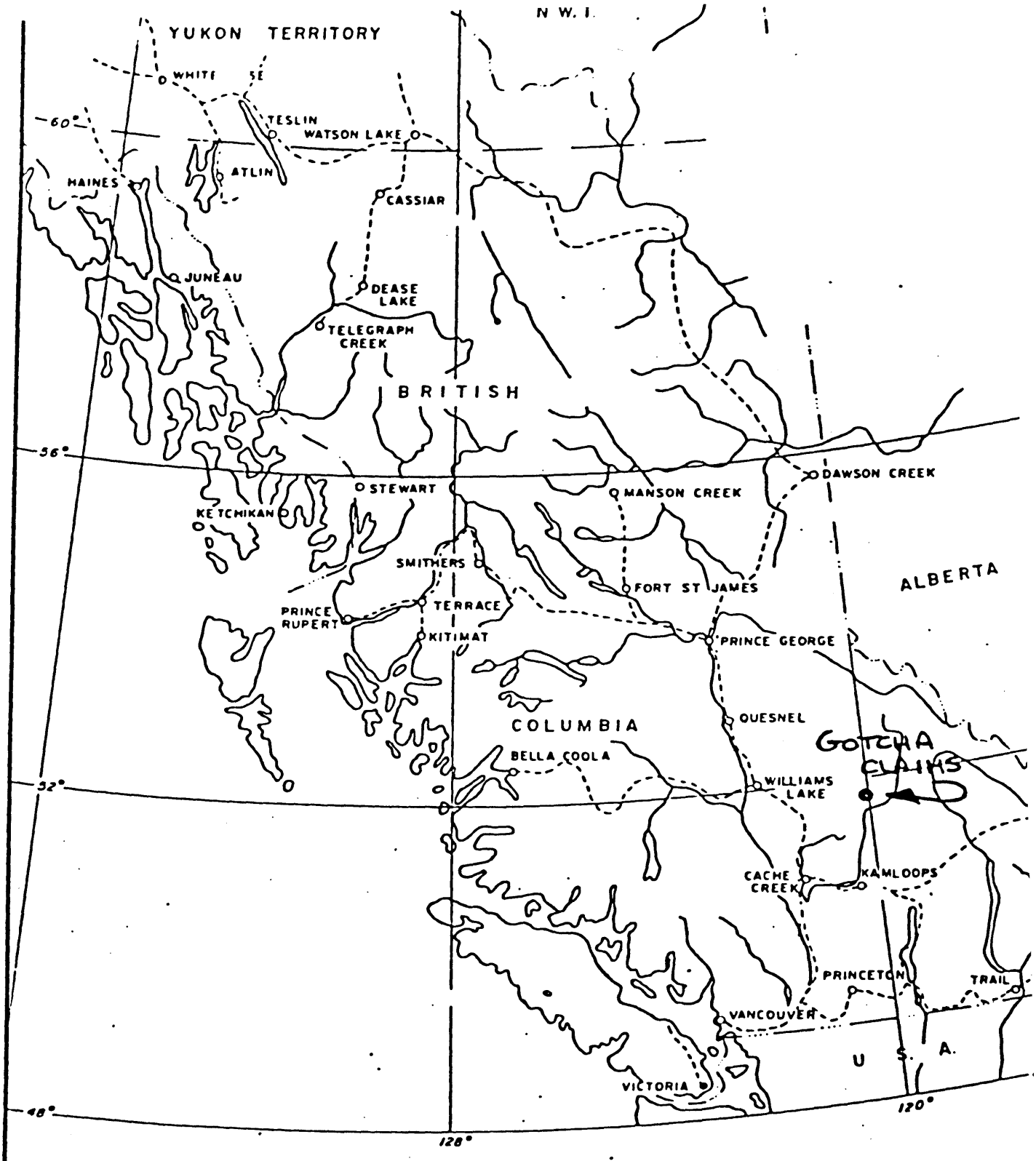
The goals of the physical work were:

1. to allow large exploration equipment and vehicles access to the known scheelite mineralized zones.
2. to expose key sections of the scheelite zones to help expand geological knowledge of the complex property.
3. to test the scheelite mineralization's continuity and dimensions.
4. to produce a representative sample of scheelite mineralized material for metallurgical and mill testing.

Physical work was undertaken during July and August 1978. The full program is continuing; laboratory metallurgical testing has just been completed and sample material is being transported to a rejuvenated flotation mill located at Lumby, B. C. Start-up of the mill-test is scheduled during the latter part of September. Pollution studies have been completed at the mill-test site and Permit No. MX-138 was obtained pursuant to Section 11 of the Mines Regulation Act regarding reclamation at the physical work site.

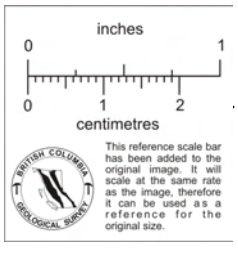
A full report of the entire program from road building through mill test will be compiled and presented prior to February 15, 1979 to the Ministry of Mines and Petroleum Resources for its evaluation and data base.

However, it is hoped that this interim report will meet the requirements of Contract 16, Mineral Exploration Incentive Program and the Maximum MEIP Committment of $25,000 will be forwarded to United Mineral Services Ltd.



UNITED MINERAL SERVICES LTD.

GENERAL LOCATION SKETCH



SCALE: 1" = 125 MILES

FIGURE 1

CLAIMS

The physical work was undertaken on the Gotcha Tungsten Property which consist of the following claims:

CLAIM	RECORD NO:	NO: OF UNITS
GOTCHA	881	1
GOTCHAZ	786	9
MAX 1	1145	1
MAX 2	1146	1

The claims are located 20 miles northeast of Clearwater, B. C. on the west side of upper Maxwell Creek. The approximate geographic centre of the claims is Lat. 51°50'N, Long. 119°42'W. (See Figure 1).

The claims are plotted on Mineral Claim Map 82M/13E.

WORK PROGRAM

A. General

Physical work completed to August 31, 1978 was contracted to and completed by Tymac Construction Ltd. which is headed by Brian McClay. Geological control to the program was provided at intervals on the property by M. McClaren and R. Dickinson of United Mineral Services Ltd.

Other contractors providing services to the project were:

1. Colin Blair (Clearwater)
2. Buck Logging Ltd. (Clearwater)
3. P. Miller (Clearwater)
4. Pashco Rock Contractors Ltd. (Kamloops)

5. Vardax Consultants Inc. (Yarrow)
6. Wadlegger Logging and Construction Ltd. (Clearwater)

The work consisted of three stages;

1. Road Building
2. Stripping
3. Trenching - drilling, blasting, mucking.

each of which will be described separately.

B. Road Building

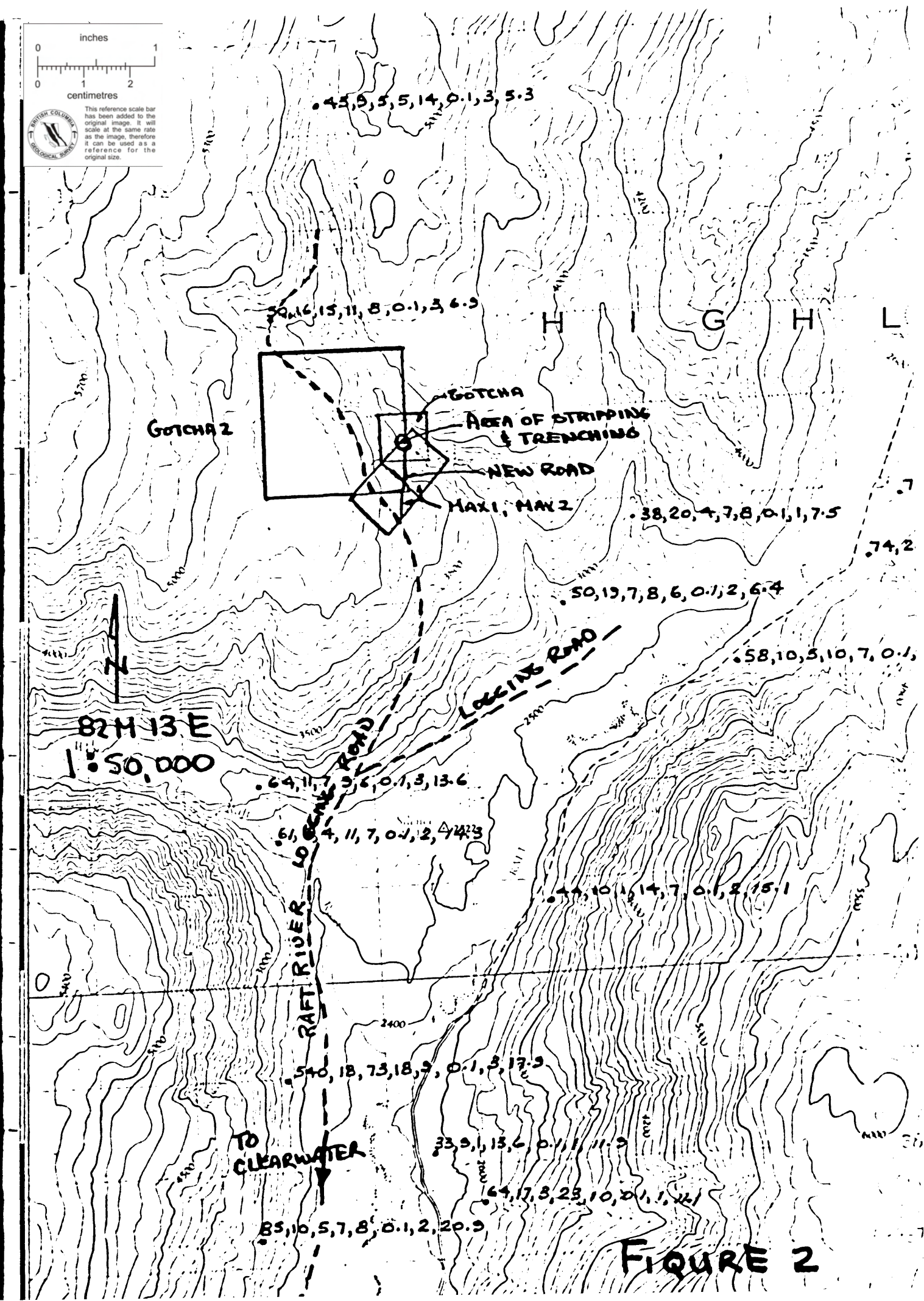
A 660 metre long by 8 metre wide road was constructed by D8 Cat. The road leads northerly off the east side of the Maxwell Creek logging road down to the main showings near the valley bottom. (See Figure 2.) The road was ditched and in three locations culverted using 24" and 18" galvanized culverts.

The D8-Cat was accessed to and from the property by Lowboy.

The road allows large equipment access to the main minerlaized zone.

C. Stripping

An area of 1456 square metres was stripped of overburden which averaged 3.5 metres in thickness (See Figure 3). Total volume of overburden stripped to bedrock was 5096 cubic metres. Initial stripping was done by D8 Cat with clean up done by D6 Cat and Insly 560 Excavator.


inches
0
1
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2
centimetres
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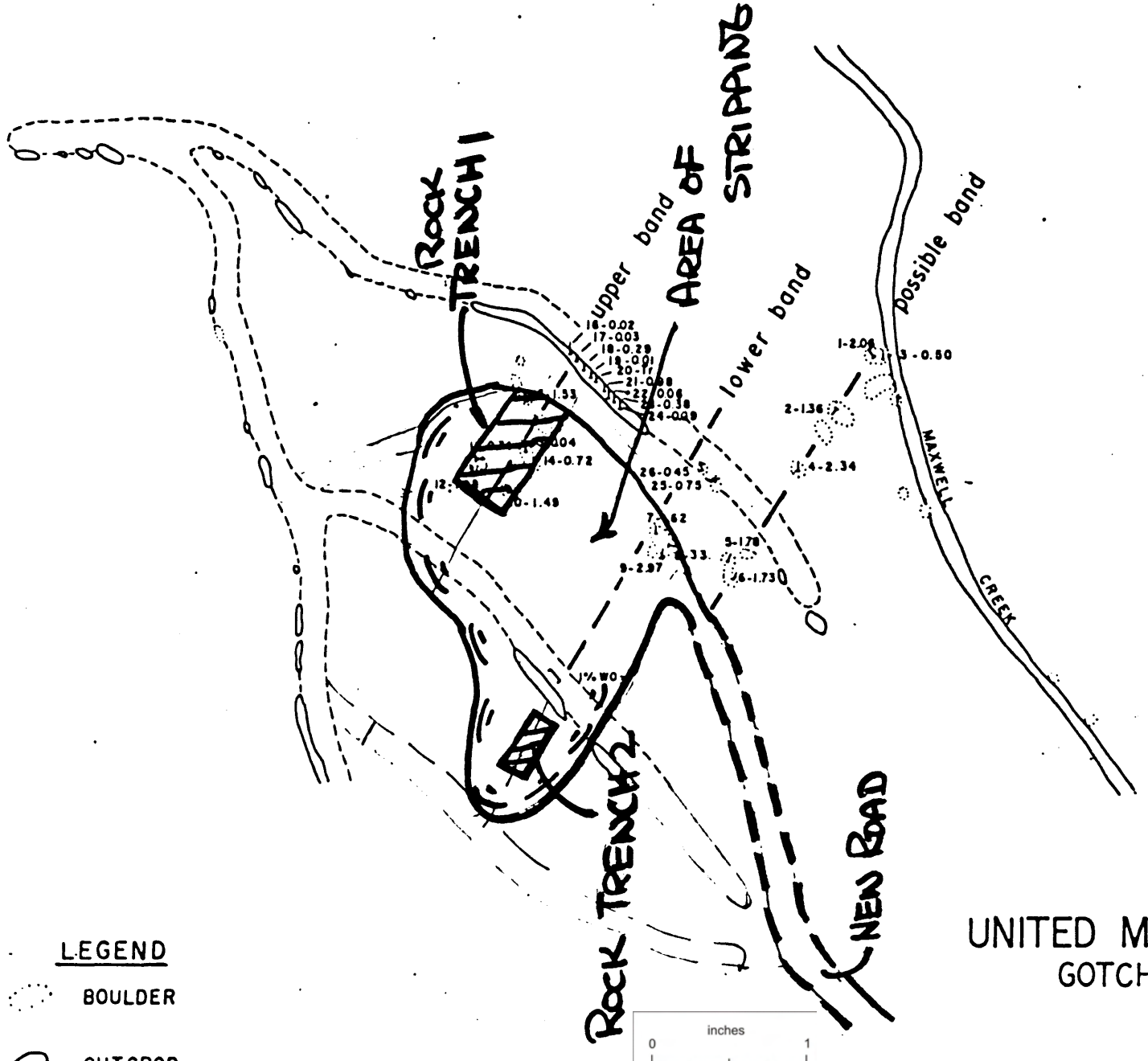
45,8,5,5,14,0.1,3,5.3
50,16,15,11,8,0.1,3,6.9
H I G H L
GOTCHA
AREA OF STRIPPING & TRENCHING
NEW ROAD
MAX1, MAX2
GOTCHA 2
38,20,4,7,8,0.1,1,7.5
74,2
50,19,7,8,6,0.1,2,6.4
58,10,5,10,7,0.1,
LOGGING ROAD
N
82M 13E
1:50,000
64,11,7,3,6,0.1,3,13.6
61,4,11,7,0.1,2,14.3
44,10,14,7,0.1,2,15.1
RAFT RIVER
2400
540,18,73,18,9,0.1,3,17.9
TO CLEARWATER
33,9,1,13,6,0.1,1,11.9
64,17,8,23,10,0.1,1,22.1
85,16,5,7,8,0.1,2,20.9
FIGURE 2

D. Trenching

Rock drilling was accomplished using air track and compressor. Drilling was on a 1.2 metre x 1.2 metre pattern. Holes were loaded with POWER FRAC which was electrically detonated. The resultant average rock size obtained was approximately .2 metres.

Two separate areas were trenched. (See Figure 3.) Trench 1 has dimensions 20 metres long, 8 metres deep and 11 metres wide or a volume of 1760 cubic metres. Trench 2 was smaller with dimensions 10 metres long, 10 metres deep and 5 metres wide or a volume of 500 cubic metres.

The trenches were mucked with a Insly 560 Excavator and D6 Cat.



LEGEND

BOULDER

OUTCROP

SHORT PERCUSSION HOLE

(AFTER: D.L. COOK)

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.

FIGURE 3

UNITED MINERAL SERVICES LTD.

GOTCHA 2 MINERAL CLAIM

KAMLOOPS M.D.

CLEARWATER, B.C.

SCALE

FEET 100 75 50 25 0 100 200 FEET

EXPENDITURE STATEMENT

ROAD BUILDING	
660 metres @ $8.43/m	5,565.00
Culverts	750.00
STRIPPING	
5096 m3 @ $11/m3	56,056.00
ROCK TRENCHING	
TRENCH 1 - 1760 m3 @ $10/m3	17,600.00
TRENCH 2 - 500 m3 @ $10/m3	5,000.00
TOTAL COST	$83,915.00

QUALIFICATION AND CERTIFICATION

I, Robert A. Dickinson, with permanent address at 1395 Ottawa Avenue, West Vancouver, B. C., declare:

1. That I graduated from the University of British Columbia with a B.Sc. degree in honours geology in 1972 and a M.Sc. degree in Business Administration in 1974.

2. That since graduation I have been employed as an exploration geologist in British Columbia, Saskatchewan, Territories, and Washington, U.S.A.

3. That I am president and exploration geologist of United Mineral Services Ltd., whose office is located at 1326 - 510 West Hastings Street, Vancouver, B. C. V6B 1L8.

4. That I have an interest in the Gotcha Property.

5. That the data of this report is based on data collected by me on the property and by presentations of contractors to me during the months July and August, 1978.

6. That to the best of my knowledge and belief the Expenditure Statement is correct.

VANCOUVER, B. C.
August 31, 1978

R. A. DICKINSON, President
UNITED MINERAL SERVICES LTD.

QUALIFICATION AND CERTIFICATION

I, Murray McClaren, with permanent address at O'Byrne Road, R.R.#3, Sardis, B. C. declare:

1. That I graduated from the University of British Columbia with a B.Sc. degree in geology in 1971.

2. That since graduation I have been employed as an exploration geologist in British Columbia, Northwest Territories and Washington State, U.S.A.

3. That I am vice-president and exploration geologist of United Mineral Services Ltd., whose office is located at 1326 - 510 West Hastings Street, Vancouver, B. C. V6B 1L8.

4. That I have an interest in the Gotcha Property.

5. That the data of this report is based on data collected by me on the property and by presentations of contractors to me during the months July and August, 1978.

6. That to the best of my knowledge and belief the Expenditure Statement is correct.

VANCOUVER, B. C.
August 31, 1978

M. MCCLAREN
UNITED MINERAL SERVICES LTD.

RESULTS

The Program to August 31, 1978 has been highly successful and underbudget. The road produced is stable and allows access to the core of the Property. Rock exposed by stripping has shown the strata to be severely faulted and folded. It now appears possible that earlier drilling may not have limited the tonnage potential as thought earlier. Murray McClaren will be mapping the property during September. At present, plans call for survey control.

Scheelite mineralization is found to be relatively continuous for skarn type deposits. Grades are typically 1% WO_3 or greater in 2 separate skarn bands at surface.

From mineralization exposed at surface it is believed a representative 'ore' sample has been collected for mill testing.

APPENDIX A

In an effort to keep the Applied Geology Section up to date with the full program and recent developments a report titled "Soil Investigation, Proposed Expansion of Tailings Pond, Lumby, B. C." by Golden Associates and recent laboratory testing results from Bacon, Donaldson and Associates are enclosed with this report.

Golder Associates
CONSULTING GEOTECHNICAL ENGINEERS

SOIL INVESTIGATION
PROPOSED EXPANSION OF TAILINGS POND

LUMBY BRITISH COLUMBIA

Distribution:

3 copies - United Mineral Services Ltd.
Vancouver, British Columbia

2 copies - Golder Geotechnical Consultants Ltd.
Kelowna, British Columbia

B78663

July, 1978

Golder Associates
CONSULTING GEOTECHNICAL ENGINEERS

July 30, 1978

United Mineral Services Ltd.
1326 - 510 W. Hastings Street
Vancouver, B.C.

Attention: Mr. M. McClaren

Re: Soil Investigation
Proposed Expansion of Tailings Pond
Lumby, B.C.

Dear Sir:

Further to your request, Golder Associates have carried out a geotechnical investigation for the above project. The purpose of the investigation was to determine the soil and ground water conditions at the site in order to make geotechnical recommendations concerning the design and construction of the proposed expansion of tailings storage facilities.

Four penetration test holes were put down to depths of 8 to 18 ft. Records of the test holes and the results of laboratory testing of representative samples are enclosed. The locations of the penetration test holes are shown on Figure 2. The field work was carried out on July 11 and 12, 1978.

SITE AND SOIL CONDITIONS

The site is located on the flood plain of Bessett Creek along the toe of the western slope of Saddle Mountain in Lumby, B.C. as shown on Figure 1.

The general site slopes gently to the west towards the creek and is covered with grass. An artesian pond was observed at the toe of the existing dyke in the southwest corner of the tailings pond area. A well installed in the pond supplies fresh water to the mill.

Based on the results of the penetration tests the dykes surrounding the proposed pond consist of a silty sand with some gravel. It appears that some compaction of the dyke fill materials was obtained during the construction as the fills were encountered to be in a compact state of relative density. The dyke slopes are standing at angles varying between 1½ to 1, and 2 to 1 (horizontal to vertical). The height of the dykes along the west and north perimeter varies between 7 and 9 ft.

Based on the results of the investigation it is inferred that the soil stratigraphy shown in Figure 3 and discussed below is typical of the flood plain in the general site area. The soil deposits in order of increasing depth are as follows:

Depth (Ft.)	Strata Description
0 - 8	Compact medium to fine sand
8 - 14	Compact sand with some fine gravel
14 +	Firm to stiff clayey silt

The ground surface in the general pond area is underlain by compact deposits of fine to medium sand overlying a relatively permeable layer of sand with fine gravel. The relative density of the latter stratum varies between compact and dense with refusal being encountered at penetration test P4 in this stratum. In penetration test P1 the granular stratum is underlain by

clayey silt sediment. The ground water level was encountered in penetration test P1 at a depth of 8 ft. below the ground surface with coincides approximately with the water level in the artesian pond.

PROPOSED DEVELOPMENT

It is understood that the existing mine facilities will be utilized to process imported tungsten ore. At present it is understood that about 500 to 600 tons of ore will be imported which will require about 400 to 500 cu. yds of additional tailings storage. If economically feasible, an additional 2500 tons of ore may be processed.

It is understood the existing pond will not be utilized as the toxic level of the reagents used for flotation are substantially different. Previously the tailings contained high arsenic concentrations and the reagents used were cyanide and xonthate which are highly toxic. The tailings water will be recycled during milling process and the proposed reagents contain mainly sodium silicate, soda ash and Pamak C4, a distilled fatty acid which would not be expected to be toxic in modest concentrations. The results of the spectrographic analysis and the flotation testing of the ore are given in Appendix 1 attached.

The tailings were previously being deposited into the southern pond measuring 75 by 75 ft. and enclosed by dykes with a crest elevation varying between 110 and 112 ft. It is understood that the pond was lined using a 10 mil P.V.C. liner placed to about elevation 108 ft. The dykes were topped with some 2 to 3 ft. of silty sand and gravel fill during a later stage in the tailings disposal.

A storage area for future tailings disposal was constructed north of the

existing pond. This pond measuring 94 by 128 ft. has been constructed to elevation 105 ft. with locally available material obtained mainly from inside the pond area. Details of the existing facilities are shown on the photograph in Figure 6.

PROPOSED TAILINGS DISPOSAL

Based on the results of the investigation the tailings disposal area is underlain by granular strata which are relatively permeable and the ground water table is encountered at shallow depth which is typical of the floodplain topography. At present we estimate that the capacity of the pond is about 3600 cu. yds. If the tailings were placed in this pond area adequate storage capacity is available. Based on the soil conditions, it is recommended that if the pond area is used an impermeable liner be provided to prevent seepage. For cost purposes we have considered using only a portion of the pond area or construct a new pond for tailings disposal to limit the amount of liner required. The two alternatives for construction of the pond are shown in Figure 4 and 5. These lined ponds would also be used for retention of tailings reagents for recycling to the mill. Consideration could be given to excavate the tailings sand from the ponds to provide additional capacity for tailings disposal in the second stage program. The capacity of the ponds and the amount of fill required for construction are given on Table 1. The dykes should be constructed using locally available materials at slopes not greater than 2 horizontal to 1 vertical on the exterior and 1.5 horizontal to 1 vertical internally. The material should be placed in maximum 1 ft. lifts and compacted to a minimum density of 95 per cent Standard Proctor Density. Based on the results of the penetration tests the existing dykes surrounding the proposed tailings disposal area are considered to be constructed adequately when

combined with the above construction.

If additional storage is required, we suggest that the accumulated coarse tailings sands from the northern portion of the proposed pond area be excavated and placed in the adjacent existing pond area. If these materials are excavated during a dewatered pond period, the moisture content is likely to be less than 10 to 15 per cent. As the moisture content is about 20 to 25 per cent in the saturated tailings, any seepage into the ground water will be minimal. The surface of the tailings should be graded at a minimum slope of 3 horizontal to 1 vertical to facilitate future run off and consideration may be given to placing a 1 ft. layer of silty sandy topsoil on top of the tailings to minimize infiltration of precipitation.

IMPERMEABLE LINERS

We recommend that a 20 mil P.V.C. plastic liner be used in the proposed secondary ponds. Prior to liner installation a bed of at least 6 in. of fine sand should be placed over the area to be covered. The sand surface should be smoothly graded and should be free from sharp rock, metal, wood or other fragments of material that are likely to cause rupture.

It is understood that compacted asphalt lining is being considered. Asphalt would provide a suitable impermeable liner for the ponds. To enable placement of the asphalt and to inhibit long term creep, the side slopes of the dykes should be constructed no steeper than 3 horizontal to 1 vertical. A minimum 2 in. layer of asphalt should be provided and the asphalt should be coated with an asphaltic sealent. A combination of asphalt lining of the bottom of the pond and placement of P.V.C. liner on the side slope may also be considered in order to enable

easy access to the pond area during excavation inside the pond area in the second stage. Alternatively a 6 in. protection layer of pit run sand and gravel with a maximum size of 1 in. can be used.

We trust that this provides the information you require at present. If we can be of further assistance to you, please contact us.

Yours very truly,

GOLDER GEOTECHNICAL CONSULTANTS LTD.

P. Hollier

for R. M. Wilson, P. Eng.

B. Carlsen, P.Eng.

BC/pjc

B78663

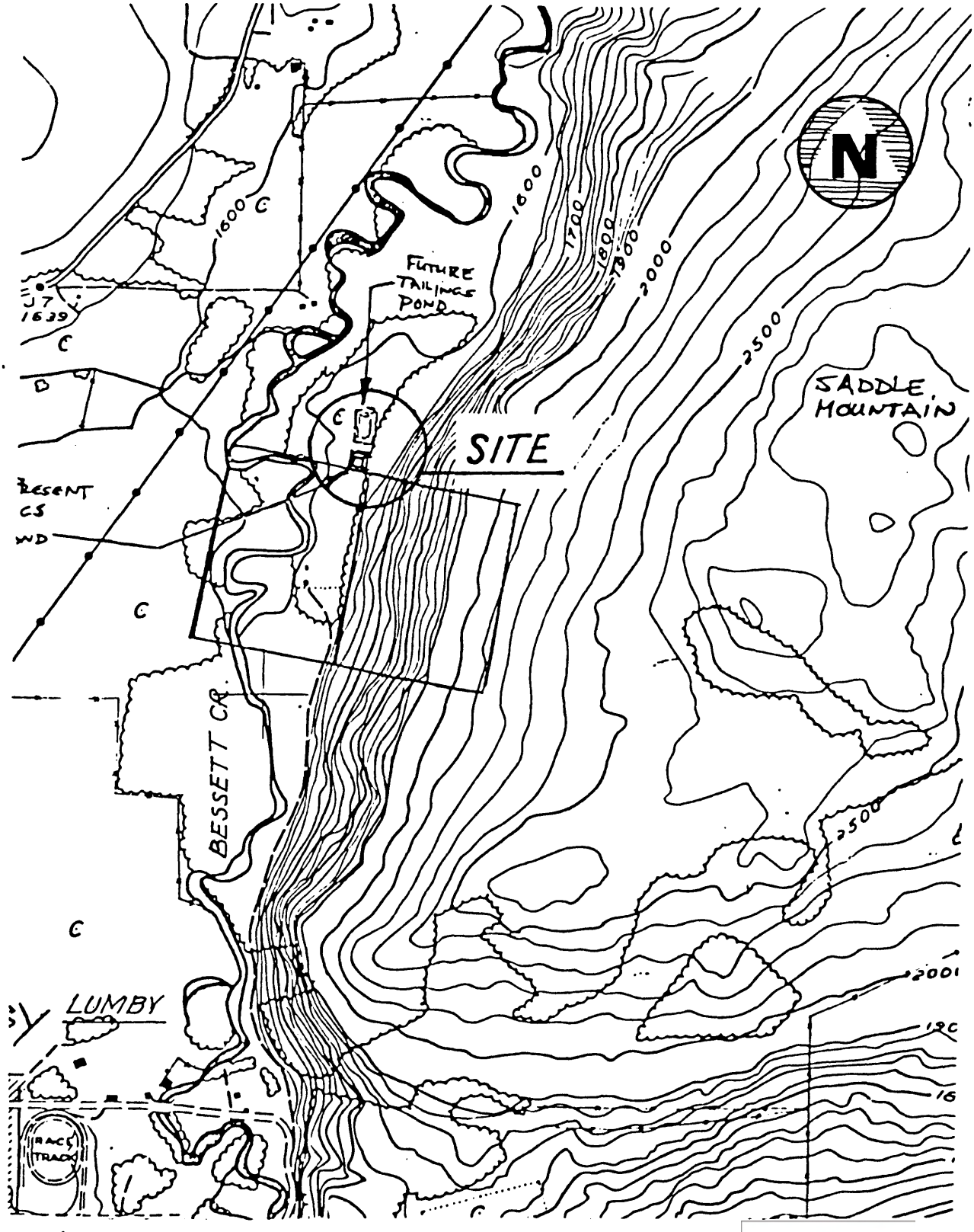
TABLE 1

TAILINGS POND CONSTRUCTION

	Capacity Cu.Yds.	Capacity Tons	Dyke Fill Cu.Yds.	Surface Area To Be Lined (Sq.Ft.) Slope	Surface Area To Be Lined (Sq.Ft.) Bottom
Alternative I	480	570	450	2700	1600
Alternative II	900	1100	900	5000	1900
Existing Northern Stage Area	3600	4300	-	7700	12000

KEY PLAN

Figure 1



SCALE 1 in. to 1000 Ft.

inches 0 1

0 1 2 centimetres

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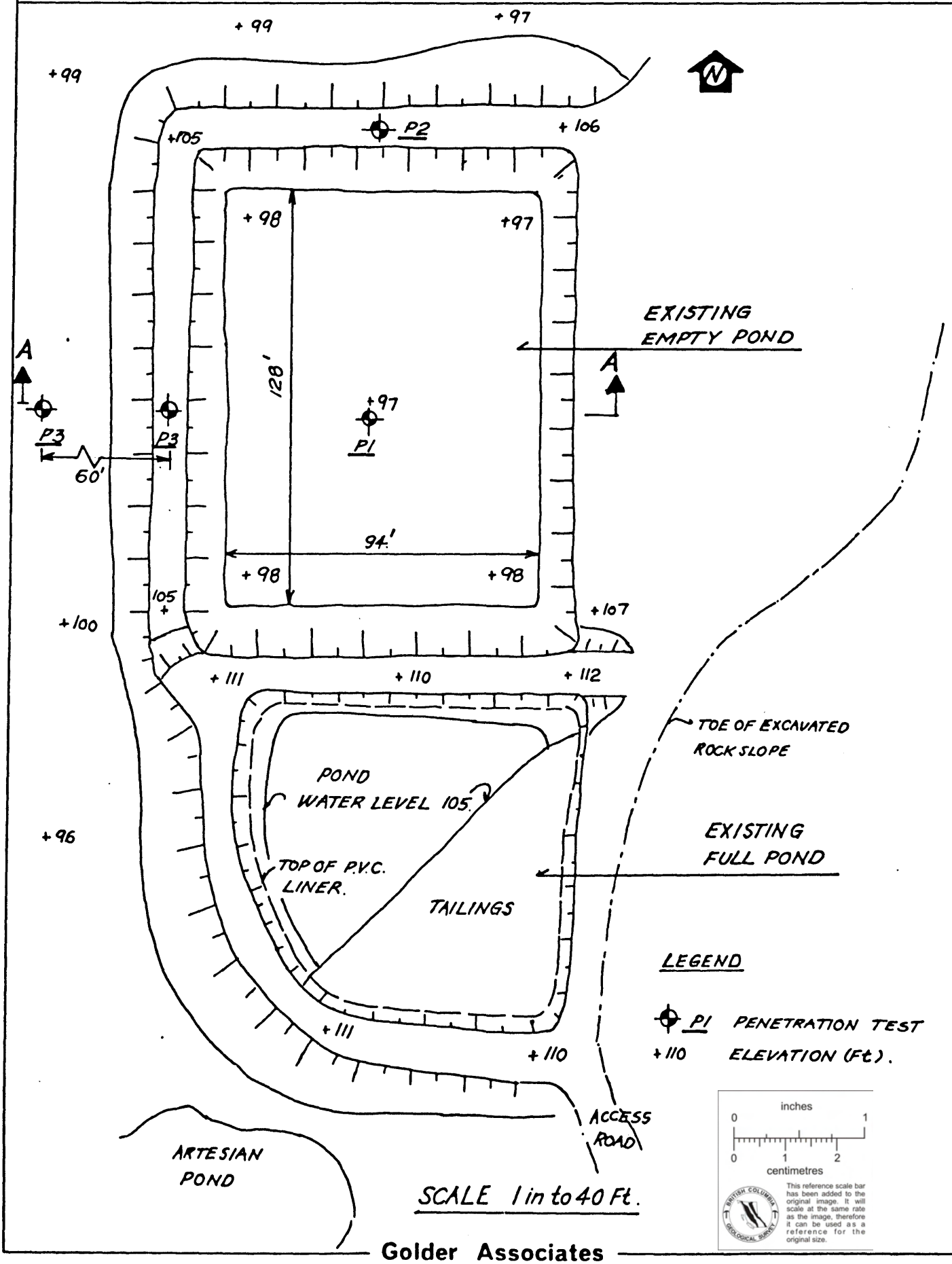
SITE PLAN

Figure 2



Project No. 878663 Drawn PR Reviewed BC Date July 25/78.

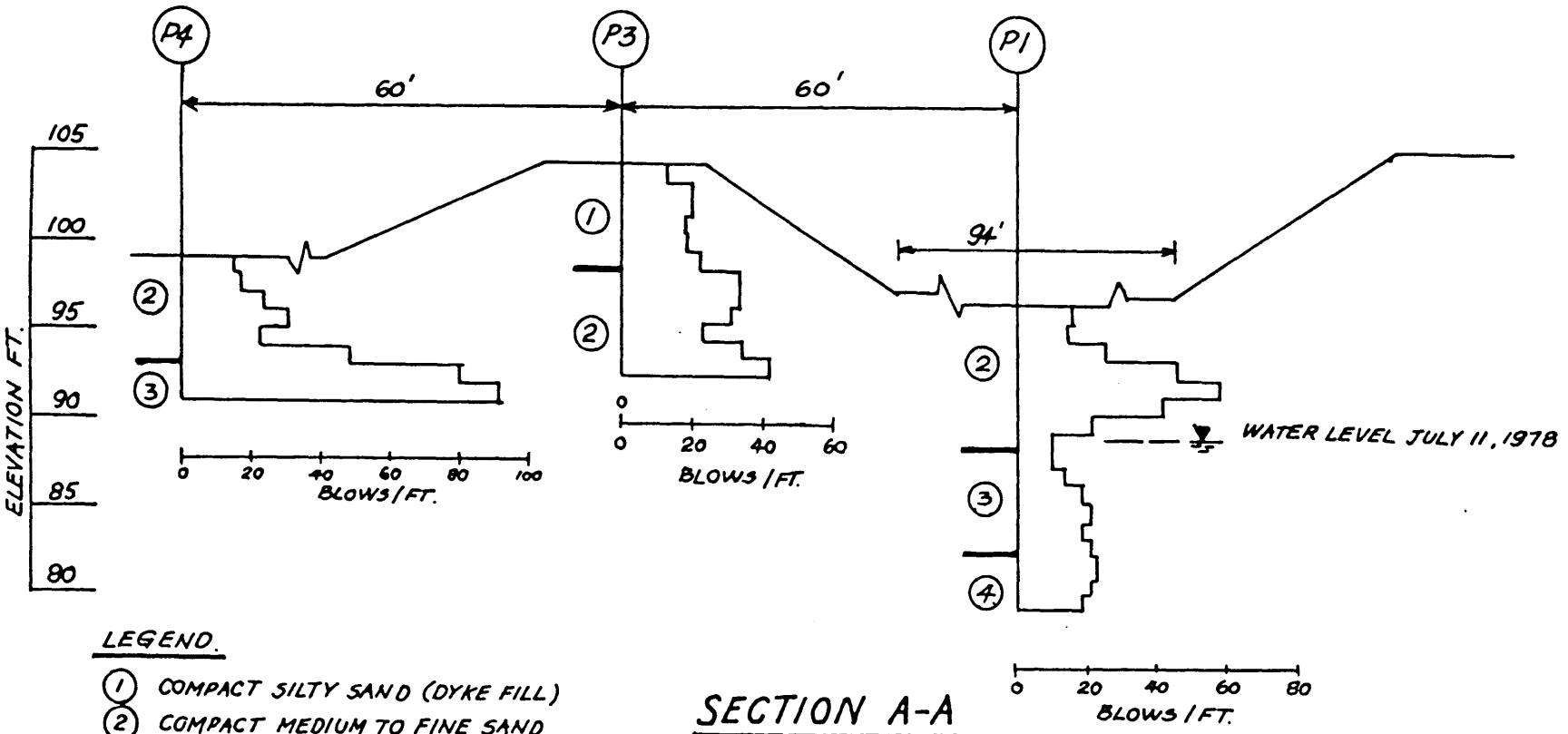
Golder Associates

Project No 878663 Drawn RH Reviewed ZC Date JULY 25/78.

Golder Associates

INFERRED SOIL STRATIGRAPHY

Figure 3



LEGEND.

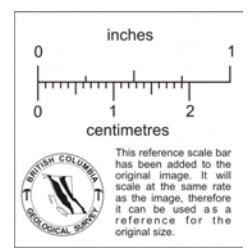
(1) COMPACT SILTY SAND (DYKE FILL)
(2) COMPACT MEDIUM TO FINE SAND.
(3) COMPACT SAND WITH SOME FINE GRravel
(4) FIRM CLAYEY SILT.

(P1) PENETRATION TEST.

FOR LOCATION OF SECTION SEE FIG. 2.

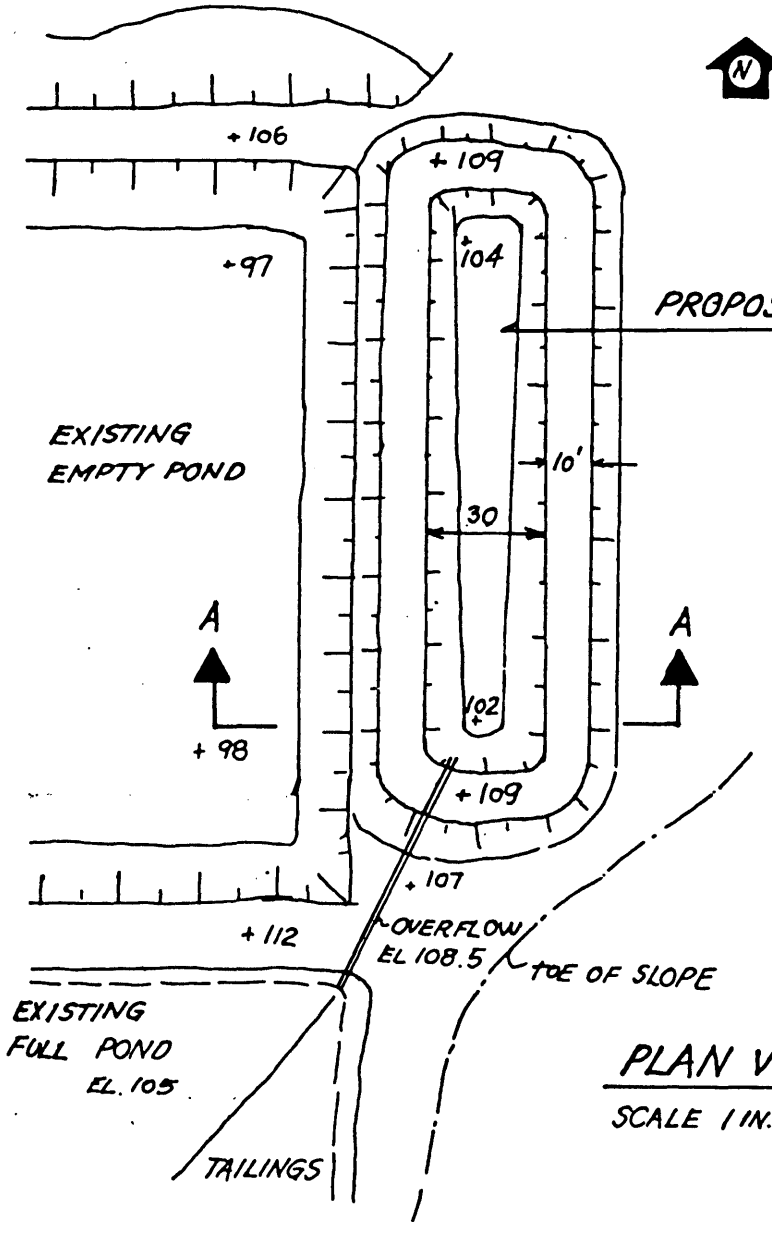
SECTION A-A

SCALE 1 in to 10 Ft.



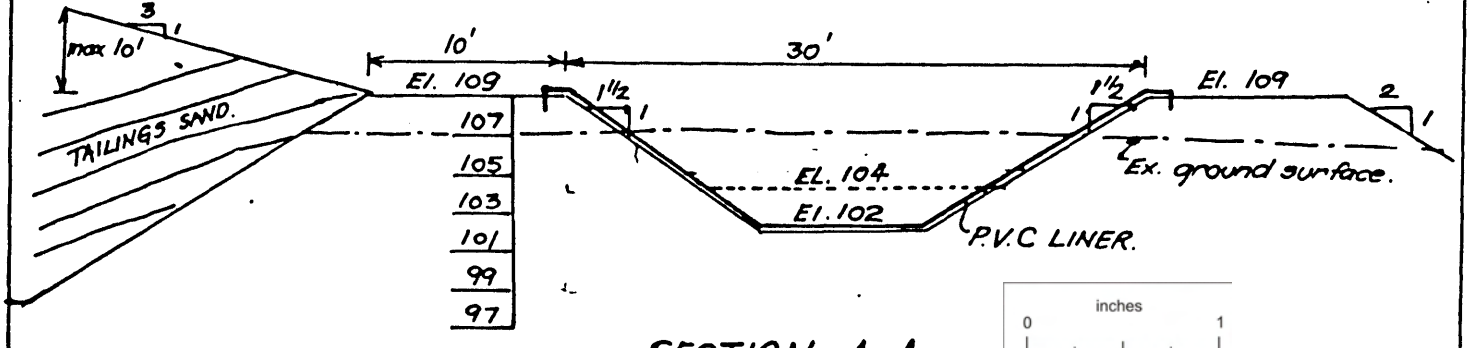
TAILINGS DISPOSAL - ALTERNATIVE I

Figure 4.



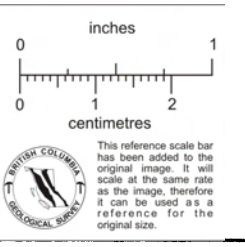
PLAN VIEW

SCALE 1 IN. TO 40 FT



SECTION A-A

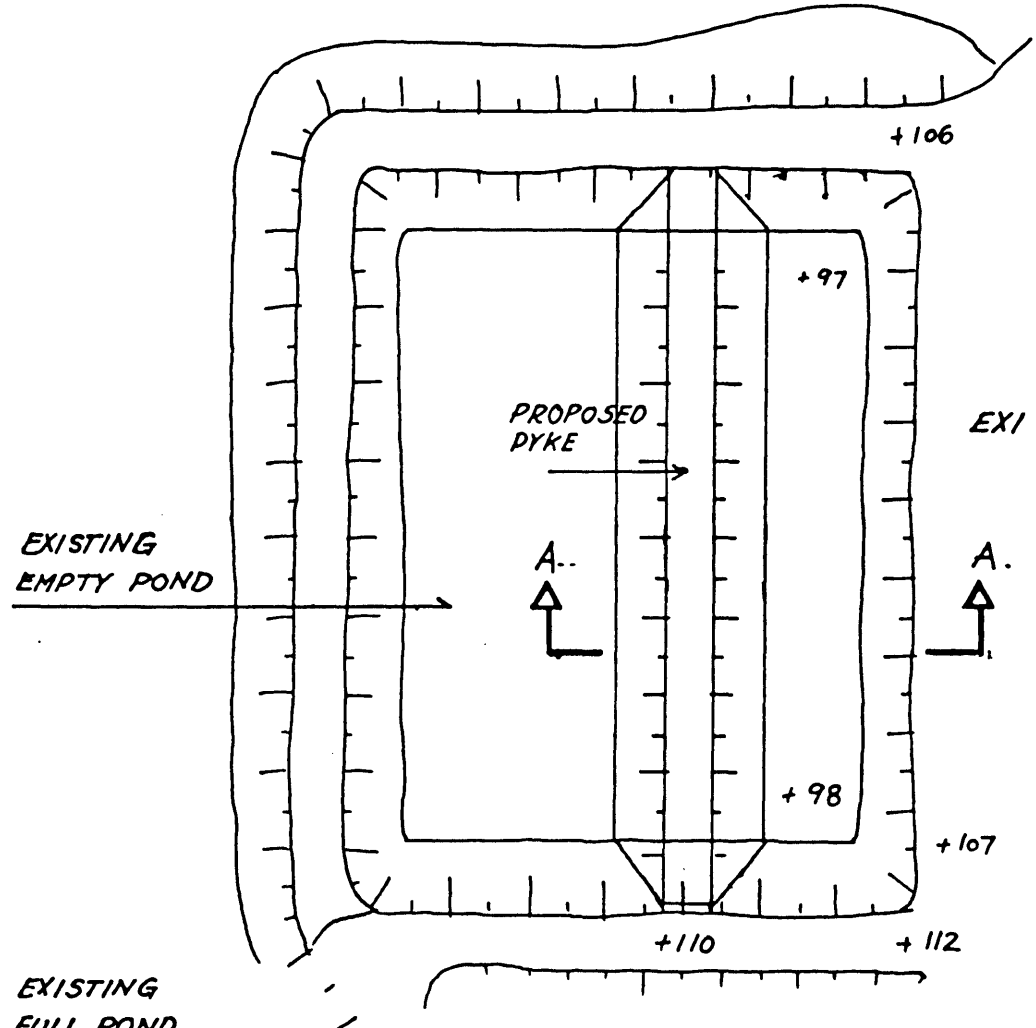
SCALE 1 IN TO 10 FT.



Project No. 878663 Drawn RH Revised RC Date JULY 26/78

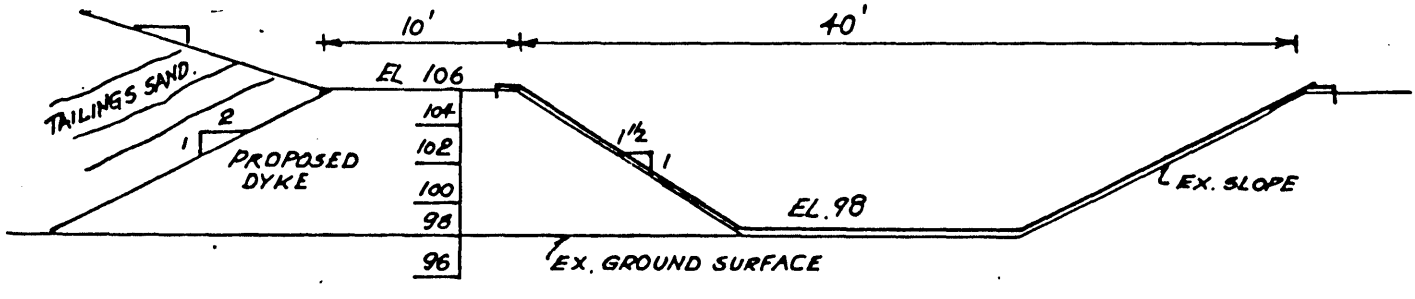
TAILINGS DISPOSAL - ALTERNATIVE II

Figure 5.



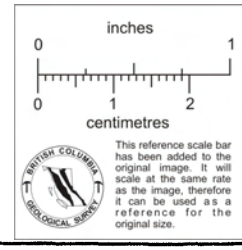
PLAN VIEW

SCALE 1 IN. TO 40 FT.



SECTION A-A

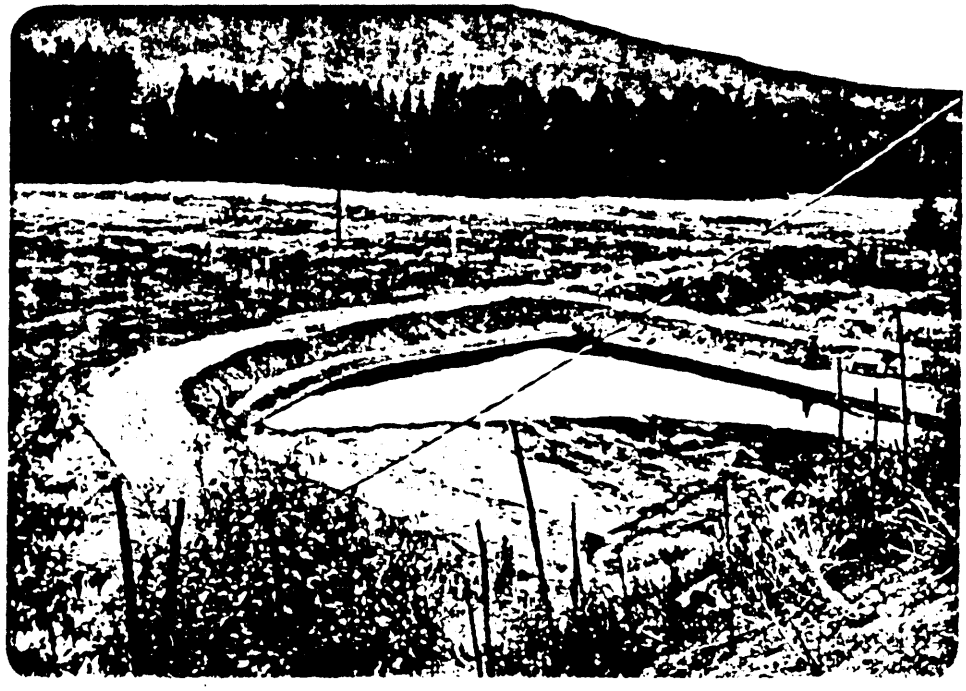
SCALE 1 IN TO 10 FT.



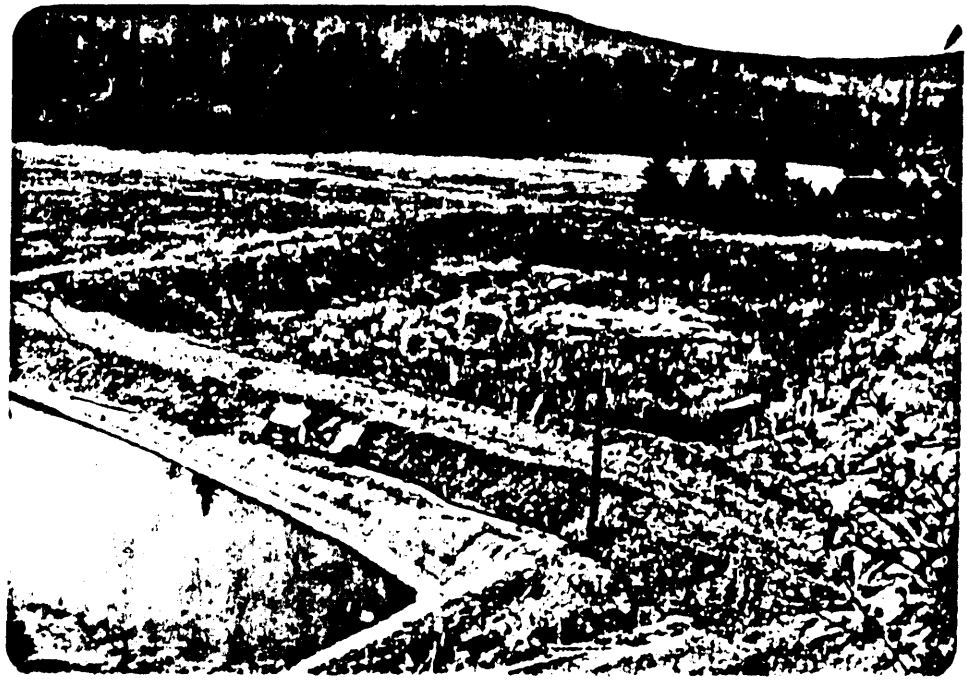
Project No. 878663 Drawn FH Reviewed JC Date AUG. 3/78.

SITE PHOTOGRAPHS.

Figure 6.



EXISTING TAILINGS POND.

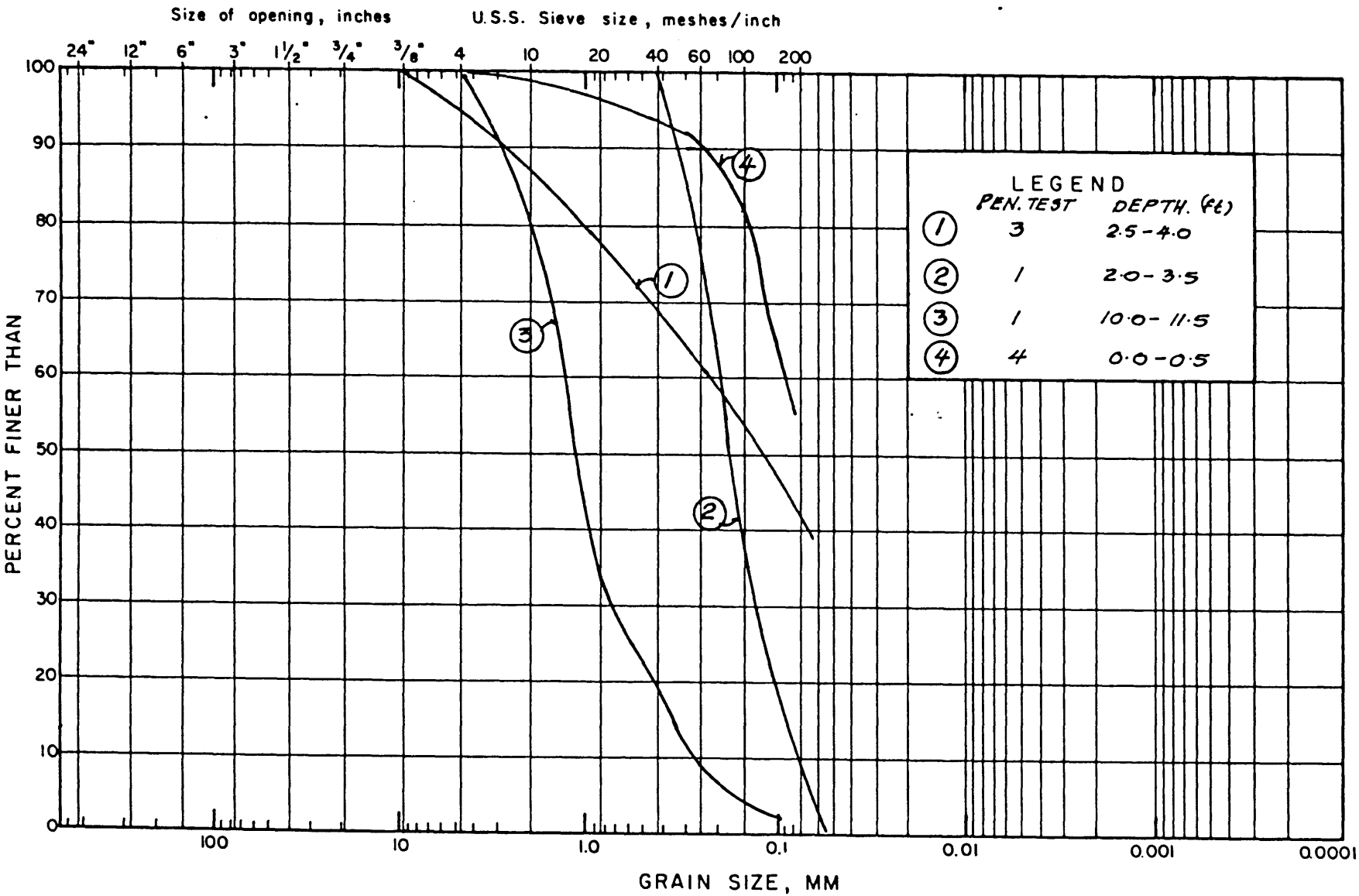


PROPOSED TAILINGS DISPOSAL

878663 FH AUG. 7/78

Project No. B78663 Drawn PH Reviewed BC Date JULY 25/78

M.I.T. GRAIN SIZE SCALE



BOULDER	COBBLE SIZE	COARSE	MEDIUM	FINE	COARSE	MEDIUM	FINE	SILT SIZE	CLAY SIZE
		GRAVEL SIZE			SAND SIZE			FINE GRAINED	

GRAIN SIZE DISTRIBUTION

Figure 7

Golder Associates

Project No. _____

PENETRATION TEST P1

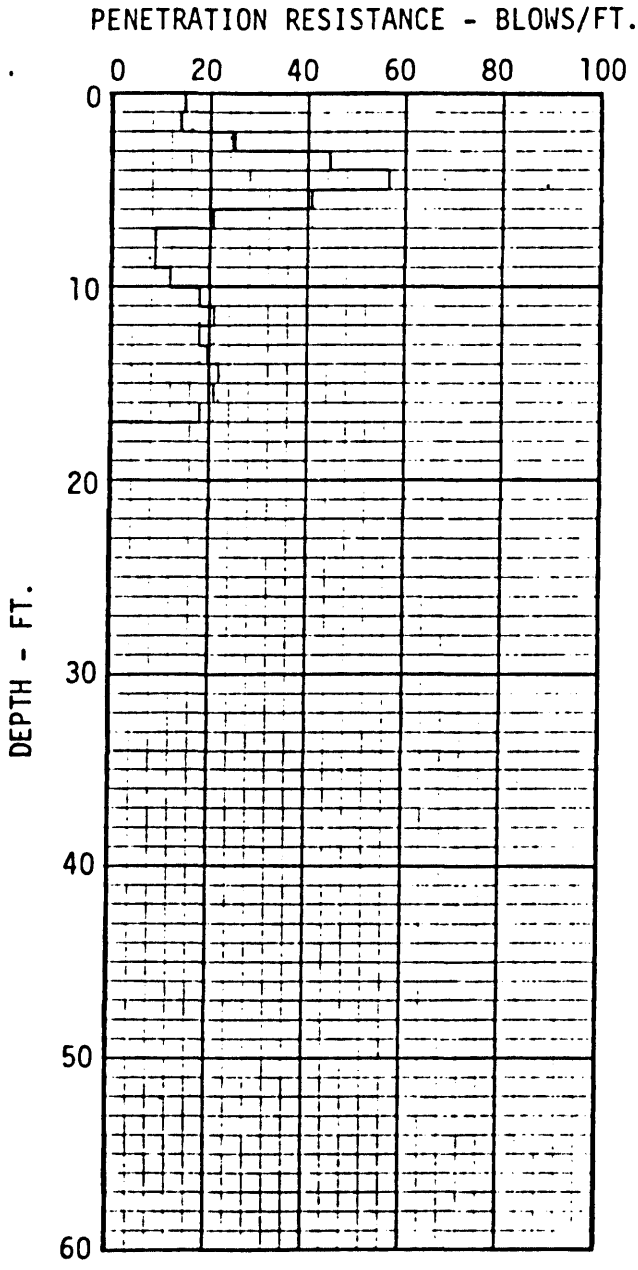
Location: (see figure 2)
Surface elev:
Hammer weight: 50 lbs.
Shoe diameter: 2 ins. (o.d.)

Date: July 11, 1978
Datum:
Hammer drop: 24 ins.
Rod diameter: 1 ins. (o.d.)

PENETRATION TEST

ELEV. / DEPTH

INFERRED SOIL PROFILE



Ground Surface

Compact Medium to Fine Sand

Compact Sand with Fine Gravel

Firm to Stiff Grey Clayey Silt

End of Penetration Test

G.B. 105

Golder Associates

Drawn: _____
Reviewed: _____

Project No. _____

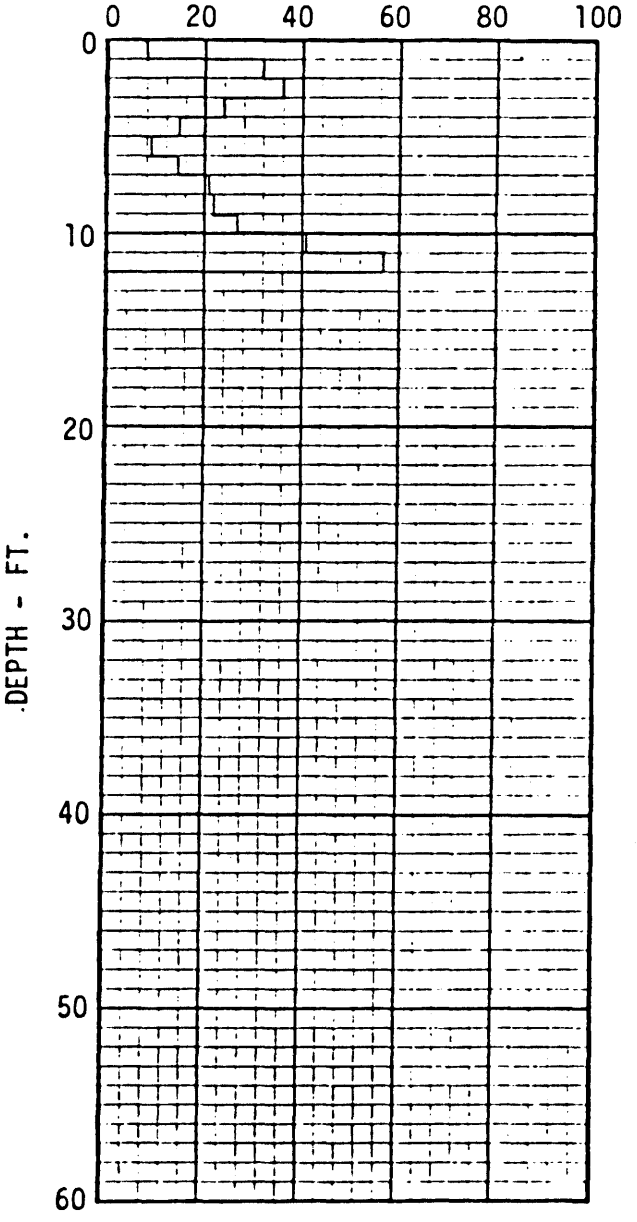
PENETRATION TEST P2

Location: (see figure 2)
Surface elev:
Hammer weight: 50 lbs.
Shoe diameter: 2 ins. (o.d.)

Date: July 12, 1978
Datum:
Hammer drop: 24 ins.
Rod diameter: 1 ins. (o.d.)

PENETRATION TEST

PENETRATION RESISTANCE - BLOWS/FT.



ELEV. / DEPTH

INFERRED SOIL PROFILE

Crest of North Dyke

Compact silty sand with some gravel (Dyke Fill)

Compact Medium to Fine Sand

End of Penetration Test

Drawn: _____
Reviewed: _____

Golder Associates

G.B. 105

Project No. _____

PENETRATION TEST P3

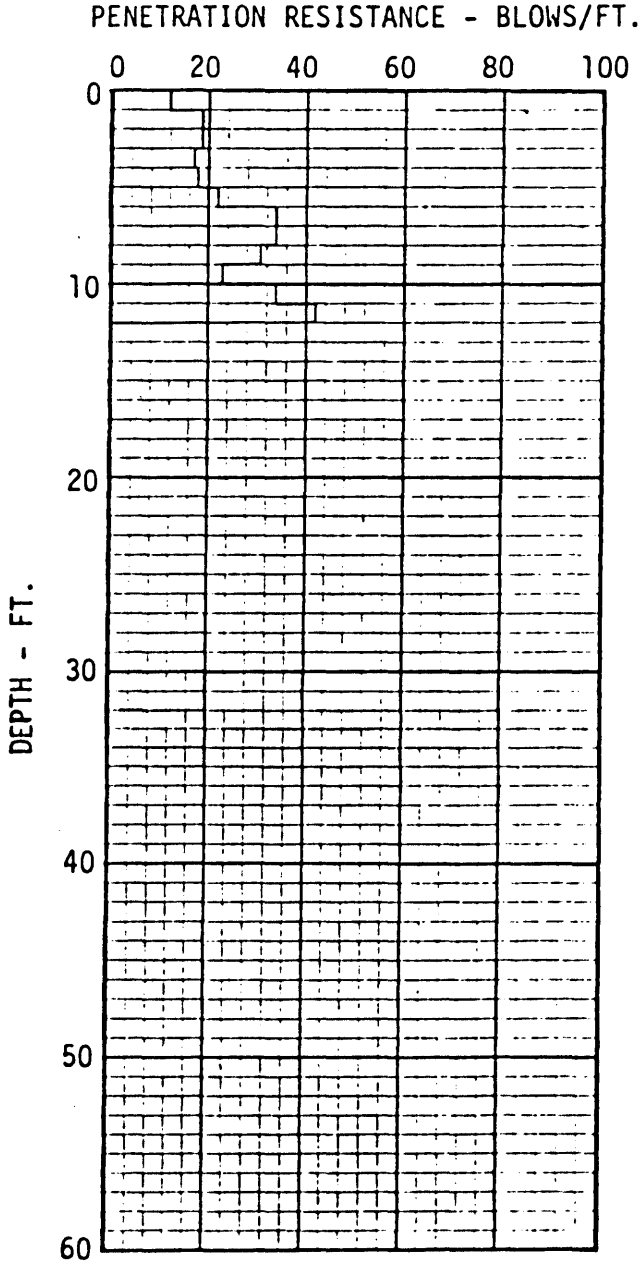
Location: (see figure 2)
Surface elev:
Hammer weight: 50 lbs.
Shoe diameter: 2 ins. (o.d.)

Date: July 12, 1978
Datum:
Hammer drop: 24 ins.
Rod diameter: 1 ins. (o.d.)

PENETRATION TEST

ELEV. / DEPTH

INFERRED SOIL PROFILE



Crest of West Dyke

Compact Silty Sand (Dyke Fill)

Compact Medium to Fine Sand

End of Penetration Test

Drawn: _____

Golder Associates

Reviewed: _____

G.B. 105

Project No. ----------

PENETRATION TEST P4

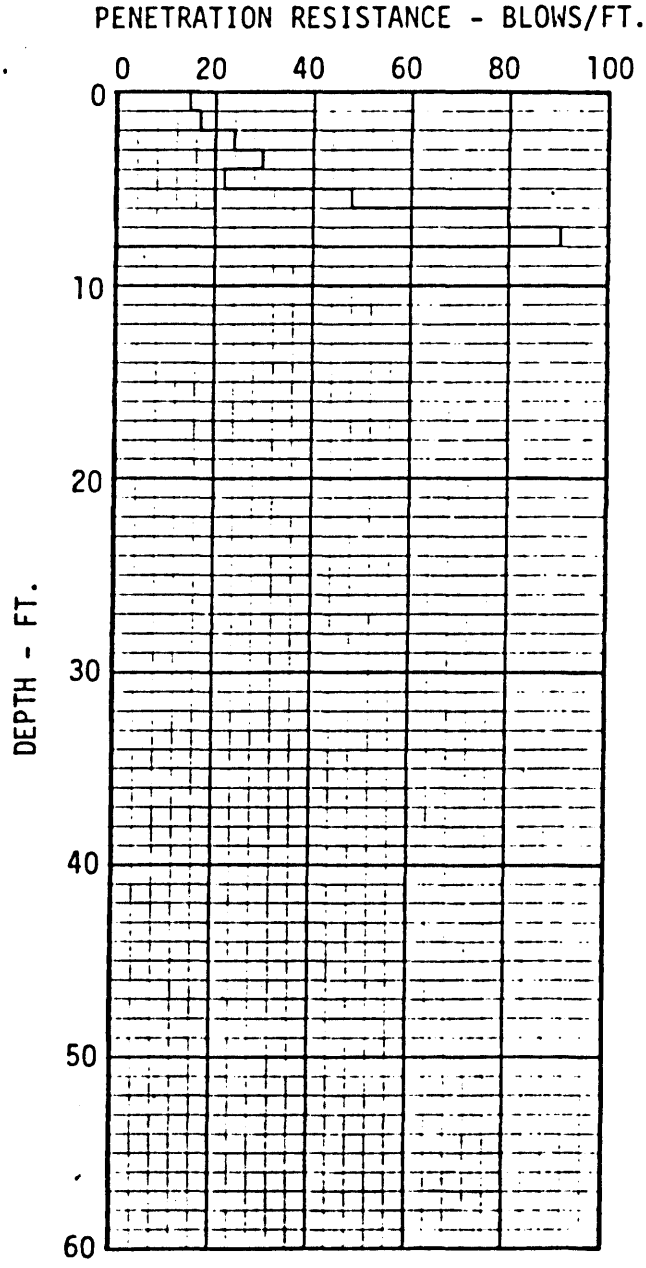
Location: (see figure 2)
Surface elev:
Hammer weight: 50 lbs.
Shoe diameter: 2 ins. (o.d.)

Date: July 12, 1978
Datum:
Hammer drop: 24 ins.
Rod diameter: 1 ins. (o.d.)

PENETRATION TEST

ELEV. / DEPTH

INFERRED SOIL PROFILE



Ground Surface

Compact Medium to Fine Sand

Compact to Medium Dense Sand and Gravel

End of Penetration Test

G.B. 105

Golder Associates

Drawn:

Reviewed:

APPENDIX I

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

DEPARTMENT OF MINES AND PETROLEUM RESOURCES
VICTORIA

SAMPLE RECEIVED FROM E. W. GROVE

ADDRESS Geological Division

SEMI QUANTITATIVE SPECTROGRAPHIC ANALYSIS

boratory No.	19689M
bmitter'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
Trace:	Bi,Ga,Mo
As	0.04
Sn	0.02
Be	0.01
W	>1.0

DATE July 5, 1978

Golder Associates
CONSULTING GEOTECHNICAL ENGINEERS

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

Attention: Mr. M. McClaren

INVOICE NO. 7-9

PROJECT NO B78663

DATE August 24, 1978

TO PROFESSIONAL SERVICES RENDERED to July 31, 1978.

Re: Soil Investigation - Proposed Tailing Pond, Lumby, B.C.

Personnel:		
PRINCIPAL		
R. Wilson	2.0 hrs. @ $50.00/hr.	$ 100.00
ASSOCIATE		
H. Hawson	1.0 hr. @ $46.00/hr.	46.00
SENIOR ENGINEER		
B. Carlsen	23.5 hrs. @ $38.22/hr.	898.17
JUNIOR ENGINEER		
D. Negussey	12.0 hrs. @ $23.82/hr.	285.84
SENIOR TECHNICIAN		
D. Holmes	15.5 hrs. @ $25.96/hr.	402.38
		$1,732.39
EXPENSES		
D. Negussey Expense A/C	$42.26	
Kits Cameras	2.30	
Lloyds World Travel Service	39.95	
	$84.51	
5% Handling Charge	4.23	
	$88.74	88.74
TOTAL THIS INVOICE		$1,821.13

NET 30 DAYS

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. 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. 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

INVOICE

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

Invoice No. 2818

File No. 1779

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

Purchase Order No.

Date July 28, 1978

RE: FLOTATION TESTING, POLLUTION INFORMATION

PROFESSIONAL SERVICES	$827.50
COSTS	61.15
	$888.65

GOTCHA PAID Aug 1/78

This is a professional invoice and is due when presented.
1½% per month charged on invoices over 30 days.

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. J. Vreugde, P. Eng.

TEST 1779-3

PROCEDURE

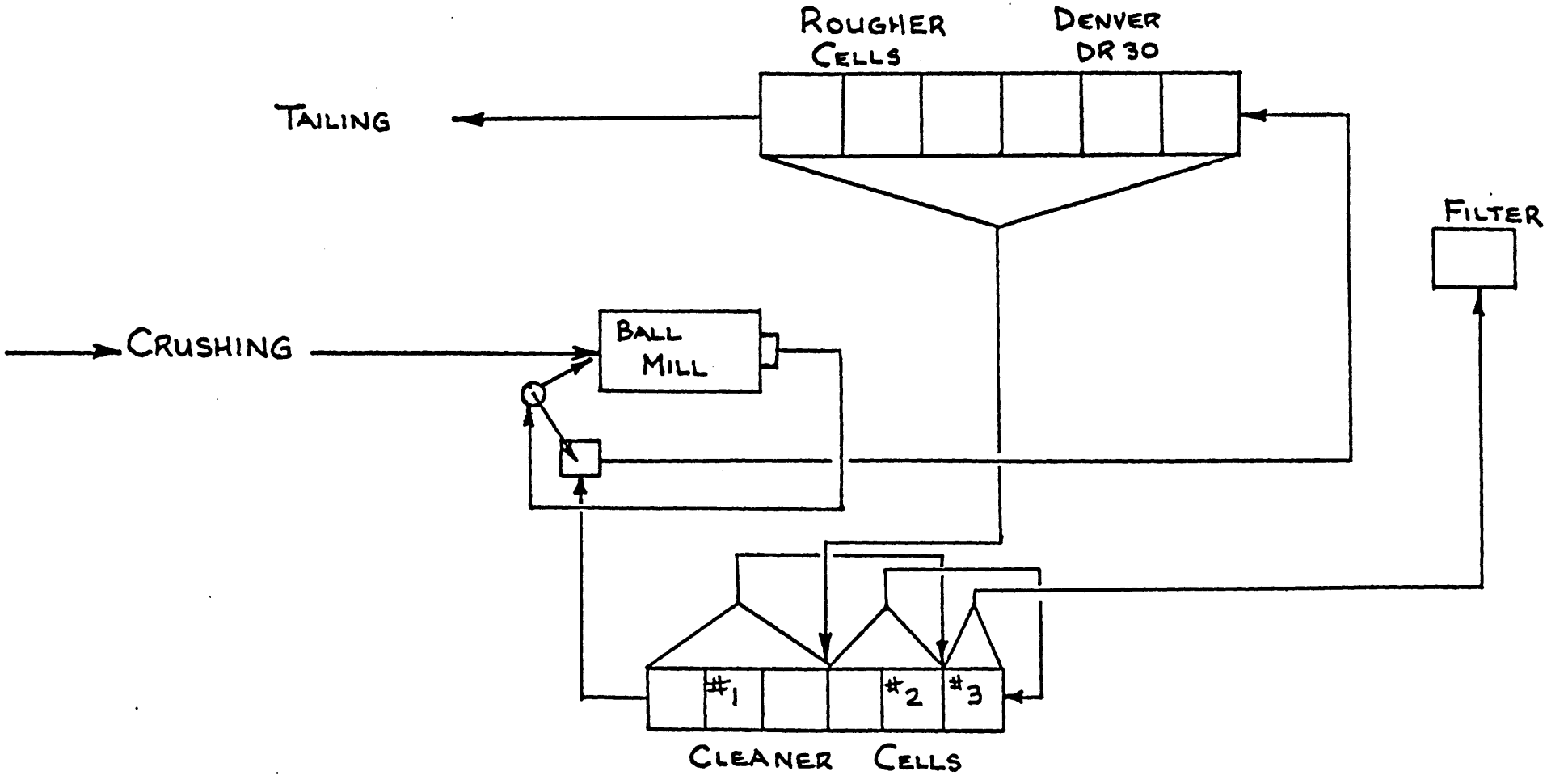
STAGE	TIME (Minutes)	ADDITIONS
Grinding	10	1 lb./ton Soda Ash 0.5 lb./ton Sodium Silicate
Condition	2	0.1 lb/ton PAMAK 4 pH = 9.8
Rougher Float	10	-
1st Cleaner	10	-
2nd Cleaner	6	0.05 lb./ton Sodium Silicate
3rd Cleaner	4.5	0.05 lb./ton Sodium Silicate

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



UNITED MINERALS - PROPOSED TUNGSTEN FLOWSHEET

AUG. 30, 1978

82M 123

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.

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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. J. Vreugde, P. Eng.

82M/123

PROPERTY FILE

PERCUSSION DRILLING RESULTS
AND PRELIMINARY COST STUDY

GOTCHA PROPERTY, CLEARWATER AREA, B. C.

BY

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

March 2, 1978

TABLE OF CONTENTS

	Page
SUMMARY	1A
INTRODUCTION	1
DRILL HOLE ASSAYS	1
ORE RESERVES	3
COST STUDY	6
CONCLUSIONS AND RECOMMENDATIONS	7
CERTIFICATE	9

MAPS

Estimated tonnage for Upper Band

Estimated tonnage for Lower Band

Detail Plan of Mineral Zone

Plan of Trenches, D. D. & Percussion Holes

APPENDIX

A - Bearing, Dip and Length of Percussion Holes

B - Assay Certificates

C - Preliminary Cost Study, dated February 20, 1978.

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.tu WO_3 of which 19,900 s.tu 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.tu 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 proceded 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		% WO_3
	From	To	
1.	20	30	0.03
	30	50	1.58
	50	60	0.85

Hole No.	Footage		% WO_3
	From	To	
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		% WO_3
	From	To	
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 RAUL 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.

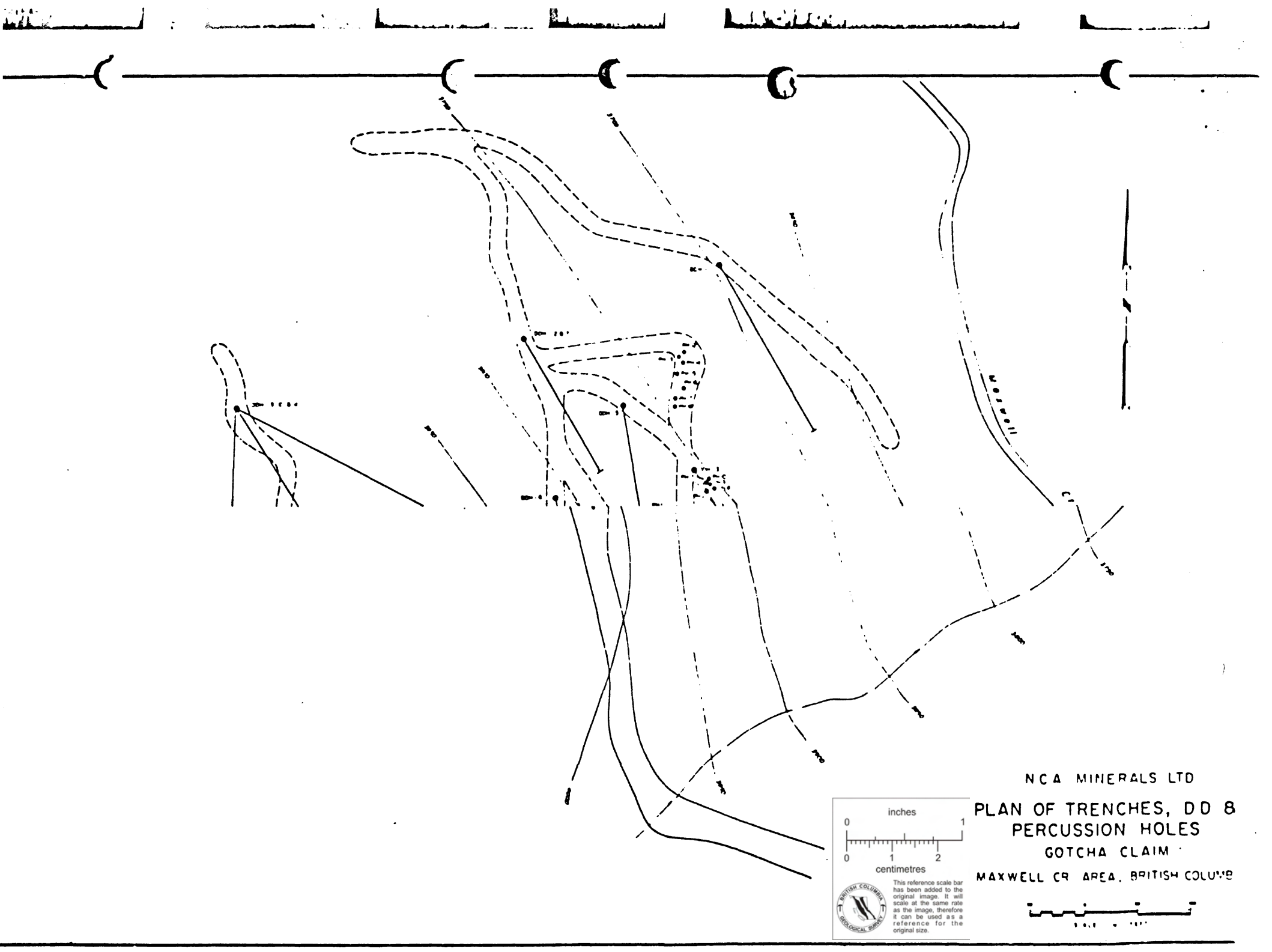
1950 - 1953: Volcan Mines Co., Peru as Mine Superintendant to General Superintendant in 300 ton/day underground lead-zinc mine using cut-and-fill stoping method. Also acting manager of 190 ton/day gold mine and cyanide mill. Mining by room and pillar method.

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.

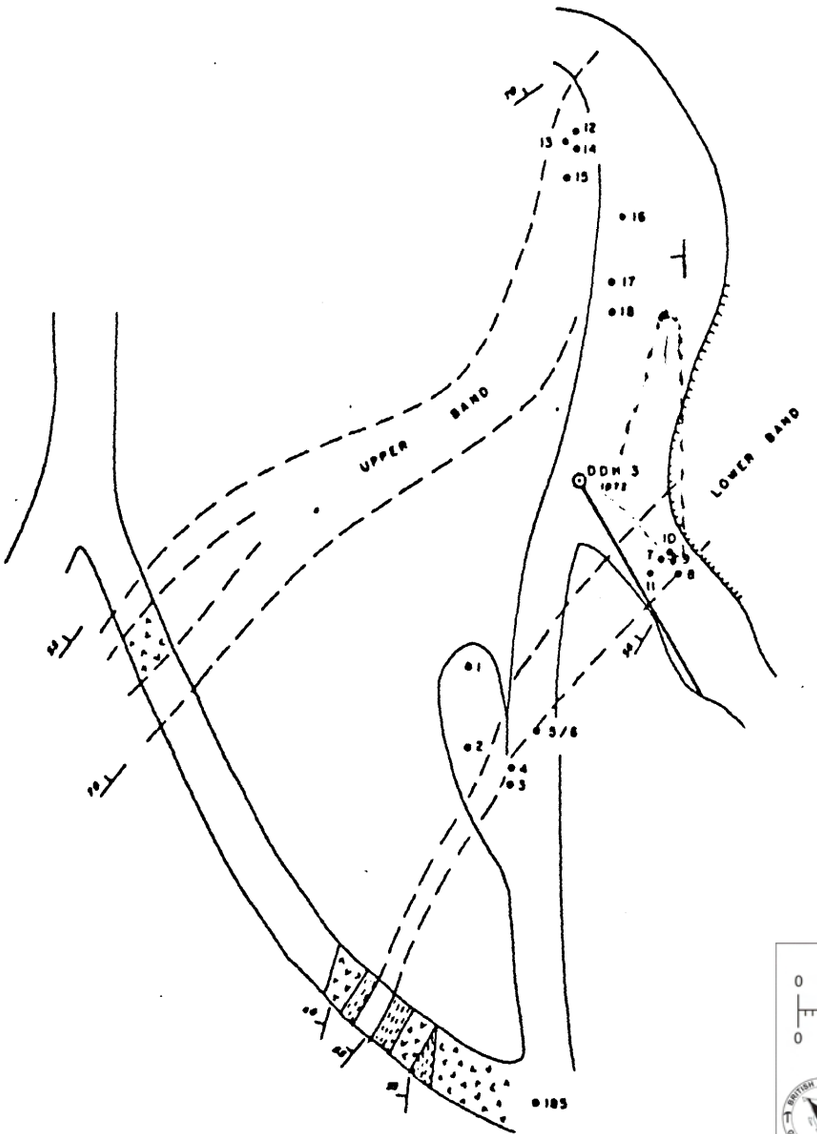


NCA MINERALS LTD

PLAN OF TRENCHES, DD 8 PERCUSSION HOLES

GOTCHA CLAIM

MAXWELL CR AREA, BRITISH COLUMBIA



inches
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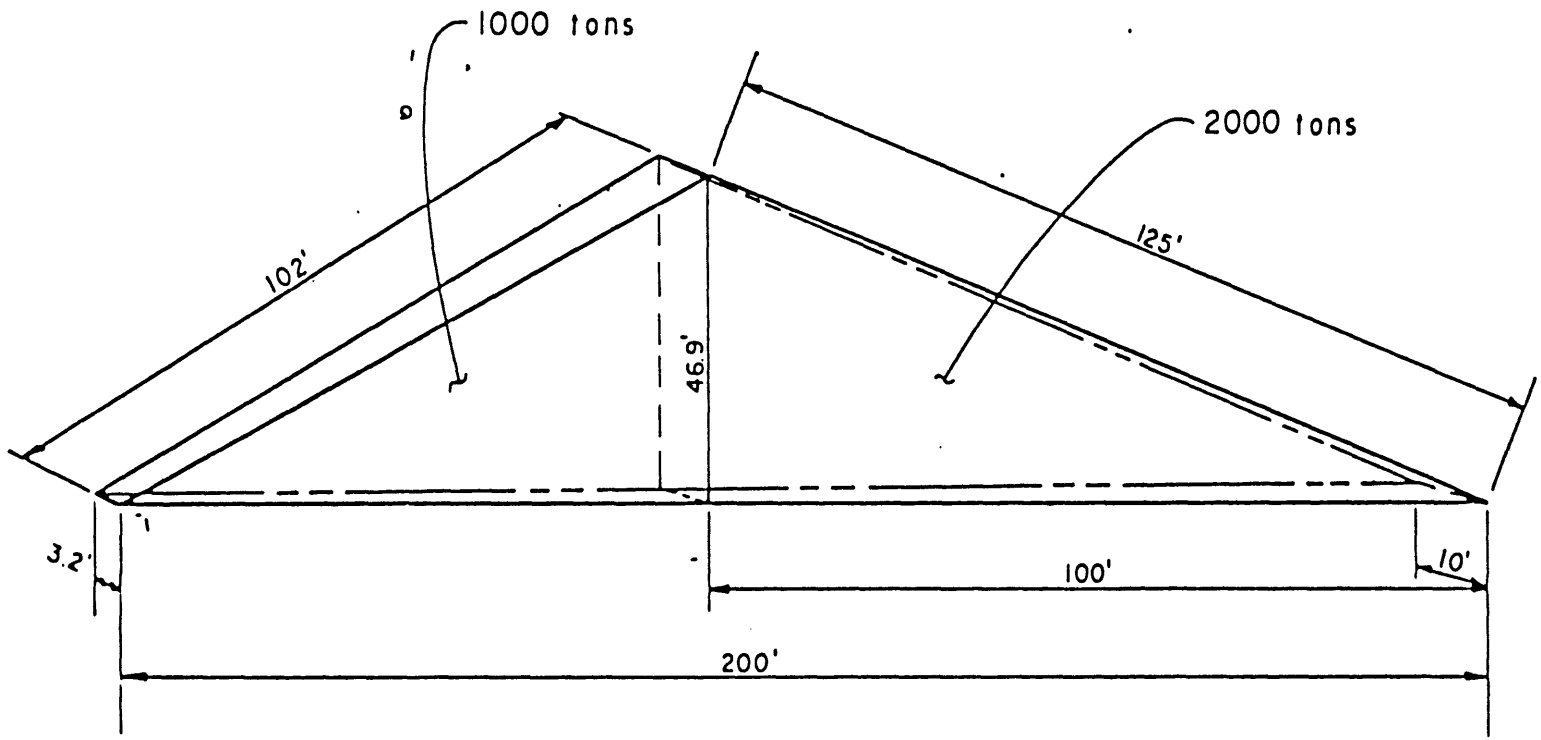
NCA MINERALS LTD

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

SCALE IN FEET

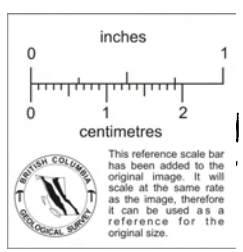


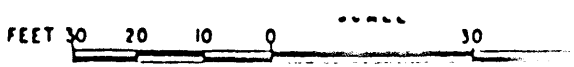
TONNAGE ≃ 3000 tons

NCA MINERALS CORP.

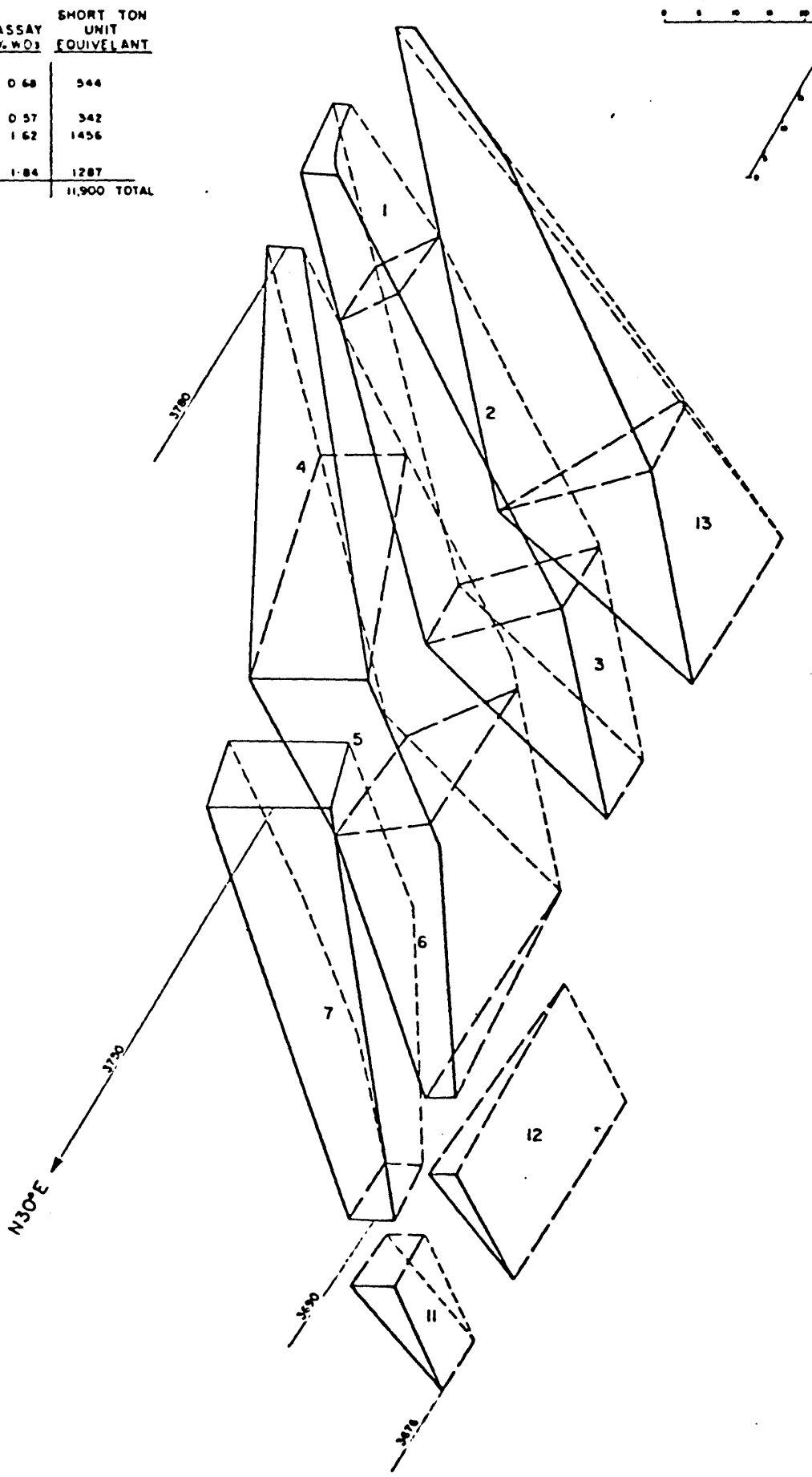
ESTIMATED TONNAGE FOR UPPER BAND GOTCHA MINERAL CLAIM

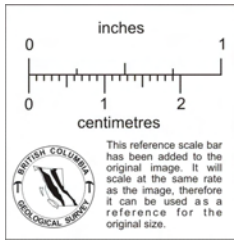
FEBRUARY, 1978





BLOCK	ESTIMATED TONNAGE	CLASS	ASSAY % WO3	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.

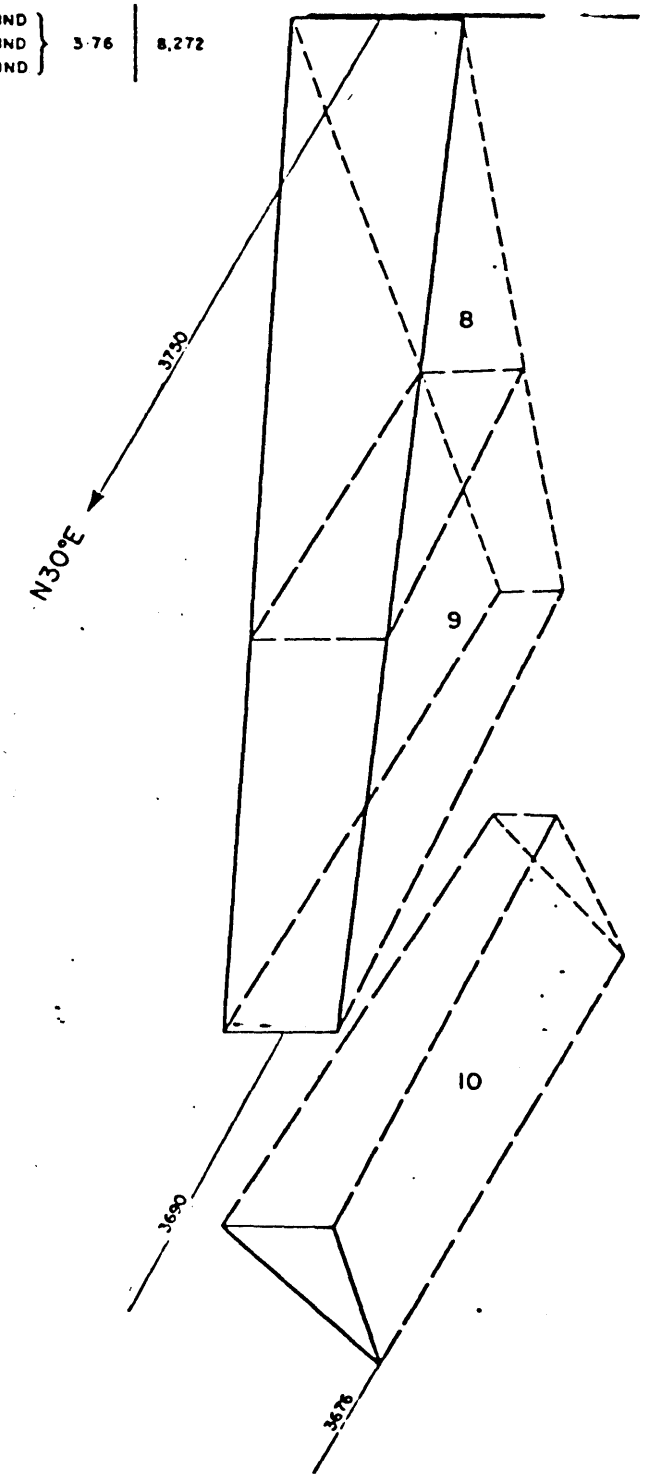
ESTIMATED TONNAGE FOR LOWER BAND

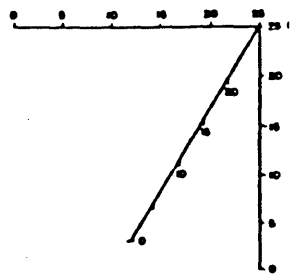
GOTCHA MINERAL CLAIM

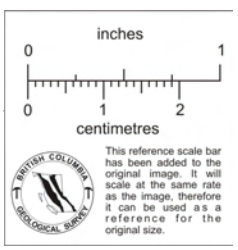
FEBRUARY 1978.

BLOCK	ESTIMATED TONNAGE	CLASS	ASSAY % WO3	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 BAN

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

To:

NCA Minerals Corporation

P.O. Box 371

Vancouver, B.C.

V6C 2N2

Attention: Mr. D. McLeod

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

Telep[illegible] 254-7238
[illegible] 04-54210

Certificate of Assay

File No. 4575C 1 of 3

Date Feb.26/78

Percussion Drill Cores

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

Sample Identification	GOLD Ounces Per Ton	SILVER Ounces Per Ton	Tungsten Percent WO_3					
				Percent	Percent	Percent	Percent	Percent
omposite 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.

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
04 54210

Certificate of Assay

File No. 4575C 2 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 Ounces Per Ton	SILVER Ounces Per Ton	Tungsten 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.

CAN TEST LTD.

Form No. 13-C

Provincial Assayer

NCA Minerals Corporation

P.O. Box 371

Vancouver, B.C.

V6C 2N2

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

Telep[illegible] 254-7278
T[illegible] 04-54210

Certificate of Assay

File No. 4575C 3 of 3

Date Feb.26/78

Attention: Attn: Mr. D. McLeod

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

Sample Identification	GOLD Ounces Per Ton	SILVER Ounces Per Ton	Tungsten Percent WO_3		Specific Gravity			
			Percent	Percent	Percent	Percent	Percent	Percent
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.

CAN TEST LTD.

Form No 13 C

SKS+ —

Provincial Assay

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

(2) Recoverable Value	75% Rec.	85% Rec.
	$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) Mining Costs (100 tons/day)

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

Contract mining costs[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) Milling Costs (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

82M 123

A GEOLOGICAL EVALUATION AND PRELIMINARY ECONOMIC EVALUATION OF THE GOTCHA 2 MINERAL CLAIM

BY

UNITED MINERAL SERVICES LTD.

MAY, 1977

TABLE OF CONTENTS

Page

SUMMARY

GEOLOGICAL EVALUATION OF GOTCHA 2 CLAIM

1. INTRODUCTION 1
2. LOCATION 1
3. PREVIOUS EXPLORATION 1
4. PROPERTY GEOLOGY 1
5. SKARN ASSEMBLAGES AND SCHEELITE MINERALIZATION 2
6. ESTIMATES OF ORE TONNAGE 2
7. EXPLORATION 4
   A. DIAMOND DRILLING 4
   B. FURTHER PROPERTY EXPLORATION 4

PRELIMINARY ECONOMIC EVALUATION OF GOTCHA 2 CLAIM

1. INTRODUCTION 5
2. DISCUSSION OF VARIABLES USED AND COST DEVELOPMENT 5
   A. EXPLORATION EXPENSE 5
   B. MARKETS AND SMELTER PRICES 6
   C. RECOVERY 6
   D. CAPITAL COST 7
   E. MINING COSTS 8
   F. MILLING COSTS 9
   G. TRANSPORTATION COSTS 10
   H. TAXES 12
3. PROFORMA INCOME STATEMENT 1 14
4. PROFORMA INCOME STATEMENT 2 15
5. CONCLUSIONS 16

PLATES

1. General Location Map 1A

FIGURES

1. Geology - Area of Interest

2. Location of Cross Section X-Y

2A. Cross Section X-Y

3. Surface Projection of Upper and Lower Bands

3A. Plan of Geology at 3692' Level

3B. Dimensions of Upper Band

3C. Dimensions of Lower Band - Block 1

3D. Dimensions of Lower Band - Block 2

4. Assays and Visual Estimates of WO_3

5. Geochemical Results and Proposed Drill Hole Locations

APPENDIX

A - Union Carbide Canada Mining Ltd.
Geological and Drill Data

B - Classification of Ore

C - Union Carbide Corporation
Schedule for Purchasing Scheelite Concentrate

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 @ 7.5% WO_3	POSSIBLE ORE

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

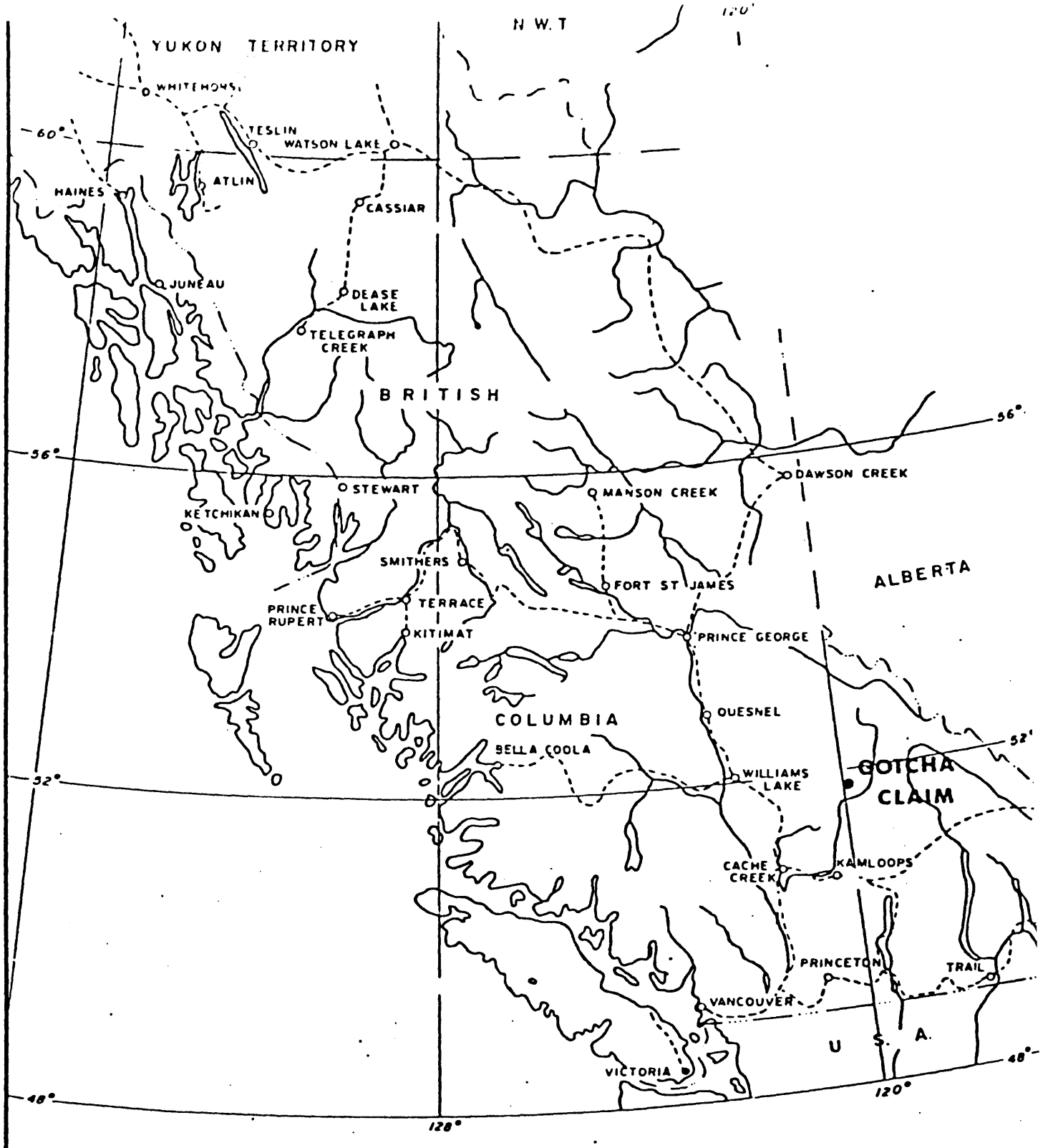
(7) This exploration would cost about $29,000.

(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.

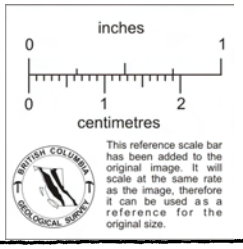
(10) Calculations suggest an after tax profit of $500,000-$600,000+ if all ore is proven.

(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.

APPENDIX A

UNION CARBIDE CANADA MINING LTD.

GEOLOGICAL AND DRILL DATA

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

Introduction

Ownership.....1

Location.....1

Access.....2

Topography.....2

Regional Geology and Structure.....2

Property Geology.....4

(a) The Metasediments.....4

(b) Intrusive Rocks.....6

Structure of the Property.....7

Scheelite Mineralization.....7

Recommendations.....10

Assessment Cost.....12

Maps

#1 Claim Map

#2 Geological Map — In Pocket

#3 Assay Map — "

6 Sections of 8 diamond drill-holes — "

#10 Drill-hole location Map — "

#11 Plan at 3692' level — "

#12 Legend — "

#4 D.D.H. No. 1

#5 D.D.H.'s Nos. 2 & 7

#6 D.D.H. No. 3

#7 D.D.H.'s Nos. 4 & 6

#8 D.D.H. No. 5

#9 D.D.H. No. 8

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.

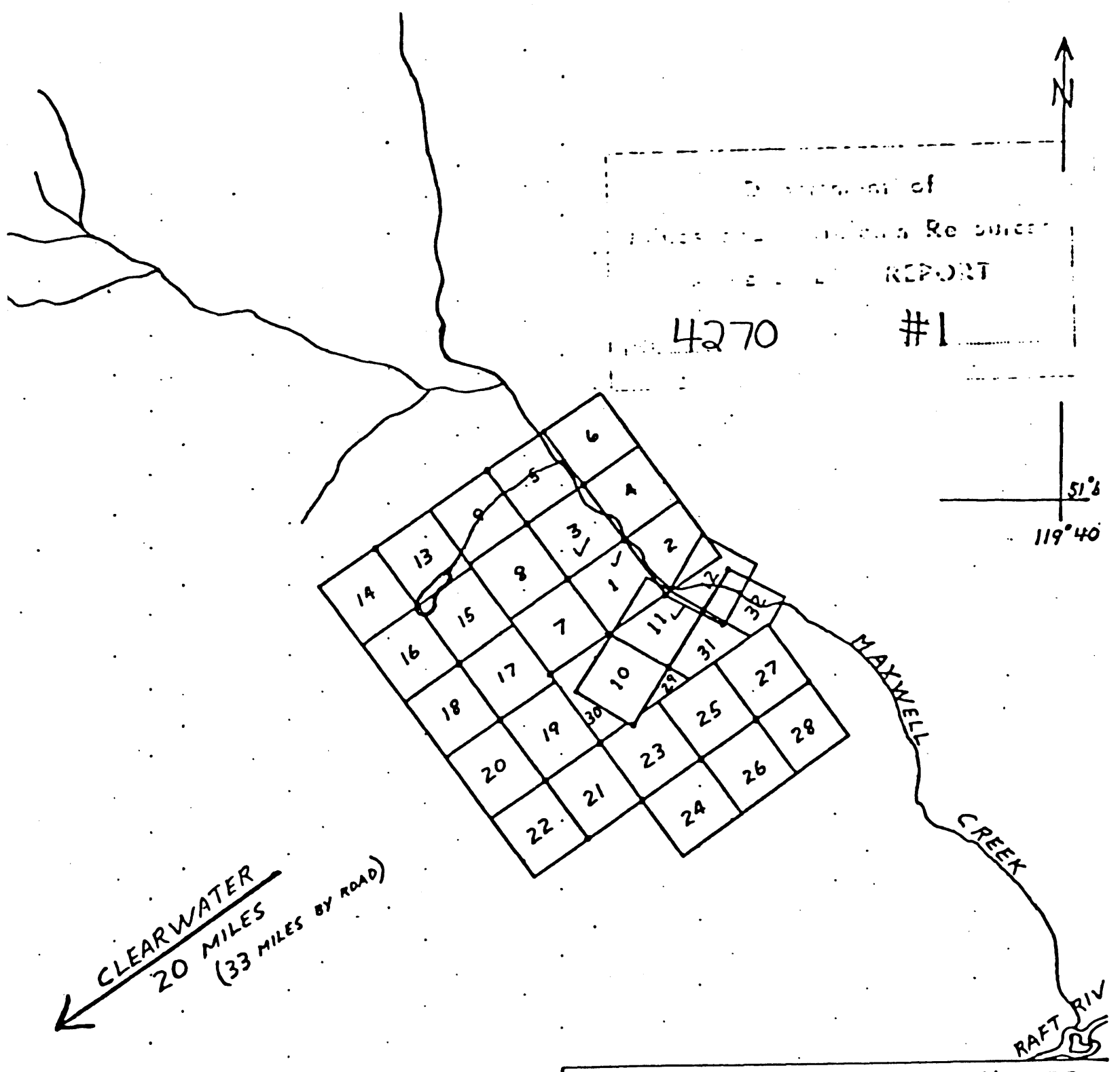
The present report outlines the results of the geological examination carried out on the property commencing August 1st, 1972.

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

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 ska

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.

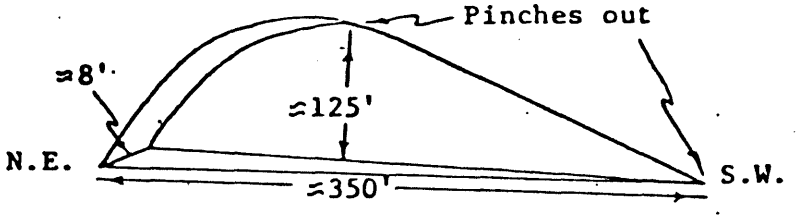
The confinement of the scheelite to one or possibly two units would help to establish the continuity of mineralization and improve the outlook for extensions. If no pattern of continuity can be seen between drill-holes and outcrop, then patchy mineralization (or continuous mineralization offset by faulting)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

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