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REPORT ON

EXPLORATION ON THE BELL GROUP OF MINERAL CLAIMS 1963

AND

PROPOSED EXPLORATION ON THE BELL GROUP FOR 1964

NEAR

CAMBORNE, B.C.

REVELSTOKE MINING DIVISION

Lat 50° Long 118° NE

By

Joseph Sullivan, P. Eng.

Vancouver, B.C. November 30, 1963

82K/13E

546

TABLE OF CONTENTS

PART 1

PART 11

See Report 546

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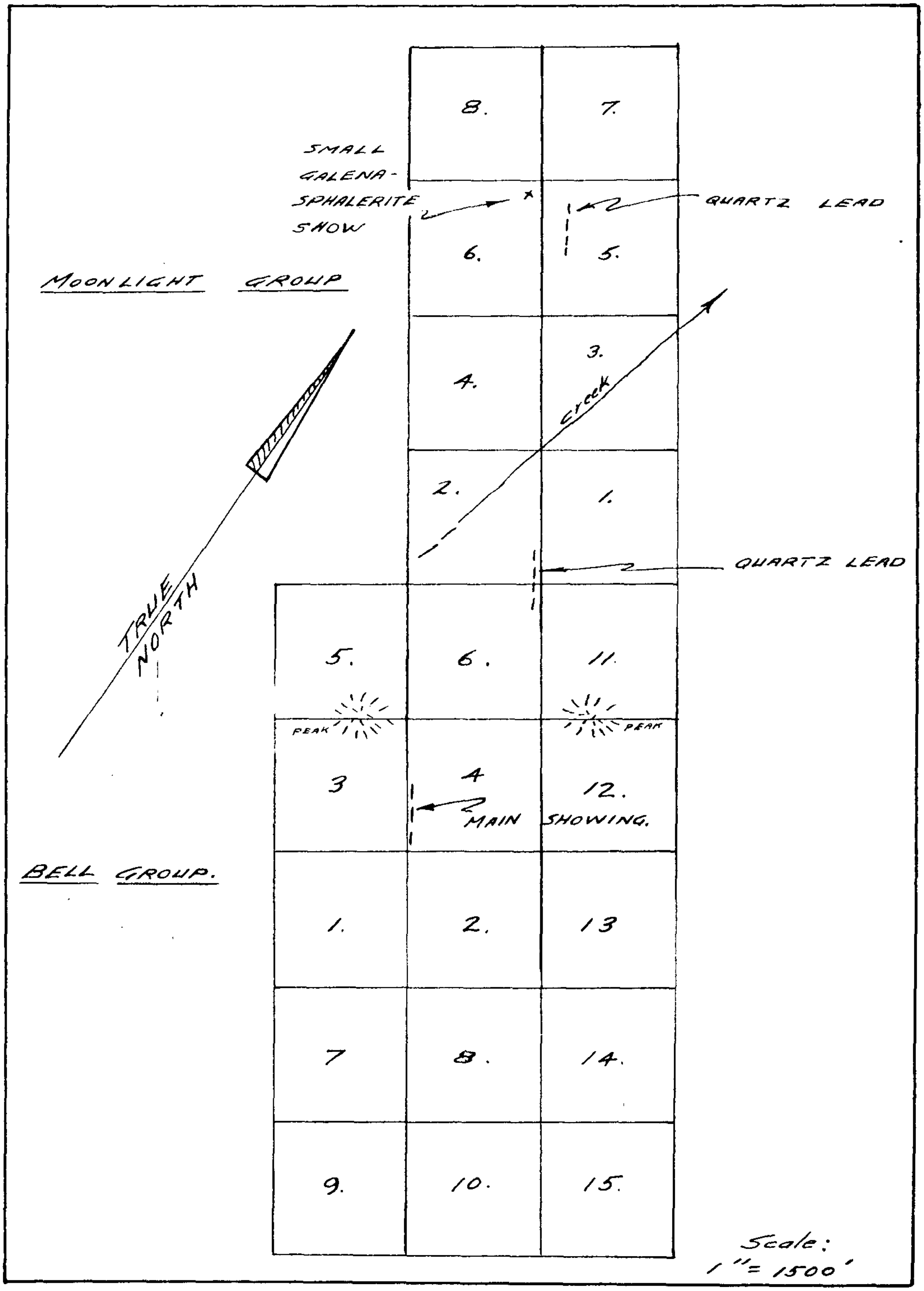
INTRODUCTION:

This property was reported on by the writer in June 1963. At that time a program, much larger than was actually executed, was proposed. However, due to a late start the work was reduced by about three-quarters. Therefore, some of the work proposed herein was originally planned for 1963.

LOCATION AND ACCESSIBILITY:

The claims lie in the Revelstoke Mining Division nine miles due north of Beaton, B.C.

The claims may be approached by road from Beaton to the first bridge crossing the Incomappleux River, then by foot to the headwater of the Middle Fork of Sable Creek. It is estimated that $5,000.00 and 3 to 4 weeks time would be necessary to open this road for a light four-wheel drive truck.



TEDDY GLACIER PROJECT
CLAIM SKETCH
OCT. 1963.

The showings lie in a rocky south facing cirque at 7300 (+) feet A.S.L.

STATUS OF PROPERTY:

There are 23 claims in one contiguous group. The Bell Nos. 1 to 15 and the Moonlight Nos. 1 to 8 are held by location by:

Sunshine Lardeau Mines Ltd.,
401 - 1033 Davie Street,
Vancouver, B. C.

The detail on these claims is as follows:

Name		Record No.	Expiry Date
Bell	1	3960	March 4, 1964
"	2	3961	" "
"	3	3962	" "
"	4	3963	" "
"	5	4596	June 14, 1964
"	6	4597	" "
"	7	4598	" "
"	8	4599	" "
"	9	4600	" "
"	10	4601	" "
"	11	4602	" "
"	12	4603	" "
"	13	4604	" "
"	14	4605	" "
"	15	4606	" "
Moonlight	1	4723	September 12, 1964
"	2	4724	" "
"	3	4725	" "
"	4	4726	" "
"	5	4727	" "
"	6	4728	" "
"	7	4729	" "
"	8	4730	" "

MAPPING AND SAMPLING:

Previously most of the maps produced for the property were assay plans of samples taken on the surface and/or underground. This year much of the work was centered on the diamond drilling and surface mapping. Since the previous work is valuable information it has been included herein. Thus, the map folders include all available maps, and the tonnage-grade calculations are a compilation of all available sample results.

Throughout this report the maps or plans are referred to by numbers allotted by the writer.

GENERAL GEOLOGY:

(1) Rock Types:

The property lies in the Lardeau Series of sedimentary rocks. This series is composed of schist, phyllite, quartzite, slate and limestone. Eight miles northeast lies the Nelson Batholith. This intrusive varies from granite to granodiorite with porphyritic varieties of both types.

(2) Structure:

Though complicated by faults, the structure is essentially complex folds, composed of isoclinal, asymmetric, and overturned types. Fold axes plunge southeast at low angles, and axial planes dip steeply southwest and northeast.

The rocks are sheared and folded to such a degree that little of the original stratigraphy remains. Members prominent in one locality may not be present a short distance away. A unit may appear as lenses that have been sheared by strike faults or may be repeated several times by isoclinal folding.

LOCAL GEOLOGY:

The underlying rocks are limey quartzite, banded graphitic quartzite, and graphitic schist, with intruding granite sills and altered felsite dykes. These rocks occur in a series of tight folds plunging from 5 to 40 degrees southeast and axial planes dipping steeply from northeast to southwest. Such structures are clearly visible over the entire claim group.

The ore shoot appears to be controlled by the axis of a plunging fold in a limey quartzite unit. Thus the so-called Dunbar Vein and the Big Showing could be parts of the same mineralized structure. (See Map Sheets 1, 2, 3, 4, and section indicated thereon.)

MINERALOGY:

The showings consist of galena, sphalerite and pyrite, with additional values in gold and silver. Associated gangue minerals are quartz and carbonate. The sulphides may appear as masses or bands of clean pyrite, galena, and sphalerite, or they may be intimately mixed with the quartz-carbonate gangue.

MINERAL EXPOSURES:

There seems to be enough evidence on maps compiled in the past by competent engineers to permit the writer to say that ore is exposed in three places. Once underground and twice on the surface. Unfortunately snow covered the surface showing until the second last day of the 1963 programme; then, only a few square feet of the "Big Showing" appeared.

The underground exposure, on the 7320 level is likely the downward extension of the "Big Showing" since the two are only about 30 feet apart vertically. Here the mineralization lies in 9x50 foot "nucleus" that branches northwest into an east vein and a west vein.

The Dunbar Vein lies 200 feet in elevation above the 7320 level and 400 feet northwest of the portal. From the mapping done by Mr. W. Blair for the Columinda Corporation, 1952, the writer has conluded that the Dunbar Vein has a similar outline as that of the lower explosures. (See Map Sheets 5, 6, 7, and U-1).

DIAMOND DRILLING:

Six holes were drilled from the surface during the 1963 season. Each hole was to test the mineralization 50 feet below the surface. The locations of the holes have been plotted on map sheet No. 1. The assay results and geology have been logged and recorded on vertical sections in

Appendix 1. All the holes intersected some degree of sulphide mineralization.

MILL TESTS:

Mr. John W. Britton, metallurgist, has been conducting experiments on sulphide recoveries from this ore. This far he appears to be having satisfactory results. His work is included herein as Appendix 11.

ORE TONNAGES AND GRADE:

Using all available information a set of figures for possible tonnage and grade between the 7320 level and the Dunbar Vein was calculated.

The grade was determined by weighting the average grades arrived at in Appendix 111.

Location	W	WxAu	WxAg	WxPb	WxZn
DDH (1963	77.8	14.45	368.04	408.32	396.98
Underground (1963)	62.0	12.00	341.22	364.23	915.61
Dunbar Vein	24.3	3.58	410.64	694.80	184.34
West Vein	16.9	1.69	151.20	307.90	86.36
East Vein	32.9	9.29	574.54	960.73	406.06
Big Showing	55.0	18.21	400.85	840.71	341.16
	274.9	59.22	2246.49	3576.69	2330.53

Tonnage $= \frac{\text{Area x Height}}{10}$

Area is the average area of the mineralised zone as measured on map sheets 5 and U-1 and as calculated from map sheet 5. Height is the vertical distance between the Dunbar Vein and the exposures on 7320 level, 200 feet. The figure 10 is an assumed tonnage factor of 10 cubic feet per ton of ore.

Thus,

$$\text{Possible Tonnage} = \frac{1117 \times 200}{10} = \underline{\underline{22,340.0 \text{ tons}}}$$

Grade:	Au	=	0.22 o/T
	Ag	=	8.17 o/T
	Pb	=	13.0 %
	Zn	=	8.5 %

DISCUSSION:

The 1963 work did much to confirm the presence of gold-silver-lead-zinc ore of mineable grade. However, the amount, total tonnage, is a long way from being determined, Even the "possible ore" figure is questionable, since the basis for projecting ore between the exposures is unproven geological theory. Some support is given by the geological mapping and the drill section for DDH No. 6, but the ideas on ore control have not yet outgrown the theory stage.

The assays used for arriving at a set of figures for the ore grade are all reported by B.C. Professional Engineers and should be reliable. Thus the grade figures reported under "Tonnage and Grade" should be very close to the actual value of the ore in place.

RECOMMENDATIONS:

The first step would be to start repairs on the road up Sable Creek. This would not be a large road building project, only repairing the present road sufficient to allow the passage of a small four wheel drive vehicle. If this was started early in June, there should be access to the property by early July.

The next step would be to follow up the 1963 indications with a conventional underground diamond drilling programme. Such a programme would require about 1,000 feet of drilling. The suggested patterns have been placed on drill sections and included in Appendix IV.

It would take until early August to complete the underground drilling. At this time sufficient rock should be exposed to permit drilling from the surface. Surface drill holes located on the basis of the results of the underground drilling could likely follow the ore shoot down the hillside for another 400 feet horizontally. This would be done in short jumps of 50 to 100 feet. Such a programme would require 2,000 feet of drilling and would be completed about the end of September.

As was suggested for 1963, a prospecting programme should be conducted in the surrounding area, - a programme that is within a five mile radius of the main operation.

Possibly geophysical prospecting will be indicated to aid in locating targets for the surface drilling. Therefore, a sum of money should be set aside to cover the cost of the additional survey work.

COST OF RECOMMENDATIONS:

The cost for such a programme may be summarized as follows:

A - Premobilization Expenses:		
Engineering fees and expenses	$ 1,500.00	
Camp supplies & equipment	750.00	
Gas caches	800.00	
Radio Rental	500.00	$ 3,550.00
B - Mobilization:		
Travel	$ 200.00	
Cartage	100.00	
Standby and Stopovers	200.00	
Reserve for Helicopter 5 hrs. @ $130.00	650.00	1,150.00
C - Wages and Salaries:		
Prospectors, 4 men - 3.5 mons $2,000.00 x 3.5 =	$ 7,000.00	
Geologist & Assistant $1,300.00 x 3.5 =	4,350.00	
Cook - $350.00 x 3.0 =	1,050.00	
W.C. and fringes & 12%	1,512.00	14,112.00
Carried Forward		$ 18,812.00

Brought Forward		$ 18,812.00
D - Technical:		
Geophysics reserve	$ 1,000.00	
Assaying, 150 samples @$9.00/sam.	1,350.00	
Engineering, supervision & travel	2,500.00	4,850.00
E - General Expenses during Season:		
Food & Hardware (4 mons.)	$ 3,000.00	
Ground transportation 4 months @ $250.00/mon.	1,000.00	
Additional freight	200.00	
Helicopter service 15 hrs. @ $90.00/hr.	1,350.00	
Office expenses @ $250.00/mon. 4 months	1,000.00	
Bulldozing - 126 hours @ $18.00/hr.	2,268.00	
Power Saw & tools (S.H.)	500.00	9,318.00
F - Diamond Drilling:		
3,000 ft. Ax drilling @ $5.75/ft. including both surface & underground =	$ 17,250.00	17,250.00
G - Demobilization:		
Travel	$ 200.00	
Cartage	100.00	
Standy & Stopovers	300.00	
Reserve for helicopter 5 hrs. @ $90.00/hr.	450.00	1,050.00
		$ 51,280.00
H - Final Adjustments:		
Total of Estimates $51,280.00		
Contingencies @ 10% =	$ 5,128.00	
$5,000.00 is estimated for this season's road repairs. This is included in above costs. Assuming Department of Mines pays 50% deduct	2,500.00	2,628.00
Final Total		$ 53,908.00
Say		$ 54,000.00

Respectfully submitted,

Jos. Sullivan, P. Eng.

Vancouver, B.C.,
November 30th, 1963.

C E R T I F I C A T E

I, Joseph Sullivan, of the City of Vancouver, in the Province of British Columbia, hereby certify as follows:

1. That I am a Registered Professional Engineer of British Columbia, residing at 2766 West 30th Avenue, Vancouver 8, B.C.

2. That I am a graduate of the University of British Columbia and have practiced my profession for 12½ years.

3. I have no direct or indirect interest in the Bell or Moonlight Groups of mineral claims.

4. The accompanying report is based on several days stay on the claims studying the geology and economics of the immediate area. This field work was done during August and September, 1963.

Jos. Sullivan

Jos. Sullivan, P. Eng.

Vancouver, B.C.,
November 30th, 1963.

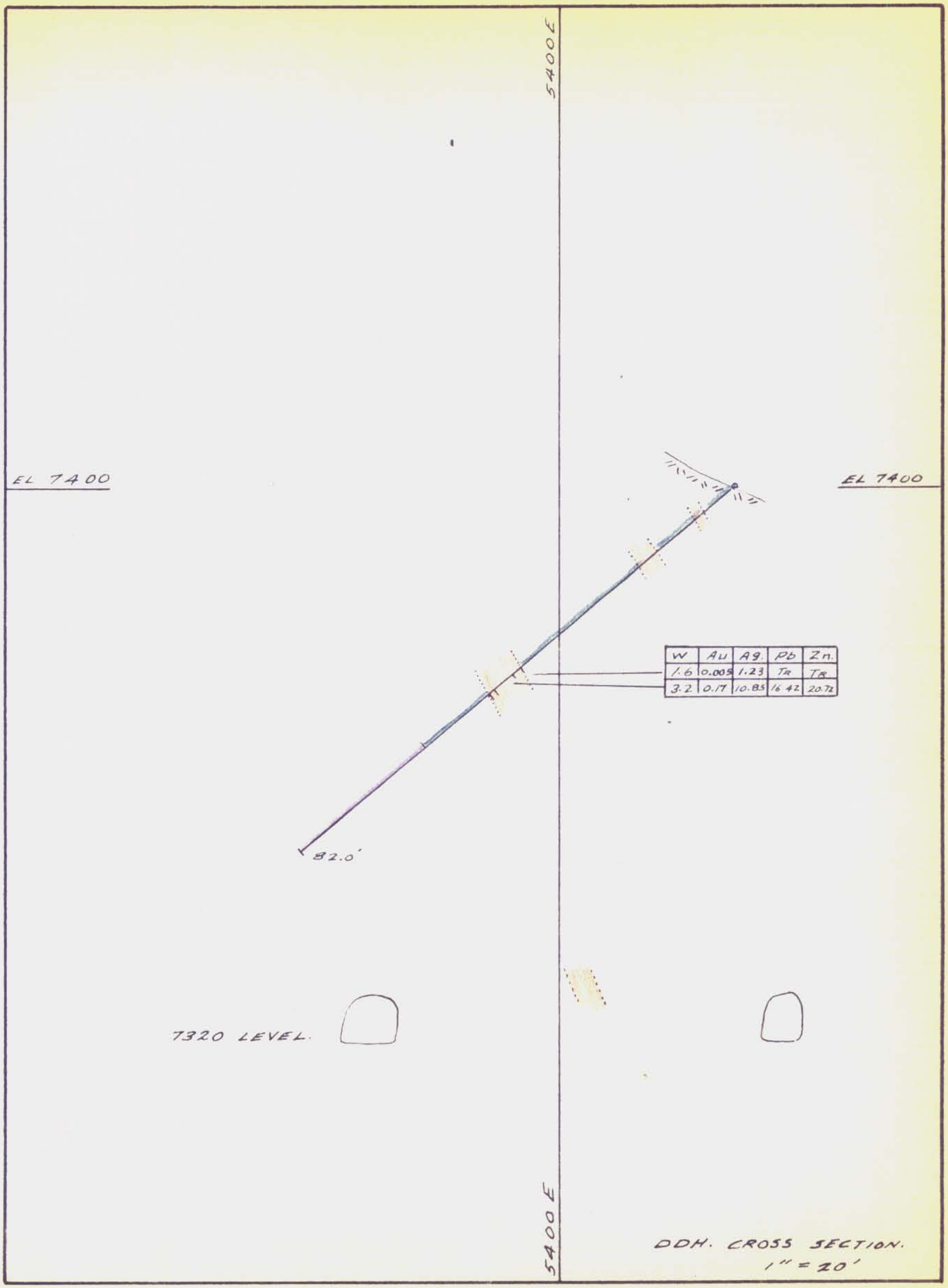
A P P E N D I X 1

DIAMOND DRILLING 1963

LOGS

AND

SECTIONS



W	Au	Ag.	Pb	Zn.
1.6	0.005	1.23	TR	TR
3.2	0.17	10.85	16.42	20.72

TEDDY GLACIER PROPERTY.

DDH. № 1.

BEARING S 60° W.

AUGUST 1963.

Map

PROPERTY Teddy Glacier

DRILL HOLE NO. 1

SHEET 1

BEARING S 60°W

LAT. 5718

DATE COMMENCED August 1963

DIP AT COLLAR -40°

DEP. 5425

DATE COMPLETED

DEPTH OF HOLE 82.0'

ELEV. 7400'

REMARKS

PURPOSE OF HOLE To explore sulphide zone at 50 foot depth.

FOOTAGE	DESCRIPTION		Recovery	SAMPLE NO.	WIDTH IN FEET	ASSAY	ASSAY	ASSAY	ASSAY
0.0 - 38.0	Medium grey, bedded quartzite with intercalated beds of limestone up to 6 inches thick. Much quartz and carbonate as bedded seams and irregular bands and threads.								
	0.0 - 5.0		3.5						
	5.0 - 8.0	50% qtz with minor py and carb	3.0						
	8.0 -14.0		8.5						
	14.0 - 17.5	50% qtz with minor py and carb	2.0						
	17.5 - 19.0		2.0						
	19.0 - 32.0	beds are 60° to core	6.5						
	32.0 - 34.0	" " 90° to core	1.0						
	34.0 - 40.2	Graphitic schistose with beds varying from 30 to 90 degrees to core, some brecciation	4.5						
		Recovery C/F	31.0						

PROPERTY Teddy Glacier

DRILL HOLE NO. 1 Map SHEET 1

BEARING

LAT.

DATE COMMENCED

DIP AT COLLAR

DEP.

DATE COMPLETED

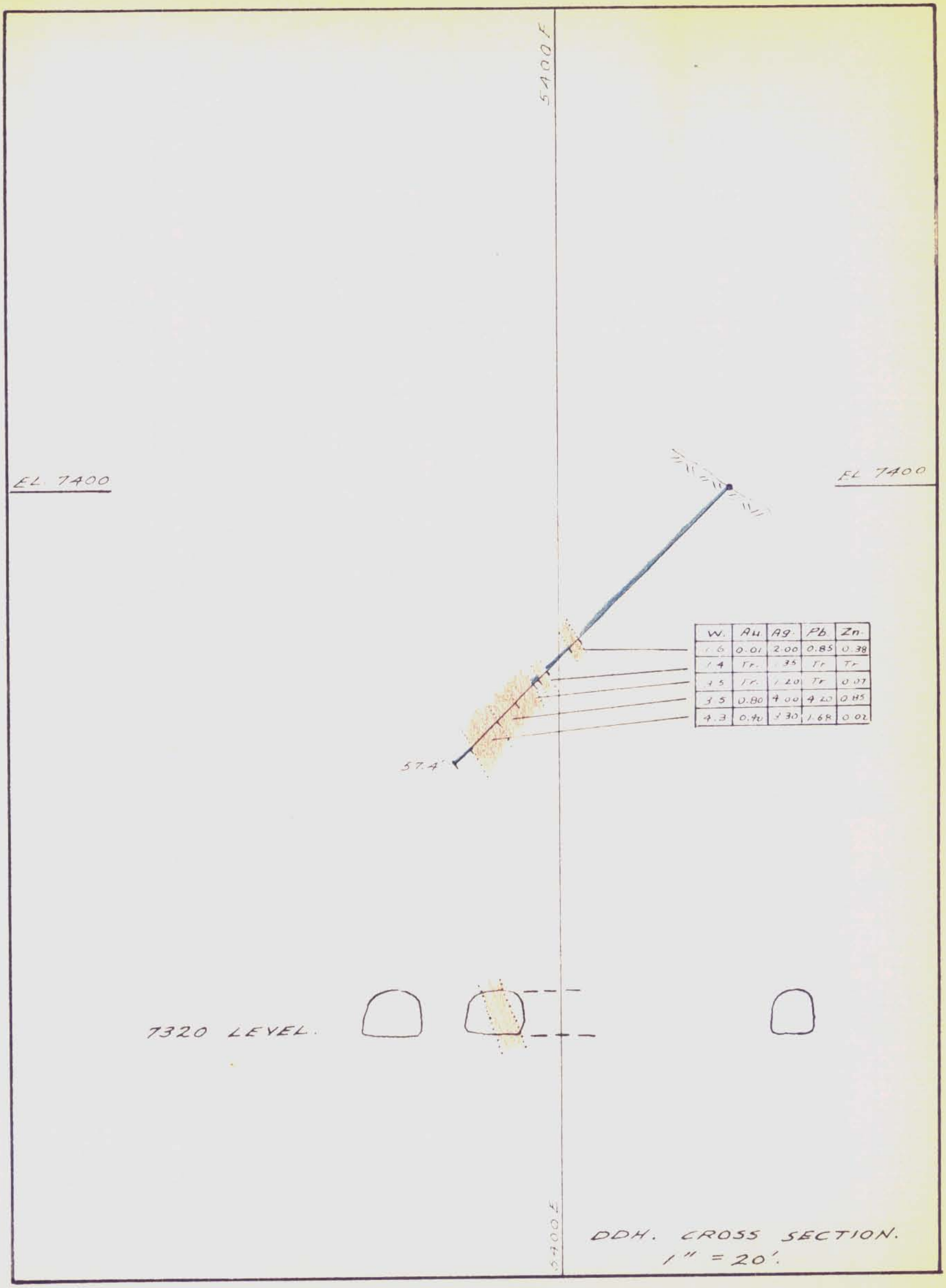
DEPTH OF HOLE

ELEV.

REMARKS

PURPOSE OF HOLE

FOOTAGE	DESCRIPTION		Recovery	SAMPLE NO.	WIDTH IN FEET	Au ASSAY	Ag ASSAY	Pb ASSAY	Zn ASSAY
	40.2 - 41.8	vein matter with sulphides of PbS, ZnS, CuS, FeS.	1.5	5001	1.6	0.005	1.23	Tr	Tr
	41.8 - 45.0	As above but sulphides are heavy	3.0	5002	3.2	0.17	10.85	16.42	20.72
	45.0 - 46.0	Milky quartz, no sulphide	1.0						
	46.0 - 58.0	Beds 45° - 55° to core	11.0						
58.0 - 82.0	Siliceous graphitic schist with minor carbonate. Tends to grade into ligher grey quartzite for lengths up to 6 inches								
	82 ft. is end of hole								
	Recovery B/F		31.0						
	Total recover		47.5						



W.	Au.	Ag.	Pb.	Zn.
1.6	0.01	2.00	0.85	0.38
1.4	Tr.	.35	Tr	Tr
4.5	Tr.	1.20	Tr	0.07
3.5	0.80	4.00	4.20	0.85
4.3	0.40	3.30	1.68	0.02

TEDDY GLACIER PROPERTY.

DDH. N° 2.

BEARING S 30° W.

AUGUST 1963.

PROPERTY Teddy Glacier DRILL HOLE NO. 6 Map SHEET 1

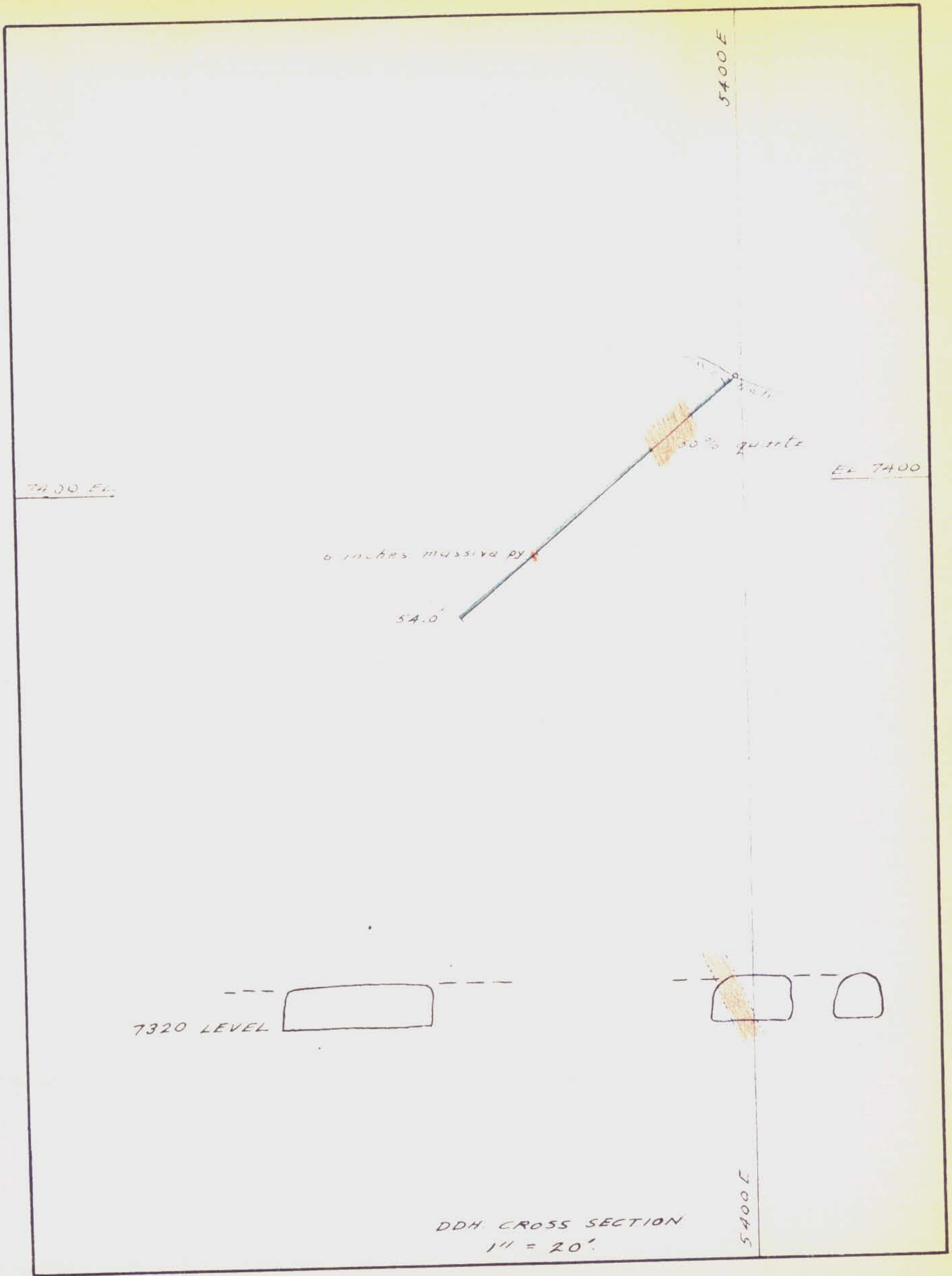
BEARING S 40°W LAT. 5621 DATE COMMENCED September 1963

DIP AT COLLAR - 40° DEP. 5512 DATE COMPLETED

DEPTH OF HOLE 109.0' ELEV. 7350' REMARKS

PURPOSE OF HOLE To explore possiblity of plunging fold structure

FOOTAGE	DESCRIPTION	Recovery	SAMPLE No.	WIDTH IN FEET	Au ASSAY	Ag ASSAY	Pb ASSAY	Zn ASSAY
0.0-29.0	Siliceous, graphitic schist, including small beds of quartzite, most partings at 70° with core	19.0						
29.0-73.0	Med. grey siliceous quartzite with graphitic partings. At 39.0' the hole intersected the crest of a small fold. At 53.0' partings are 60° to core	34.0						
73.0-109.0	Mixed graphitic schist and banded quartzite							
	73.0 - 78.0	4.0						
78.0 - 88.0	Suiphides Py.Gal.Sph.Cpy	2.0	5012	10.0	0.06	2.90	1.40	2.15
88.0 - 93.0	" " " " "	1.3	5013	5.0	0.005	1.60	0.33	0.06
93.0 - 98.5	" " " " "	3.0	5014	5.5	0.52	5.70	6.50	7.17
98.5 -103.5	" " " " "	4.5	5015	5.0	0.14	3.45	7.03	10.35
103.5 -109.0	" " " " "	1.7	5016	5.5	0.005	2.40	0.80	0.23
68.0 - 73.0	Sludge				0.005	1.05	Tr	Tr
78.0 - 83.0	"				0.24	3.85	Tr	Tr
83.0 - 88.0	"				0.005	1.16	4.03	9.65
89.0 - 93.6	"				0.04	1.90	0.73	1.05
93.0 - 98.0	"				0.22	3.50	1.90	9.40
98.0 -101.0	"				0.44	4.65	2.77	15.05
101.0 -104.0	"				0.12	2.50	1.65	3.43
104.0 -109.0	"				0.10	2.25	0.64	0.26
88.0 - 89.0	"				0.16	2.90	1.60	5.73
	109.0' is end of hole							
	Recovery	69.5'						



TEDDY GLACIER PROPERTY.

DDH Nº 3.

BEARING S 60° W.

AUGUST 1963.

PROPERTY Teddy Glacier | DRILL HOLE NO. 5 | Map SHEET 1

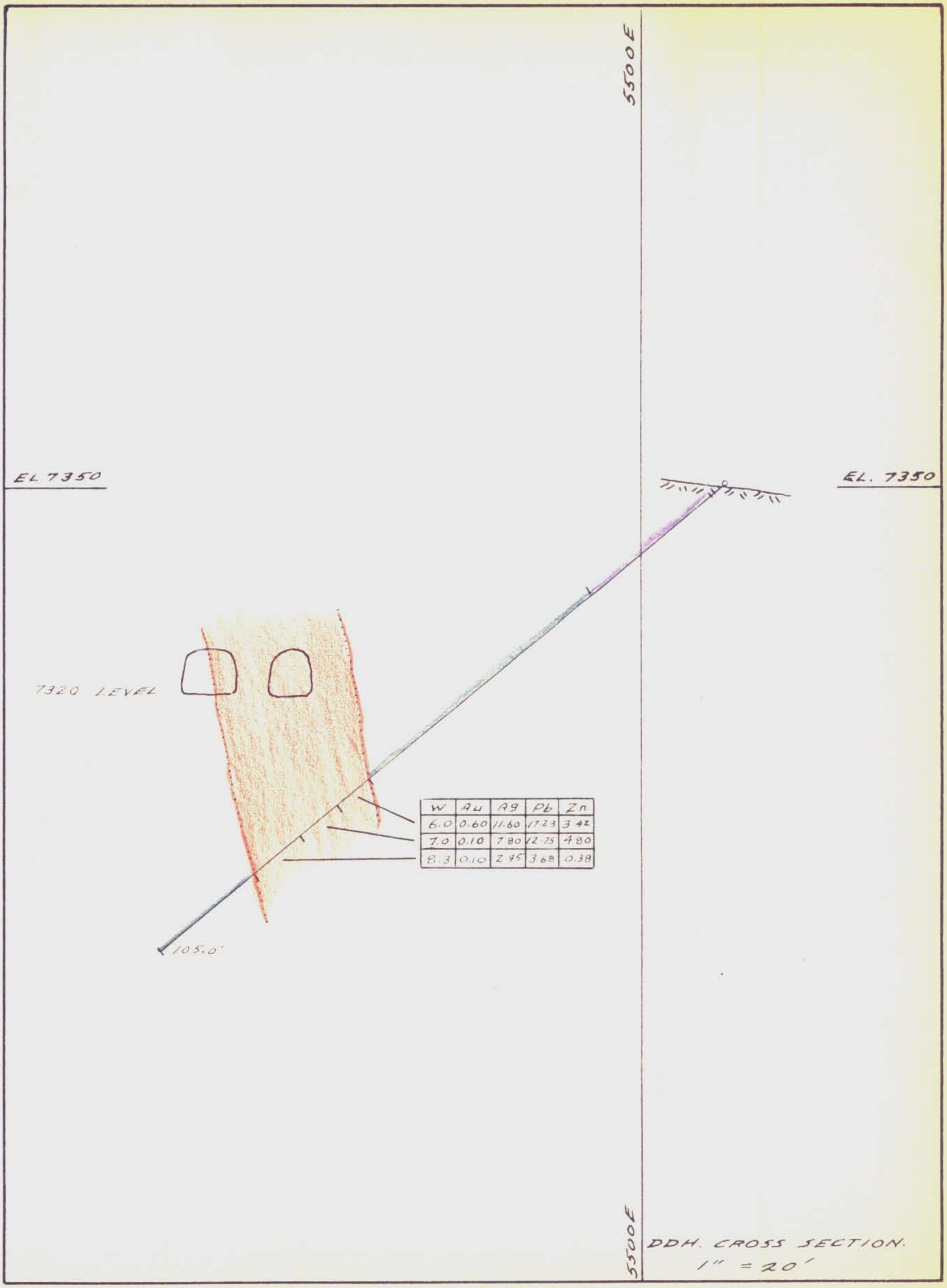
BEARING West | LAT. 5626 | DATE COMMENCED September 1963

DIP AT COLLAR - 40° | DEP. 5511 | DATE COMPLETED

DEPTH OF HOLE 89.0' | ELEV. 7350' | REMARKS

PURPOSE OF HOLE

FOOTAGE	DESCRIPTION		Recovery	SAMPLE No.	WIDTH IN FEET	Au ASSAY	Ag ASSAY	Pb ASSAY	Zn ASSAY
0.0 - 21.0	Siliceous graphitic schist		14.0						
21.0 - 55.0	Mixed siliceous graphitic schist and medium grey quartzite, There is a marked increase in the quartz stringers in this unit		30.0						
55.0 - 67.5	Siliceous graphitic schist		11.0						
67.8 - 89.0	Medium grey quartzite with quartz-carbonate sulphide zone. Py, gal, sph, cpy.								
	67.8 - 74.0	Sulphide zone	4.3	5009	6.2	0.12	4.70	3.50	16.62
	74.0 - 77.5	"	3.0	5010	3.5	0.20	3.20	1.84	9.23
	77.5 - 80.3	"	1.8	5011	2.8	0.10	3.70	4.28	7.22
	67.5 - 72.5	Sludge				0.52	5.55	3.25	12.17
	72.5 - 76.0	"				0.63	5.15	2.20	35.19
	76.0 - 80.6	"				0.14	6.10	8.40	7.50
	80.6 - 84.0	"				0.03	1.50	0.38	Tr
	80.0 - 89.0	"	8.0						
		89.0' is end of hole							
		Recovery	72.1'						



W	Au	Ag	Pb	Zn.
6.0	0.60	11.60	17.23	3.42
7.0	0.10	7.80	12.75	4.80
8.3	0.10	2.95	3.68	0.38

TEDDY GLACIER PROPERTY.

DDH № 4.

BEARING S 60° W

AUGUST 1963.

Map SHEET 1

DRILL HOLE NO. 4

PROPERTY Teddy Glacier

DATE COMMENCED August 1963

DATE COMPLETED

REMARKS

BEARING S 60°W

LAT. 5623

DIP AT COLLAR - 40°

DEP. 5511

DEPTH OF HOLE 105.0'

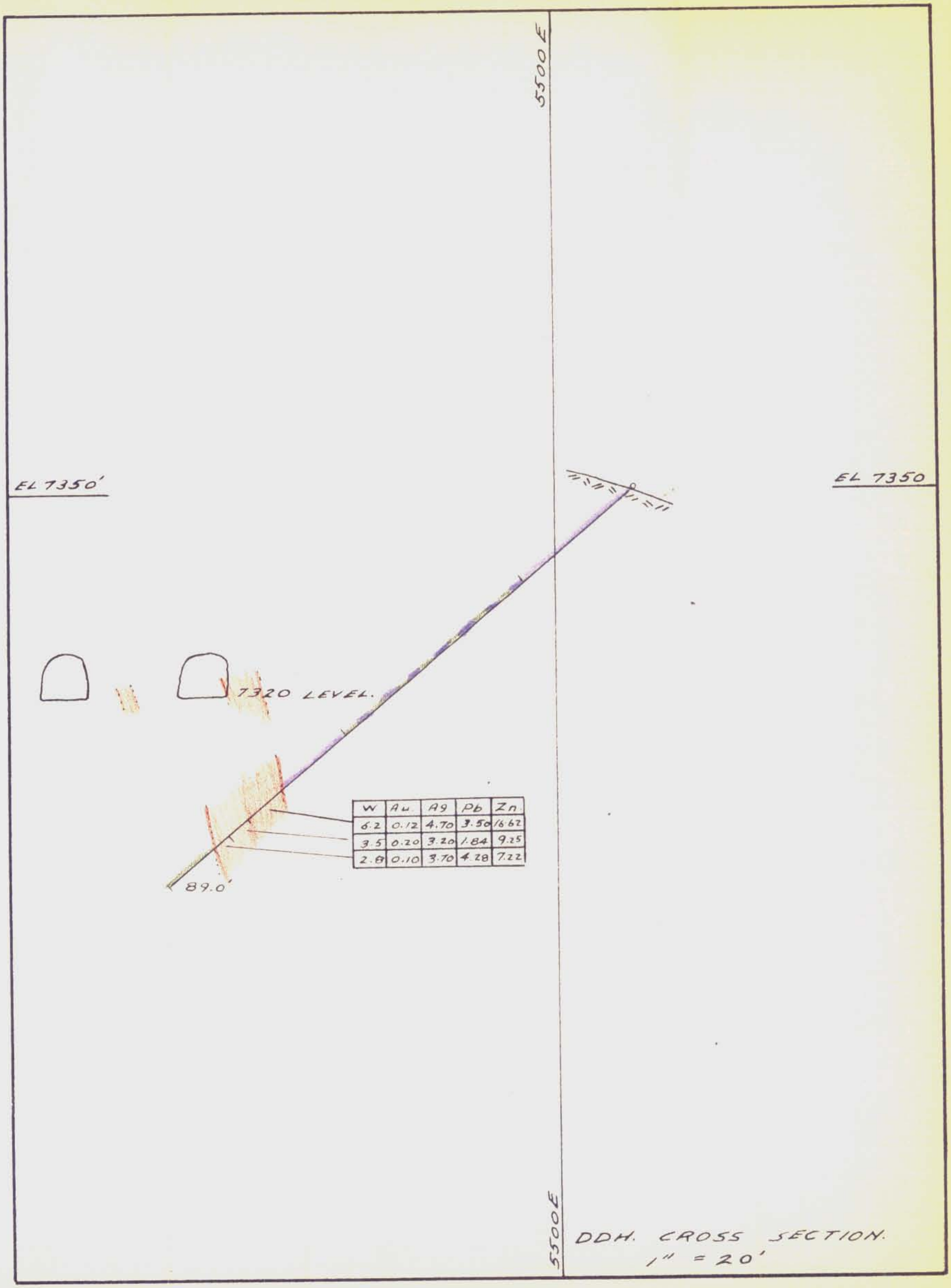
ELEV. 7350'

PURPOSE OF HOLE To explore sulphide zone at 50 foot depth.

FOOTAGE	DESCRIPTION		Recovery	SAMPLE No.	WIDTH IN FEET	Au ASSAY	Ag ASSAY	Pb ASSAY	Zn ASSAY
0.0 - 24.0	Black, siliceous, graphitic schist Partings 90° to core		13.0						
24.0 -105.0	Chiefly siliceous, bedded quartzite with thin beds of limestone								
	24.0 - 35.3	---	10.5						
	35.3 - 53.0	---	13.0						
	53.0 - 66.0	---	11.5						
	66.0 - 72.0	Sulphide zone, Py.gal,sph,cpy.	15.0	10293B	6.0	0.60	11.60	17.23	3.42
	72.0 - 79.0	" " " " " "	4.2	10294B	7.0	0.10	7.80	12.75	4.80
	79.0 - 87.3	" " " " " "	3.6	10295B	8.3	0.10	2.95	3.68	0.38
	67.0 - 72.0	S L U D G E		-		0.90	14.30	14.13	8.08
	72.0 - 76.0	"		-		0.40	4.40	2.97	4.11
	76.0 - 79.0	"		-		0.46	7.20	9.35	8.50
	79.0 - 81.0	"		-		0.28	5.10	4.78	7.15
	81.0 - 83.0	"		-		0.22	4.00	2.93	4.37
	83.0 - 84.0)	"		-		-	-	-	-
	84.0 - 85.5)	"		-		0.14	2.10	0.74	0.66
	85.5 - 87.3	"		-		0.21	3.00	2.60	1.90
	87.3 - 88.9	"		-		0.16	2.50	2.04	0.74
	88.9 - 94.0	"		-		0.04	3.10	0.37	4.50
	94.0 - 96.0	"		-		0.12	2.05	0.22	0.82
	96.0 -101.0	"		-		0.01	1.60	Tr	0.32
	87.3 -105.0		6.0						

105.0 is end of hole

Recovery 70.8'



W	Au.	Ag	Pb	Zn.
6.2	0.12	4.70	3.50	16.62
3.5	0.20	3.20	1.84	9.25
2.8	0.10	3.70	4.28	7.22

TEDDY GLACIER PROPERTY

DDH. Nº 5

BEARING: WEST.

SEPTEMBER 1963.

PROPERTY Teddy Glacier

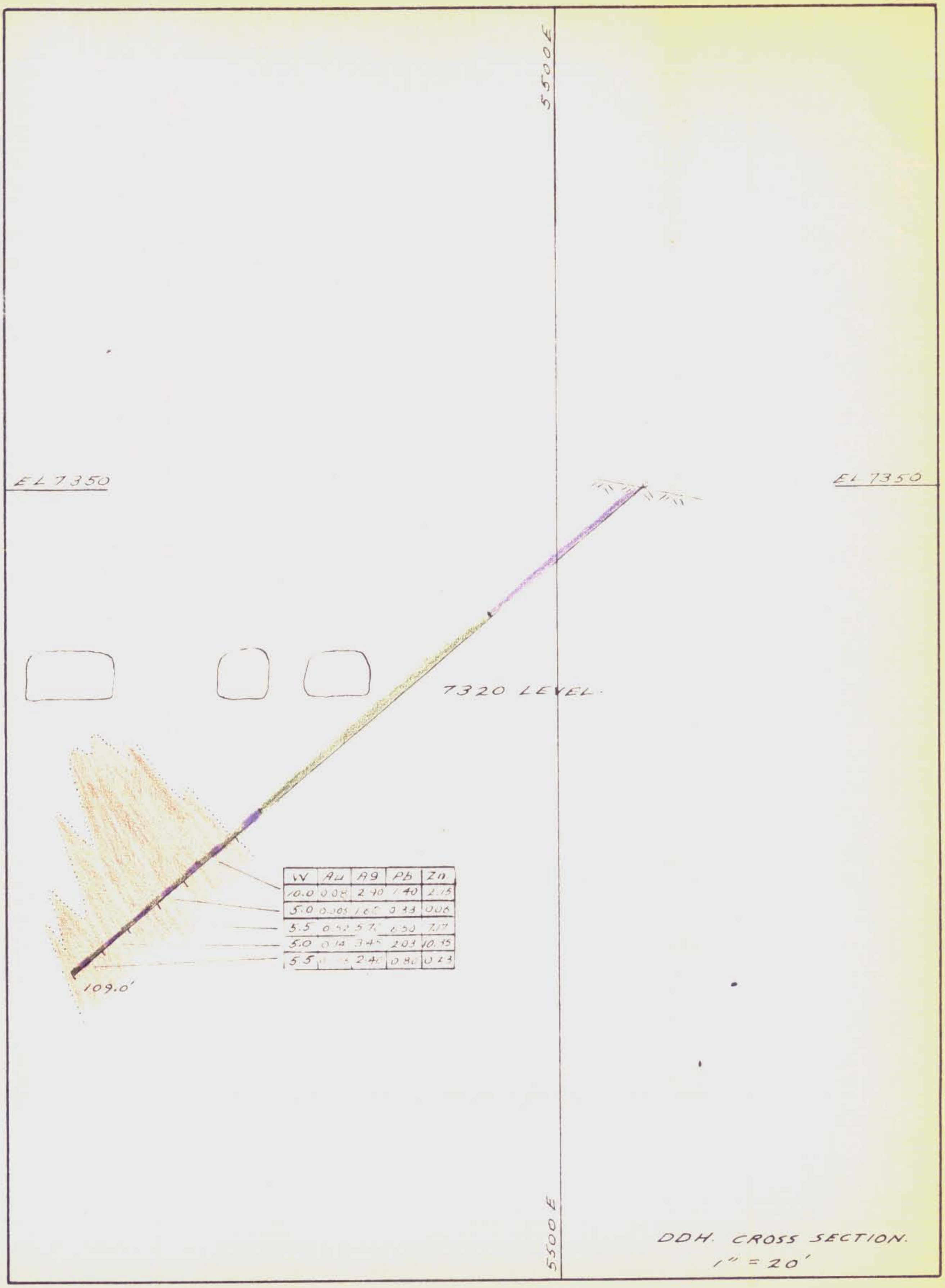
DRILL HOLE NO. 3

Map SHEET 1

B. RING	S 60°W	LAT.	5747	DATE COMMENCED	August 1963
DIP AT COLLAR	- 40°	DEP.	5399	DATE COMPLETED	
DEPTH OF HOLE	54.0'	ELEV.	7416'	REMARKS	This hole appears to be above the target

PURPOSE OF HOLE To explore sulphide zone at a depth of 30 feet.

FOOTAGE	DESCRIPTION		Recovery	SAMPLE No.	WIDTH IN FEET	ASSAY	ASSAY	ASSAY	ASSAY
0.0 - 54.0	Light to med. grey banded quartzite with irregular spaced soft beds of limestone. Irregular bands and threads of quartz carbonate.								
	0.0 - 8.0	-----	4.0						
	8.0 -16.0	50% vein matter, no sulphide	7.0						
	16.0 -27.0	beds normal to core	9.0						
	27.0 -29.5	light buff alteration	2.0						
	29.5 -39.0	-----	8.0						
*	39.0	bunches massive							
	39.0 -44.0	beds soft and dark	4.0						
	44.0 -54.0	-----	5.5						
		54.0' is end of hole	____						
		Total recovery	39.5						



W	Au	Ag	Pb	Zn
10.0	0.08	2.90	1.40	2.15
5.0	0.005	1.6[illegible]	0.33	0.06
5.5	0.52	5.7[illegible]	6.50	7.17
5.0	0.14	3.45	2.03	10.35
5.5	[illegible]	2.40	0.80	0.23

TEDDY GLACIER PROPERTY.

DDH Nº 6.

BEARING S 40° W.

SEPTEMBER 1963.

PROPERTY Teddy Glacier DRILL HOLE NO. 2 SHEET 1

BEARING

LAT.

DATE COMMENCED August 1963

DIP AT COLLAR

DEP.

DATE COMPLETED

DEPTH OF HOLE

ELEV.

REMARKS

PURPOSE OF HOLE

FOOTAGE	DESCRIPTION	Recovery	SAMPLE NO.	WIDTH IN FEET	Au ASSAY	Ag ASSAY	Pb ASSAY	Zn ASSAY
31.6 - 33.2	Rusty siliceous zone with minor py, gal, sph.	1.5	5003	1.6	0.01	2.00	0.85	0.38
38.0 - 39.4		1.4	5004	1.4	Tr	1.35	Tr	Tr
40.3 - 44.0		3.5	5005	3.7	Tr	1.20	Tr	0.07
44.0 - 48.0		3.5	5006	4.0	0.80	4.00	4.20	0.85
48.0 - 53.8		4.3	5007	5.8	0.40	3.30	1.68	0.02
48.0 - 53.8	SLUDGE	--	5008	5.8	0.26	4.45	3.25	1.68
53.8 - 57.0	Beds 60° to core	4.0						
	57.0 is end of hole							
	Recovery B/F	23.5						
	Total Recovery	41.7						

PROPERTY Teddy Glacier

DRILL HOLE NO. 2

Map
SHEET 1

[illegible]NG	S 30°W	LAT	5718	DATE COMMENCED	August 1963
DIP AT COLLAR	- 45°	DEP	5625	DATE COMPLETED	
DEPTH OF HOLE	57.4'	ELEV	7400	REMARKS	

PURPOSE OF HOLE To e plore sulphide zone at 50 foot depth.

FOOTAGE	DESCRIPTION	Recovery	SAMPLE NO.	WIDTH IN FEET	ASSAY	ASSAY	ASSAY	ASSAY
0.0 - 57.6	Light to medium grey banded quartzite with irregular soft bands of light grey limestone. Some bands and threads of quartz and lesser carbonate.							
	0.0-16.0 appears to be silicified limestone, but present hardness is equal to quartzite. This hole was collared in a highly distorted zone with much quartz and some py and carbonate (assume this mineralized length covers from 0.0 - 6.5 feet)	10.0						
	16.5-36.5 beds 80 - 90 degrees with core some graphitic partings 26.0-29.5, siliceous with buff alteration	13.5						
	Recovery C/F	23.5						

APPENDIX 11

METALLURGY

BY

BRITTON RESEARCH LABORATORIES

Mr. L. G. White,
Sunshine Lardeau Mines Ltd.,
Suite 401, 1033 Davie Street,
Vancouver 5, B.C.

November 5, 1963

Dear Len:

Re: Teddy Glacier Metallurgy

We give below a summary of the preliminary test-work which we have carried out on the bulk sample of ore referred to in your letter of September 26, 1963:

Assay of Ore:

Gold	0.40 ounces per ton
Silver	0.8 " " "
Copper	2.56%
Lead	12.06%
Zinc	13.21%
Cadmium	0.04%
Iron (acid-sol.)	10.63%
Sulphur	21.99%
Antimony	0.03%
Arsenic	Trace
Barium	Not detected

Method of Concentration:

After grinding the ore to 64% minus 200 mesh, the copper and lead were floated, together with gold and silver. This was followed by flotation of the zinc and finally the pyrite. The copper-lead concentration was cleaned once and the copper and lead minerals were then separated in two stages, using dichromate to depress the lead whilst the copper was floated. The zinc concentrate was cleaned once.

Test Results:

(a) Copper Concentrate (weight = 8.0% of ore)

	ASSAYS	RECOVERIES
Gold	0.65 ounces/ton	14.3%
Silver	24.6 ounces per ton	23.5%
Copper	25.45%	79.6%
Lead	8.82%	6.0%
Zinc	4.02%	2.4%

L.G. White (Cont'd)

(b) Lead concentrate (weight = 16.6% of ore)

	Assays	Recoveries
Gold	1.21 ounces per ton	52.6%
Silver	29.1 ounces per ton	57.6%
Copper	0.81%	5.1%
Lead	62.43%	87.1%
Zinc	5.11%	6.4%

(c) Zinc concentrate (weight = 17.4% of ore)

	Assays	Recoveries
Gold	0.06 ounces per ton	2.7%
Silver	2.4 ounces per ton	5.0%
Copper	0.60%	4.0%
Lead	0.26%	0.4%
Zinc	61.83%	81.1%
Cadmium	0.31%	-

(d) Pyrite concentrate (weight = 20.2% of ore)

	Assays	Recoveries
Gold	0.41 ounces per ton	21.7%
Silver	2.8 ounces per ton	6.7%
Copper	0.99%	7.7%
Lead	1.30%	2.2%
Zinc	1.58%	4.0%
Pyrite	73.6%	72.8%

(e) Overall recoveries (total rougher and concentrates)

Gold	97.0%
Silver	99.3%
Copper	99.7%
Lead	99.3%
Zinc	99.9%
Pyrite	98.0%

Tests showed that 70% of the gold and 60% of the silver in the pyrite concentrate could be recovered by cyanidation

L.G. White (Cont'd)

but the cyanide consumption was high (13 pounds per ton of concentrate), due to the presence of soluble copper minerals.

The results indicate that, when treating similar ore in a full-scale plant, at least 80% of the copper, 90% of the lead and 85% of the zinc should be recoverable in the form of separate concentrates assaying over 25% copper, 60% lead and 60% zinc respectively; about 70% of the gold and 85% of the silver could be recovered with the lead and copper. Even better results should be obtained when treating fresh ore. It may also be possible to recover some of the gold and silver which reports with the pyrite. No unusual problems are anticipated in treating the ore.

Yours very truly,

BRITTON RESEARCH LABORATORIES

(Sgd.) John W. Britton, P. Eng.

John W. Britton, P. Eng.

(SEAL)

METALLURGICAL TESTS ON SAMPLES OF ORE FROM THE TEDDY GLACIER PROPERTY OF SUNSHINE LARDEAU MINES LIMITED

Report No. 1

Project No.: B34
Date: December 24, 1963.

Investigation by: John W. Britton, P. Eng.,
Britton Research Laboratories,
755 Beatty Street,
Vancouver 3, B. C.

INTRODUCTION

A bulk sample consisting of 200 pounds of ore from the Teddy Glacier property of Sunshine Lardeau Mines was received on September 26, 1963. Results of preliminary metallurgical tests on the sample were given in a letter report dated November 5, 1963.

Samples of split drill core were received on November 3, 1963. A composite sample was made up using all of the available mineralised sections of the core (see Appendix).

Instructions to proceed with the tests were given by L. G. White, P. Eng., President of Sunshine Lardeau Mines Limited.

SUMMARY

Results of the best test on each sample are shown in the table on pages 4 and 5 of this report. The table also shows anticipated results when treating ore of grade equal to that of the present estimated ore reserves, namely 0.22 ounces of gold per ton, 8.17 ounces of silver per ton, 0.7% copper, 13.00% lead and 8.50% zinc. The results indicate that about 60% of the copper, 90% of the lead and 80% of the zinc should be recovered in the form of separate concentrates assaying about 20% Cu, 60% Pb and 60% Zn respectively.

Calculations have also been made of the gross value and probable net smelter returns when treating ore of the above grade. These are as follows:

Gross value of ore	$82.46 per ton
Net smelter return (before freight)	$41.71 per ton
Net smelter return (after freight)	$38.65 per ton

In calculating the figures for net smelter return, no allowance has been made for the cost of mining and milling the ore. Losses of the various metals in the milling operation are, however, taken into account.

Respectfully submitted,

John W. Britton, P. Eng.

John W. Britton, P. Eng.

ASSAYS OF HEAD SAMPLES

		Bulk sample	D.D. Core composite sample
Gold	oz/ton	0.40	0.31
Silver	oz/ton	8.8	3.42
Copper	%	2.56	0.91
Lead	%	12.06	5.48
Zinc	%	13.21	6.78
Cadmium	%	0.05	0.04
Iron (acid-soluble)	%	13.68	N.A.
Sulphur	%	21.99	19.96
Antimony	%	0.03	N.A.
Arsenic	%	Trace	N.A.
Barium	%	Not detected	N.A.

N.A. = Not assayed.

SPECIFIC GRAVITY OF ORE

	Bulk sample	D.D. Core composite sample
Specific gravity	3.75	3.46
Cubic feet per ton (2000 lb.)	8.55	9.26

METHOD OF TREATMENT

The treatment method used was essentially the same for both samples and involved the following steps:

(1) Crushing to minus 10 mesh;
(2) Wet grinding in a ball mill;
(3) Flotation of copper and lead together;
(4) Cleaning the copper-lead concentrate;
(5) Separation of copper and lead;
(6) Flotation of zinc from the copper-lead rougher tailing;
(7) Cleaning the zinc concentrate;
(8) Flotation of pyrite from the zinc rougher tailing.

Method of Treatment (Cont'd.)

In the case of the bulk sample, a satisfactory separation of the copper and lead minerals was obtained by using dichromate as a lead depressant whilst the copper was floated from the mixed concentrate; the operation was repeated once in order to reduce the amount of lead remaining in the copper concentrate. This method was, however, only partly successful when treating the drill core sample. The best results were obtained by carrying out a first separation, using sulphur dioxide and potassium dichromate as lead depressants, followed by two stages in which the copper concentrate from the first separation was conditioned with cyanide in order to depress the copper whilst the remaining lead was floated. Unfortunately part of the galena was depressed with the chalcopyrite and the final copper concentrate was still relatively high in lead (24.57%).

The difficulty experienced in separating the chalcopyrite from the galena is believed to have been at least partly due to the presence of oil and grease in the drill core sample. On treating the lead sample and the copper and lead concentrates with acetone it was possible to isolate a greasy substance. It is probable that films of this substance on the surface of the galena had prevented its depression by sulphur dioxide and dichromate. As indicated by the much better results obtained on the bulk sample, little difficulty should be experienced when treating freshly-mined ore provided care is taken to prevent excessive contamination with oily substances. If necessary, some method of overcoming such contamination could probably be developed.

It was found that the pyrite concentrates, which were floated after the zinc, had relatively high gold contents (up to 0.44 ounces per ton). An attempt was made to recover this gold by cyaniding the concentrate. Although 70% of the gold was recovered, the cyanide consumption was excessive, due to the presence of cyanide-soluble copper minerals. It was later found when treating the drill core composite sample, that most of the gold in the pyrite could be liberated by finer grinding of the ore, thus enabling it to be recovered with the copper and lead minerals.

Summary of best results obtained on bulk sample and drill core composite sample, with probable results for estimated ore.

			Bulk Sample	Drill Core	Estimated Ore
Assay of Ore:					
Gold		Oz/ton	0.40	0.31	0.22
Silver		Oz/ton	8.8	3.4	8.17
Copper		%	2.56	0.91	0.7
Lead		%	12.06	5.48	13.00
Zinc		%	13.21	6.78	8.50
Cadmium		%	0.05	0.04	0.05 *
Copper Concentrate					
Weight		% of ore	8.02	3.40	2.1
Assays:	Au	Oz/ton	0.68	0.80	0.8
	Ag	Oz/ton	24.6	18.4	20
	Cu	%	25.45	17.29	20
	Pb	%	8.82	24.57	20
	Zn	%	4.02	2.86	4
	Fe	%		23.51	23
	Moisture	%		9.0	9
Recoveries:	Au	%	14.3	13.2	8
	Ag	%	23.5	18.5	5
	Cu	%	78.6	63.4	60
	Pb	%	6.0	15.2	3
	Zn	%	2.4	1.4	1
Lead Concentrate:					
Weight		% of ore	16.57	7.48	19.5
Assays:	Au	Oz/ton	1.21	1.64	0.8
	Ag	Oz/ton	29.1	26.8	34
	Cu	%	0.81	1.05	0.7
	Pb	%	62.43	56.59	60
	Zn	%	5.11	6.57	5
	Fe	%		10.12	10
	Moisture	%		4.6	5

Cont'd.......

* Assumed (JWB)

			Bulk Sample	Drill Core	Estimated Ore
Lead Concentrate:	(Cont'd.)				
Recoveries:	Au	%	52.6	59.5	72
	Ag	%	57.6	59.1	60
	Cu	%	5.1	8.5	20
	Pb	%	87.1	77.1	90
	Zn	%	6.4	7.1	11
Zinc Concentrate:					
Weight		% of ore	17.43	8.82	11.3
Assays:	Au	oz/ton	0.06	0.04	0.06
	Ag	oz/ton	2.4	2.1	4
	Cu	%	0.60	0.39	0.5
	Pb	%	0.26	0.41	1
	Zn	%	61.83	62.03	60
	Cd	%	0.31	0.29	0.3
	Fe	%		3.56	5
Moisture		%		8.6	9
Recoveries:	Au	%	2.7	1.7	3
	Ag	%	5.0	5.5	6
	Cu	%	4.0	3.7	8
	Pb	%	0.4	0.7	1
	Zn	%	81.1	79.5	80
	Cd	%	100 (?)	64	68

Calculation of net smelter returns

(1) Assumed Metal Prices:

		U.S. Price	Canadian Price (1)	T.N.R.P. (2)
Gold	$/oz.	35.00	37.45	N.A.
Silver	$/oz.	1.29	1.38	N.A.
Copper	¢/lb.	28.0	N.A.	N.A.
Lead	¢/lb.	11.8	12.6	9.1
Zinc	¢/lb.	12.5	13.4	10.1
Cadmium	$/lb.	N.A.	N.A.	2.50

Notes: (1) Assumed rate of exchange $1.00 U.S. = $1.07 Canadian.

(2) Tadanac Net Realised Price.

It is assumed that the copper concentrate would be shipped to Tacoma for smelting and the lead and zinc concentrates would be shipped to Trail.

(2) Assumed Freight Rates:

Copper concentrate (to Tacoma)	$14.00 per ton (dry weight)
Lead concentrate (to Trail)	$ 9.00 per ton (dry weight)
Zinc concentrate (to Trail)	$ 9.00 per ton (dry weight)

(3) Assumed Smelter Schedules:

(a) Copper Concentrate

Credits: Gold - Pay for 96.75% at the U.S. Mint price (minimum deduction 0.015 oz/ton).

Silver - Pay for 95% at Handy and Harman N.Y. quotations (minimum deduction 0.5 oz/ton)

Copper - Pay for total copper less 20 pounds per ton at the E. & M.J. Export Refinery price for electrolytic wire bars less 2.5¢/lb.

Debits: Smelter charges $13.00 (U.S.) per dry ton.

(3) Assumed Smelter Schedules: (Cont'd.)

(b) Lead Concentrate

Credits: Gold - Pay for 95% at the Royal Canadian Mint price less $1.25/oz.

Silver - Pay for 95% (minimum deduction 1oz/ton) at the E.M. & J., N. Y. price less 2¢/oz.

Lead - Pay for 92.5% (minimum deduction 20 lb/ton) at the Tadanac Net Realised Price less 0.6¢/lb.

Zinc - Pay for 47% at the T. N. R. P. less 5.5¢/lb.

Debits: Smelter charges - $11.76 per dry ton

Moisture - Up to 5% free; over 5% deduct 20¢/unit per dry ton for all units over 5%.

(c) Zinc Concentrate

Credits: Gold - Pay for 80% at the Royal Canadian mint price less $1.25/oz. (no payment if less than 0.03 oz./ton).

Silver - Pay for 80% at E. & M.J., N.Y. quotation less 2¢/oz. (minimum deduction 1 oz./ton)

Lead - Pay for 80% (minimum deduction 20 lb./ton) at the T.N.R.P. less 3.35¢/lb.

Zinc - Pay for 86% at the T.N.R.P. less 2.6¢/lb.

Cadmium - Pay for 70% (minimum deduction 5 lb./ton) at the T.N.R.P. less 40¢/lb.

Debits: Smelter Charges - $12.00 per dry ton.

Iron - Up to 5% free; 50¢/unit per dry ton on all over 5%.

Moisture - Up to 10% free; over 10% charge 10¢ per unit per dry ton for all units over 10%.

(4) Assays of Concentrates * :

	Au oz./ton	Ag oz./ton	Cu %	Pb %	Zn %	Cd %	Fe %	H_2O %
Copper concentrate	0.80	20	20	20	4	--	23	9
Lead concentrate	0.80	34	0.7	60	5	--	10	5
Zinc concentrate	0.06	4	0.5	1	60	0.3	5	9

* On dry basis.

(5) Weights of Concentrates

	Per ton of mill feed
Copper concentrate	0.021 ton
Lead concentrate	0.195 ton
Zinc concentrate	0.113 ton
Total concentrates	0.329 ton

(6) Net smelter returns per ton of concentrate

A Copper concentrate:

			$/ton of Concentrate	
Credits:				
Gold	0.9675 x 0.8 x $35.00	=	27.09	(U.S.)
Silver	0.95 x 20 x $1.29	=	24.51	"
Copper	(400 - 20) x (28.0 - 2.5)¢	=	96.90	"
Total Credits			$148.50	"
Debits:				
Smelter charge			13.00	"
Net smelter return (before freight)			$135.50	(U.S.)
Net smelter return (before freight)			144.99	(Can.)
Freight			14.00	"
Net smelter return (after freight)			$130.99	"

(6) Net smelter returns per ton of concentrate (Cont'd.)

B Lead Concentrate:

		$/ton of Concentrate	
Credits:			
Gold	0.95 x 0.8 x $(37.45 - 1.25) =	27.51	(Can.)
Silver	0.95 x 34 x $(1.38 - 0.02) =	43.93	"
Lead	0.925 x 1200 x (9.1 - 0.6)¢ =	94.35	"
Zinc	0.47 x 100 x (10.1 - 5.5)¢ =	2.16	"
Total Credits		$167.95	
Debits:			
Smelter charge		11.76	"
Net smelter return (before freight)		$156.19	"
Freight		9.00	"
Net smelter return (after freight)		$147.19	"

C Zinc concentrate:

		$/ton of Concentrate	
Credits:			
Gold	0.80 x 0.06 x $(37.45 - 1.25) =	1.74	(Can.)
Silver	(4 - 1) x $(1.38 - 0.02) =	4.08	"
Zinc	0.86 x 1200 x (10.1 - 2.6)¢ =	77.40	"
Cadmium	70 x (6 - 5) x $(2.50 - 0.40) =	1.47	"
Total Credits		$ 84.69	
Debits:			
Smelter charge		12.00	"
Net smelter return (before freight)		$ 72.69	"
Freight		9.00	"
Net smelter return (after freight)		$ 63.69	"

(7) Net smelter returns per ton of ore

A. Before freight:

Concentrate	Ton of Concentrate per ton of ore	Net smelter return per ton of concentrate $ (Canadian)	Net smelter return per ton of ore $ (Canadian)
Copper	0.021	144.99	3.04
Lead	0.195	156.19	30.46
Zinc	0.113	72.69	8.21
Total Concentrates	0.329	--	41.71

B. After freight:

Concentrate	Tons of Concentrate per ton of ore	Net smelter return per ton of concentrate $ (Canadian)	Net smelter return per ton of ore. $ (Canadian)
Copper	0.021	130.99	2.75
Lead	0.195	147.19	28.70
Zinc	0.113	63.69	7.20
Total Concentrates	0.329	--	38.65

(8) Gross value of estimated ore

Metal	Assay	oz. or Lb/ton	Price per oz. or lb.	Value per ton of ore $ (Canadian)
Gold	0.22oz/ton	0.22 oz.	$37.45 (Can.)	8.24
Silver	8.17oz/ton	8.17 oz.	$ 1.38 "	11.27
Copper	0.7%	14.0 lb.	31.5¢ "	4.41
Lead	13.00%	260.0 lb.	12.6¢ "	32.76
Zinc	8.50%	170.0 lb.	13.4¢ "	22.78
Cadmium	0.05% *	1,0 lb.	$ 3.00 "	3.00
Total	--	--	--	82.46

* Assumed (J. other assays supplied by Sullivan, P. Eng.

(9) Ratio of net smelter return to gross value of ore

A Before freight: $\frac{41.71}{82.46}$ = 50.6%

B After freight: $\frac{38.65}{82.46}$ = 46.9%

Note: The above calculations of net smelter returns do not include the cost of mining and milling the ore.

APPENDIX

Drill core compositing data

D.D.H. No.	Footage	Length Feet	Weight taken Grams	Grams per Foot
1	40.2 - 41.8	1.6	213	133
"	41.8 - 45.0	3.2	688	215
2	44.0 - 48.0	4.0	545	136
"	48.0 - 53.8	5.8	799	138
3 *	--	--	Nil *	--
4	66.0 - 72.0	6.0	787	131
"	72.0 - 79.0	7.0	534	76
"	79.0 - 87.3	8.3	468	56
5	67.8 - 74.0	6.2	710	115
"	74.0 - 77.5	3.5	368	105
"	77.5 - 80.3	2.8	252	90
6	78.0 - 88.0	10.0	198	20
"	88.0 - 93.0	5.0	66	13
"	93.0 - 98.5	5.5	441	80
"	98.5 -103.5	5.0	845	169
Total	--	73.9	6914	94

* Sample from hole 3, 38.7 - 39.3' taken for assay only:

Gold	1.47 oz/ton	Lead	0.05%
Silver	0.99 oz/ton	Zinc	0.15%
Copper	0.04%	Sulphur	46.88%
		Pyrite (calc.)	87.5%

A P P E N D I X 111

AVERAGE OF ASSAYS

AVERAGE OF ASSAYS

The following calculations are based on the surface assay plan published by American Lead Silver Mines Ltd., November 30, 1948. The plan is a compilation of samples by:

A. G. Langley, P. Eng., M.E.
B. T. O'Grady, P. Eng., M.E.
H. S. Sargent, P. Eng., M.E.
B. Brynelsen, P. Eng., M.E.
G. C. Rutherford, P. Eng., M.E.
A. M. Richmond, P. Eng., M.E.
N. E. Nelson, P. Eng., M.E.

(Map Sheet No. 5)

1. Dunbar Vein:

W	WxAu	WxAg	WxPb	WxZn
4.0	0.24	22.00	56.00	48.00
5.0	0.80	92.50	139.00	46.50
2.3	0.46	56.58	78.20	6.44
8.0	1.28	142.56	261.60	70.40
5.0	0.80	97.00	160.00	13.00
24.3	3.58	410.64	694.80	184.34

Weighted Averages:

Au = 0.147o/T
Ag = 16.89o/T
Pb = 28.59%
Zn = 7.58%

2. West Vein:

W	WxAu	WxAg	WxPb	WxZn
2.8	0.56	23.00	43.40	-
1.5	0.30	27.00	47.10	8.40
6.0	0.54	36.00	85.20	51.00
2.6	0.21	36.40	80.60	19.76
4.0	0.08	28.00	51.60	7.20
16.9	1.69	151.20	307.90	86.36

Weighted Averages:

Au = 0.10 o/T
Ag = 8.94 o/T
Pb = 18.21 %
Zn = 5.11%

3. <u>East Vein:</u>

W	WxAu	WxAg	WxPb	WxZn
3.3	0.33	51.15	93.39	98.01
4.0	0.64	43.20	92.00	20.80
5.5	1.59	96.80	172.15	39.60
3.5	2.80	41.30	67.20	9.10
2.6	0.10	5.98	23.40	74.88?
4.0	1.28	25.60	40.80	94.40
2.0	0.56	49.60	65.60	22.00
2.0	0.12	7.70	10.60	2.60
3.5	0.32	33.25	44.80	17.15
3.8	0.42	125.02	203.60	8.74
4.7	1.13	94.94	147.11	18.80
38.9	9.29	574.54	960.73	406.08

<u>Weighted Averages:</u>

Au = 0.238 o/T
Ag = 14.77 o/T
Pb = 24.70 o/T
Zn = 10.44 %

4. <u>Big Showing:</u>

W	WxAu	WxAg	WxPb	WxZn
4.7	1.13	94.94	147.11	18.80
10.0	2.00	73.00	196.00	16.00
13.0	10.40	72.41	101.00	133.90
16.0	4.16	129.60	206.40	113.60
9.3	0.28	9.30	130.20	20.46
2.0	0.24	21.60	60.00	38.40
55.0	18.21	400.85	840.71	341.16

<u>Weighted Averages:</u>

Au = 0.33 o/T
Ag = 7.29 o/T
Pb = 15.29 %
Zn = 6.20 %

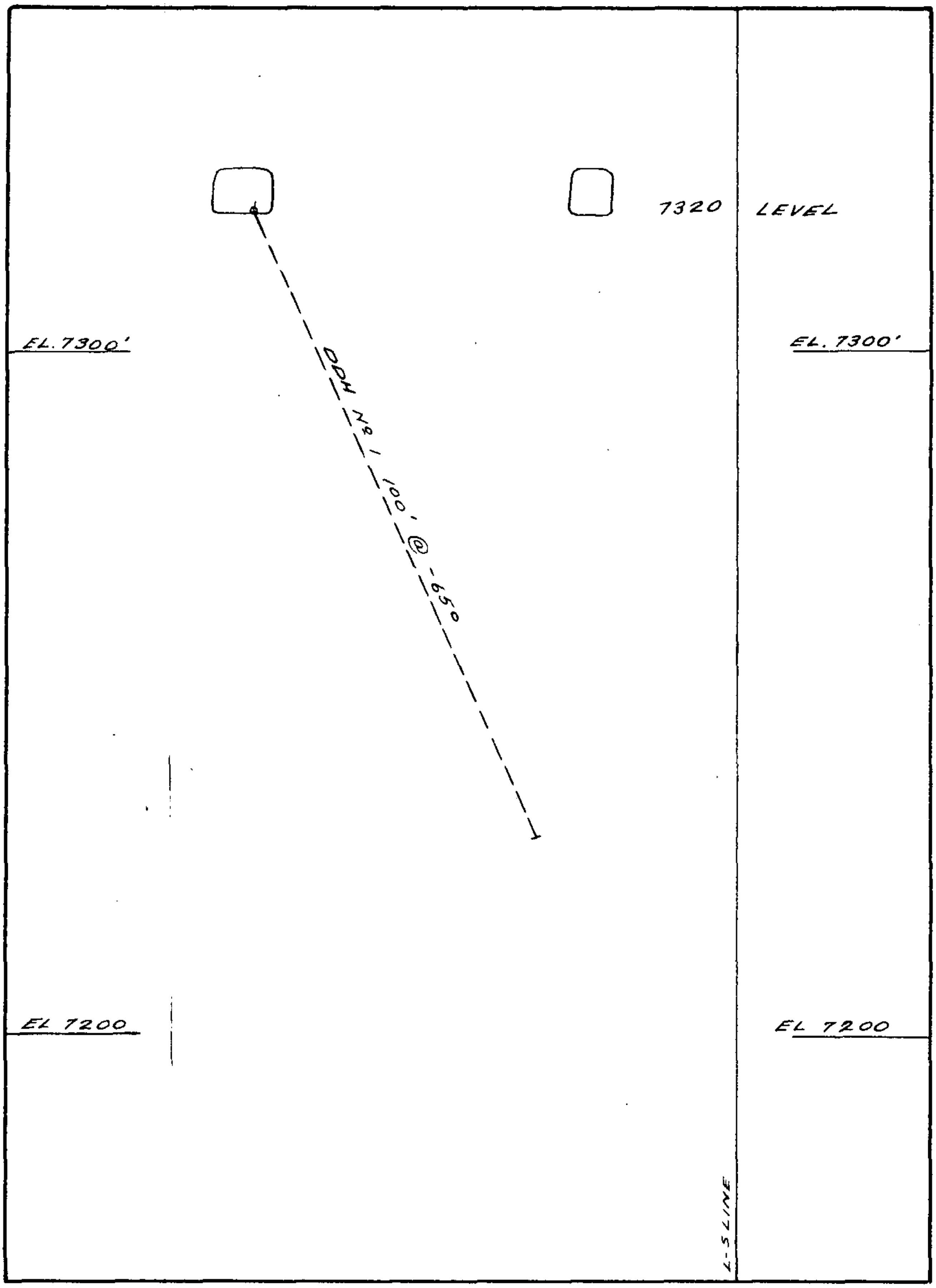
The following calculations are based on the diamond drill and underground samples taken by J. Sullivan during the 1963 field season. (See drill sections and Map Sheet U-1)

1. **D. D. Holes:**

Hole No.	W	WxAu	WxAg	WxPb	WxZn
1	3.2	0.54	34.72	52.54	66.30
2	4.0	0.32	16.00	16.00	3.40
2	5.8	2.32	19.14	9.74	0.12
4	6.0	3.60	69.60	103.38	20.52
4	7.0	0.70	54.60	89.25	33.60
4	8.3	0.83	24.48	30.54	3.15
5	6.2	0.74	29.14	21.70	103.04
5	3.5	0.70	11.20	6.44	32.38
5	2.8	0.28	10.36	11.98	20.22
6	10.0	0.80	29.00	14.00	21.50
6	5.0	0.03	8.00	1.65	0.30
6	5.5	2.86	31.35	35.75	39.43
6	5.0	0.70	17.25	10.15	51.75
6	5.8	0.03	13.20	4.40	1.27
	77.8'	14.45	368.04	408.32	396.98

2. **Underground:**

W	WxAu	WxAg	WxPb	WxZn
8.0	0.64	32.00	22.96	88.96
1.0	0.32	12.00	10.20	16.28
6.6	0.58	31.02	36.83	34.65
4.3	0.86	59.34	95.33	122.68
8.0	1.44	38.40	42.56	222.40
4.7	1.32	24.91	26.65	140.62
6.4	4.61	36.48	33.92	70.14
7.5	0.15	29.25	23.85	41.40
3.7	0.04	6.29	7.03	7.92
5.8	0.35	25.23	17.98	28.24
6.0	1.68	46.20	46.92	142.32
62.0'	12.00	341.22	364.23	915.61



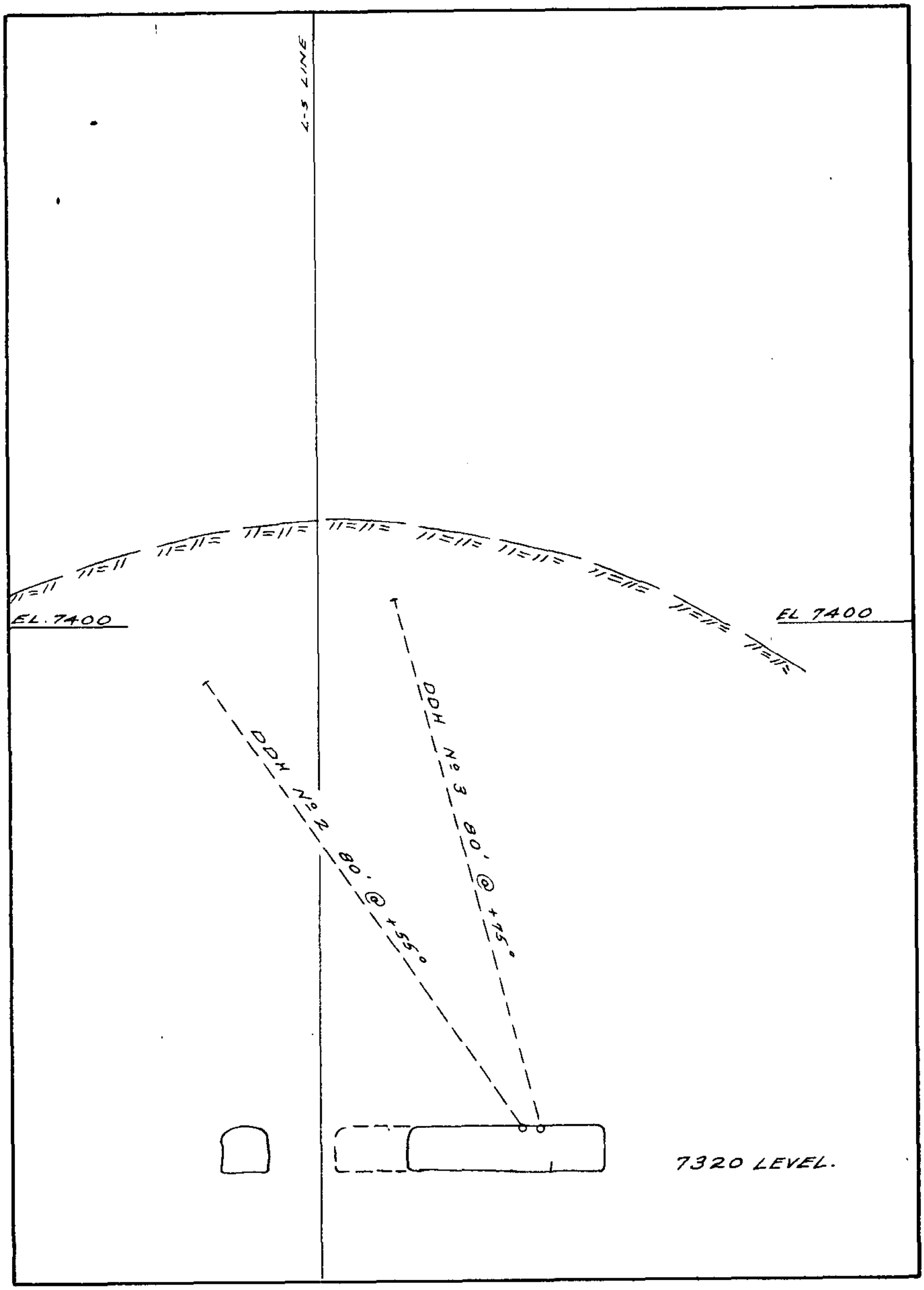
TEDDY GLACIER PROPERTY
PROPOSED DRILL SECTION "A"
BEARING N 38° E
MAP SHEET Nº U-1

A P P E N D I X IV

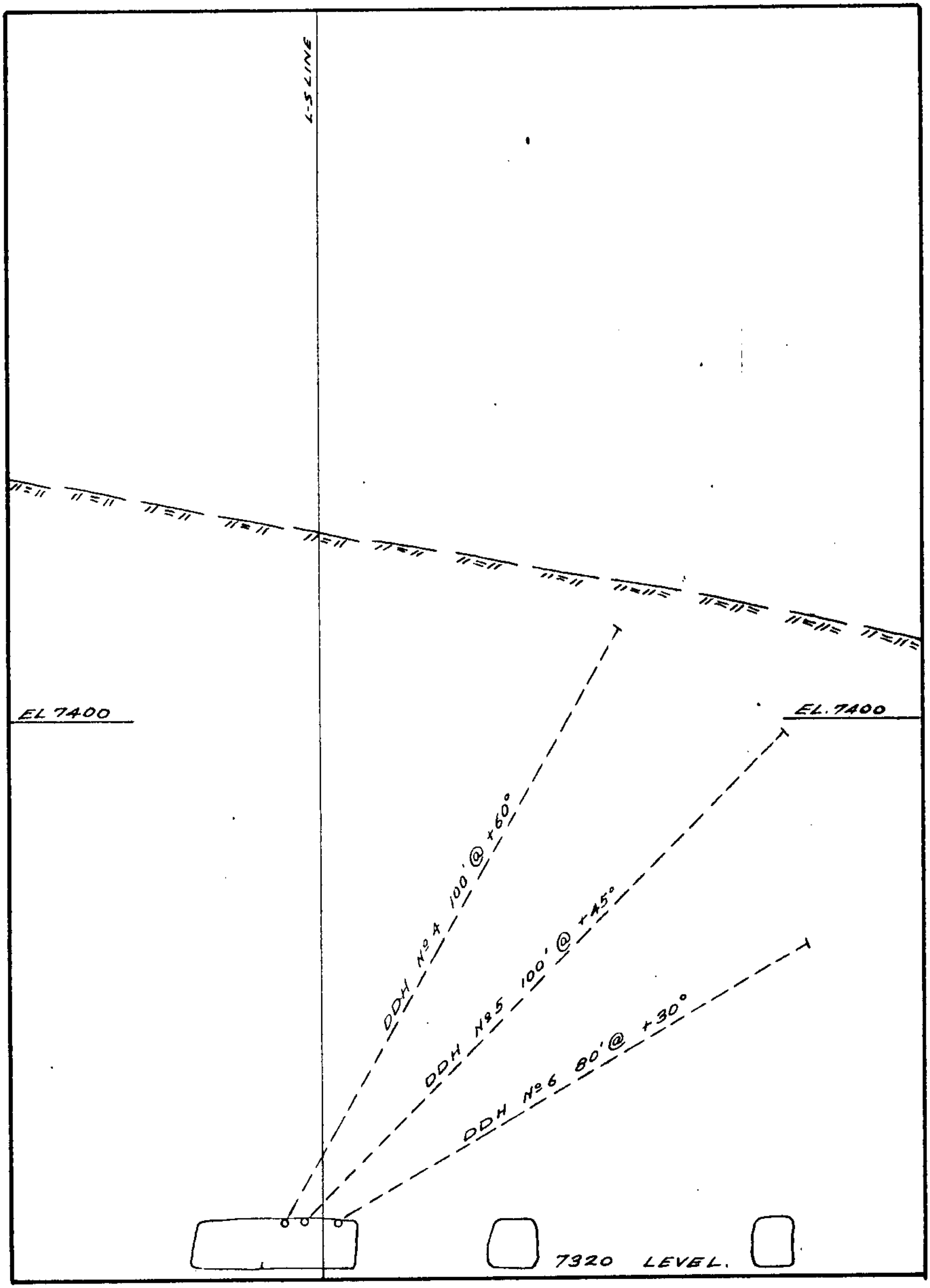
DIAMOND DRILLING 1964

UNDERGROUND PORTION

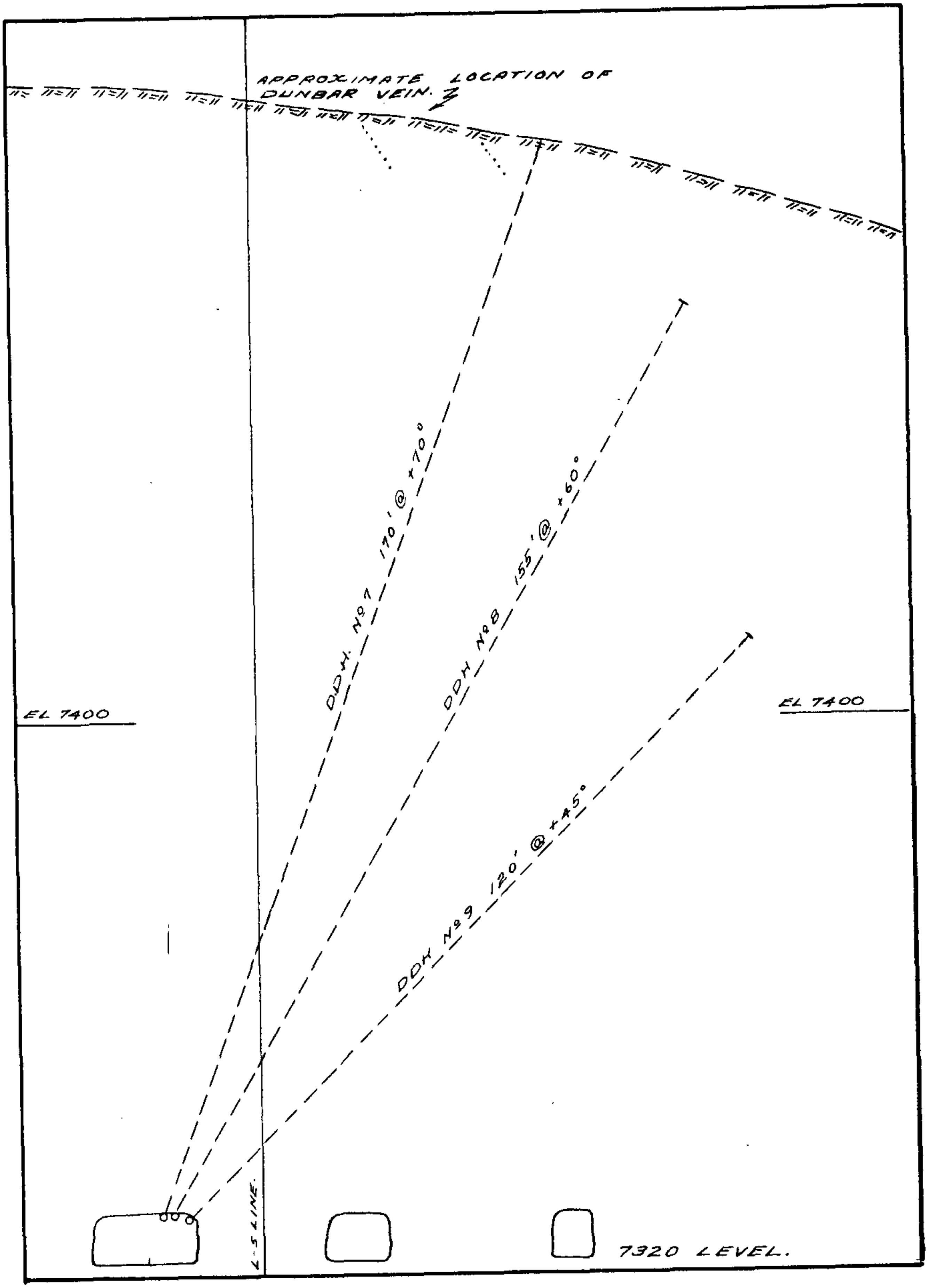
PROPOSED SECTIONS



TEDDY GLACIER PROPERTY.
PROPOSED DRILL SECTION "B"
BEARING S 38° W.
MAP SHEET Nº U-1.



TEDDY GLACIER PROPERTY.
PROPOSED DRILL SECTION "C"
BEARING N 38° E
MAP SHEET Nº U-1.



TEDDY GLACIER PROPERTY.
PROPOSED DRILL SECTION "D"
BEARING N 38°E.
MAP SHEET Nº U-1.