92H-11(W½)

810491

on behalf of

CAROLIN MINES LTD.

by

D. R. Cochrane, P. Eng.
June 21, 1974
Delta, B.C.

Report on the Drilling
of the

IDAHO ZONE

Situated on Ladner Creek

10 air miles northeast of
Hope
New Westminster M. D.
Southern British Columbia

on behalf of

CAROLIN MINES LTD.

by

D. R. Cochrane, P. Eng.
June 21, 1974
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with appended reports by

(a) Dr. J. H. Montgomery on the Petrology of the Idaho Zones
(b) Dr. A. J. Sinclair, Polished Section Study of Idaho Mineralization
(c) J. Britton (Britton Reasearch) on a Mill Test of the Idaho Upper Zone

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geology
geophysics
geochemistry

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PART A - SUMMARY AND CONCLUSIONS

A-1 INTRODUCTION

This report describes Carolin Mines "Idaho" gold zone, a series of auriferous sulphide-arsenide bands situated approximately ten (10) air miles north-northeast of Hope in Southern British Columbia. Carolin optioned the "core" group of thirty-six (36) claims and eight (8) crown grants early in 1973 from Summit Mining, and shortly thereafter added an additional group of thirty-four (34) claims. The claims cover two former producing gold mines, the Aurum and the Pipestem (or Home Gold Mine), and, in addition cover several gold "showings" which as yet are largely unexplored. Re-evaluation of this property commenced early in 1973 and was prompted by the increase in the price of gold. Work soon focused on the Idaho Zone, which is an "old" showing, drilled in the late 40's and then trenched and resampled in the early 60's. This previous work indicated that a moderate sized deposit of close to 0.20 oz. Au per ton was possible, and that additional work was certainly warranted. Since 1973, Carolin has completed nineteen B.Q.W.L. drill holes for an aggregate total of just over 7000 lineal feet, and the mineralized bands are still open in several directions.

This report summarizes the data available as of the report date on the Idaho Zone, and was written by an independent (see appended certificate) consulting geologist who has been engaged "first hand" on the project since it's inception in early 1973. Reports by other independent consultants are also appended and these include reports by Britton Research (mill testing), Dr. J. H. Montgomery (Petrology) and Dr. A. J. Sinclair (polished section work). Drill holes and surface mapping of the property was completed by Mr. D. Griffith (B. Sc. Geology) and the field work and all important "day to day" activities were managed by Mr. M. Mathieu of Carolin Mines.

A-2 SUMMARY AND CONCLUSIONS

1. The southern portion of the seventy eight (78) claim "Aurum/Idaho/Pipestem" property of Carolin Mines is accessible by 4x4 truck and is centered approximately 20 road miles northeast of the town of Hope in Southern British Columbia. Road access from Hope is via the Coquihalla and the Ladner Creek road.

2. The property lies in the northern portion of the Cascade Mountains, which is a moderately rugged upland belt characterized by deeply incised, well forested valleys. The Idaho Zone is centered at 3500 feet above sea level on the southwest sloping flank of an unnamed ridge which rises to just over 5000 feet.

3. The claims straddle a northerly trending bedrock sequence which includes; a western area of Paleozoic Hozameen

Group metasediments; a narrow steeply dipping serpentine band bounded on the east by the Hozameen fault; and an eastern area underlain by the Jurassic Ladner State Group. The Idaho Zone lies a few hundred feet east of the serpentine belt within Ladner Group (?) slates, argillites and greywackes.

4. The results of the '73 geochemical soil sampling and reconnaissance geological work have been discussed in an earlier report (Cochrane, Griffith, March 1, 1974 and others). This report deals exclusively with a summary of the diamond drill program and assay results of split core on the Idaho Zone.

5. Diamond drilling and surface sampling has revealed the presence of a series of moderate to "low grade" (depending on cut off used) auriferous zones consisting of pyrrhotite, pyrite, arsenopyrite and chalcopyrite in an albite-carbonate-silica host rock.

6. These bands dip at shallow to moderate angles northerly and range from a few feet to just over 100 feet thick. To date two main zones have been intersected in drilling, namely the Upper and Lower Idaho Zones. They are separated by 50 to 130 feet of relatively barren slate. The lower zone has just recently been discovered and deepening of earlier drill holes is now necessary.

7. Drilling has been conducted on a modified 50 foot grid pattern; the modifications necessary because of the difficulty in accessing drill sites. Drilling and trenching has been conducted within an area 350 feet wide by 400 feet long, and the mineralized bands are open, and presumed to continue northerly.

8. Petrographic, Mill Test and Polished section reports accompany this report on the drilling and appendex C, D and E respectively.

9. J. W. Britton reported "The results showed that the sample of ore was amenable to direct cyanidation, giving an extraction of 92.2% after grinding to 95% minus 200 mesh. It would, however, almost certainly be more profitable to pre-concentrate the ore by flotation, followed by cyanidation of the reground concentrate, the flotation tailing being discarded. In this way, an overall recovery of about 88% of the gold can be expected."

10. The "Upper" Idaho Zone grade and tonnage estimate (to and including hole 19) is just in excess of 550,000 diamond drill indicated tons averaging 0.092 troy ounces of gold per short ton. The lower zone estimate is 150,000 diamond drill indicated tons averaging 0.085 troy ounces of gold per short ton. As previously mentioned, both zones are open in several

directions and there is no geological evidence at this point to suggest an abrupt termination of the zones to the north.

11. There is of course considerable additional reserves of lower grade material, as, for example, the Lower Zone of drill hole OG19 which cut 434 feet of alternating bands of good, then poorly mineralized rock which averaged approximately 0.03 Au/ton over the entire length.

12. The present data indicates the "Idaho" deserves serious and uninterupted economic consideration, and the author strongly recommends that the drilling program continue in order to determine the limits of the zones, and to search for additional (sub)parallel ones. Surface exploration should also be conducted to the north as several "showings" and high rock sample assays suggest that additional zones of economic interest may exist and possibly overly the Upper Idaho Zone. If such is the case, the contingency of large scale open pit mining of the Idaho Zones should be entertained.

Respectfully submitted,

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
D. R. COCHRANE

D. R. Cochrane, P. Eng.,
June 21, 1974.

PART B: SETTING

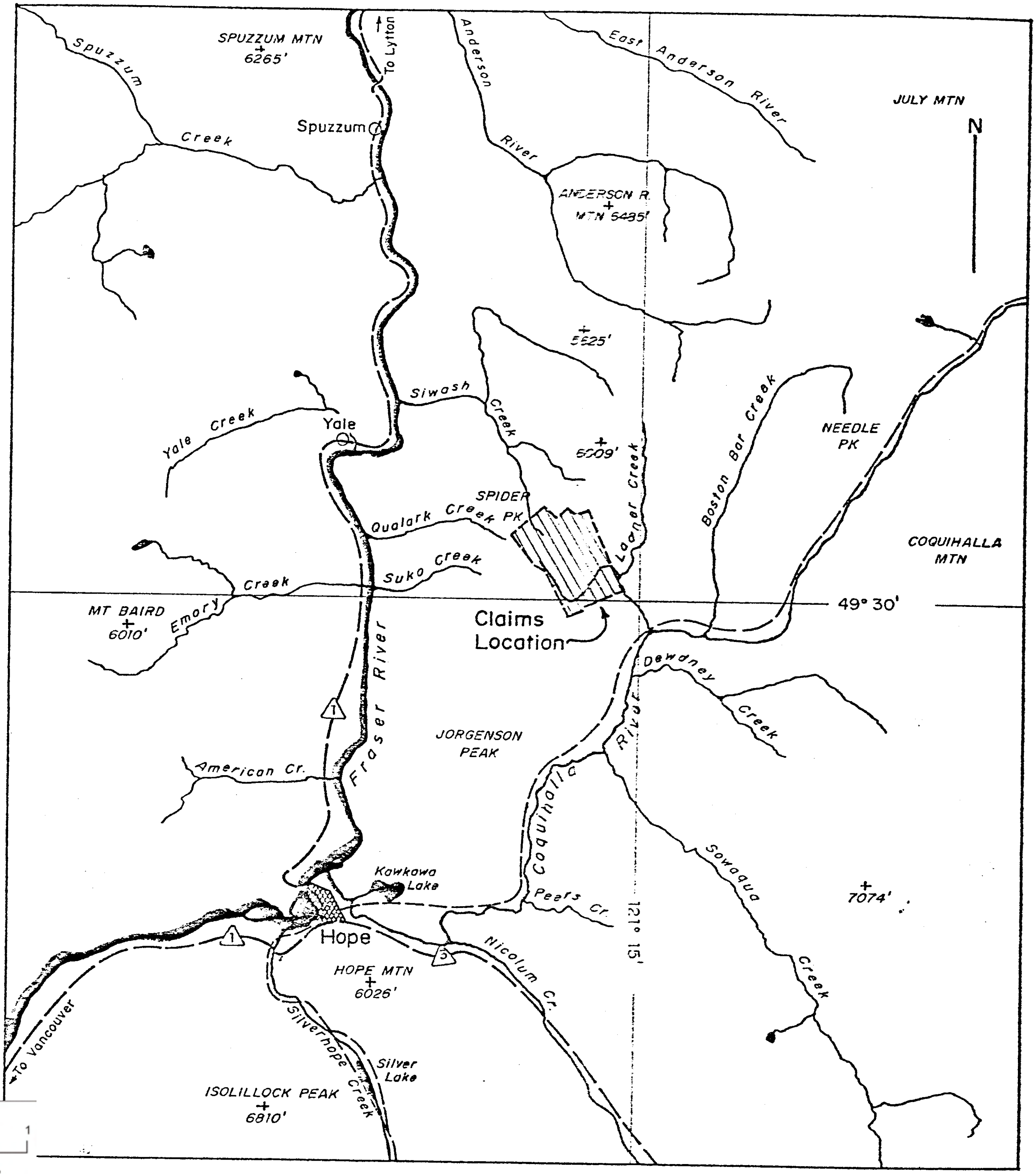
B-1 LOCATION AND ACCESS

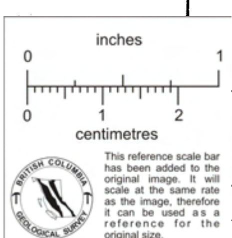
The property is situated in the northern portion of the Cascade Mountains, some 12 air miles northeast of the town of Hope, in southern British Columbia. Most of the claims lie north of the "South West Fork" of Ladner Creek, and west of the Main Fork. Facile road access to the southern claims sector is provided by a logging-mining road which proceeds northerly then westerly from the Coquihalla River road at a point near mile 18 and close to "Jessica" a station on the now abandoned West Kettle Railway Line. The access road is easily passable by car or truck in the summer months, but is often "washed out" in the freshet seasons. Access to the northernmost claims is by pack trail or by charter helicopter and the latter is available from a base in Hope.

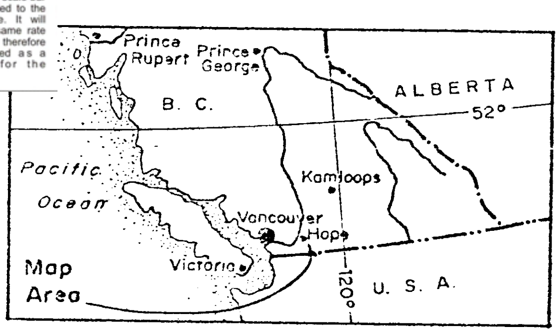
The National Topographic System Code for the claims area is 92 H/11 (west ½): the latitude is 49°30'N, and longitude 121°15'W. (see Figure 1)

B-2 PROPERTY

Carolin Mines Ltd., with an office at 811-850 West Hastings in Vancouver, B.C. optioned eight Crown granted claims and thirty-six (36) located claims and fractions from Summit Mining Co. of Hope, B.C. In addition, Carolin owns outright thirty-four (34) adjoining







LOCATION MAP Fig. 1

carolin mines limited

Aurum / Idaho / Pipestem Project
Coquihalla River / Ladner Creek Area
New Westminster Mining Division, B. C.

Cochrane Consultants Limited

N. T. S. 92 H / 11 west half

Scale 1:250,000

"CARO" claims and fractions. The claims form a contiguous block situated in the New Westminster Mining Division and are shown on B. C. Department of Mines claim Map 92H/11W (M). (Also see Figure 2)

The following tables list pertinent claims information.

TABLE A OPTIONED MINERAL CLAIMS

1. Crown Grants

Name	Lot No.	Acres	Expiry Dates
Aurum #1 to #6 incl.	1236 to 1241	227.52	July 2, 1979
Idaho	1234	40.75	" "
Tramway	1235	51.15	" "

2. Located Claims and Fractions

Name	Record No.(s)	Expiry Date
Home Gold #1 to #15	14723-14737	August 21, 1982
Gold Star #1 to #4	11365-11368	July 28, 1980
Cabin #1 to #14	11903-11916	July 21, 1980
Cabin #20 FR	11917	July 21, 1980
Cabin #21 FR	11918	July 21, 1980
Sylvia FR	11364	July 20, 1980

TABLE B CAROLIN MINES - list of claims and fractions owned outright

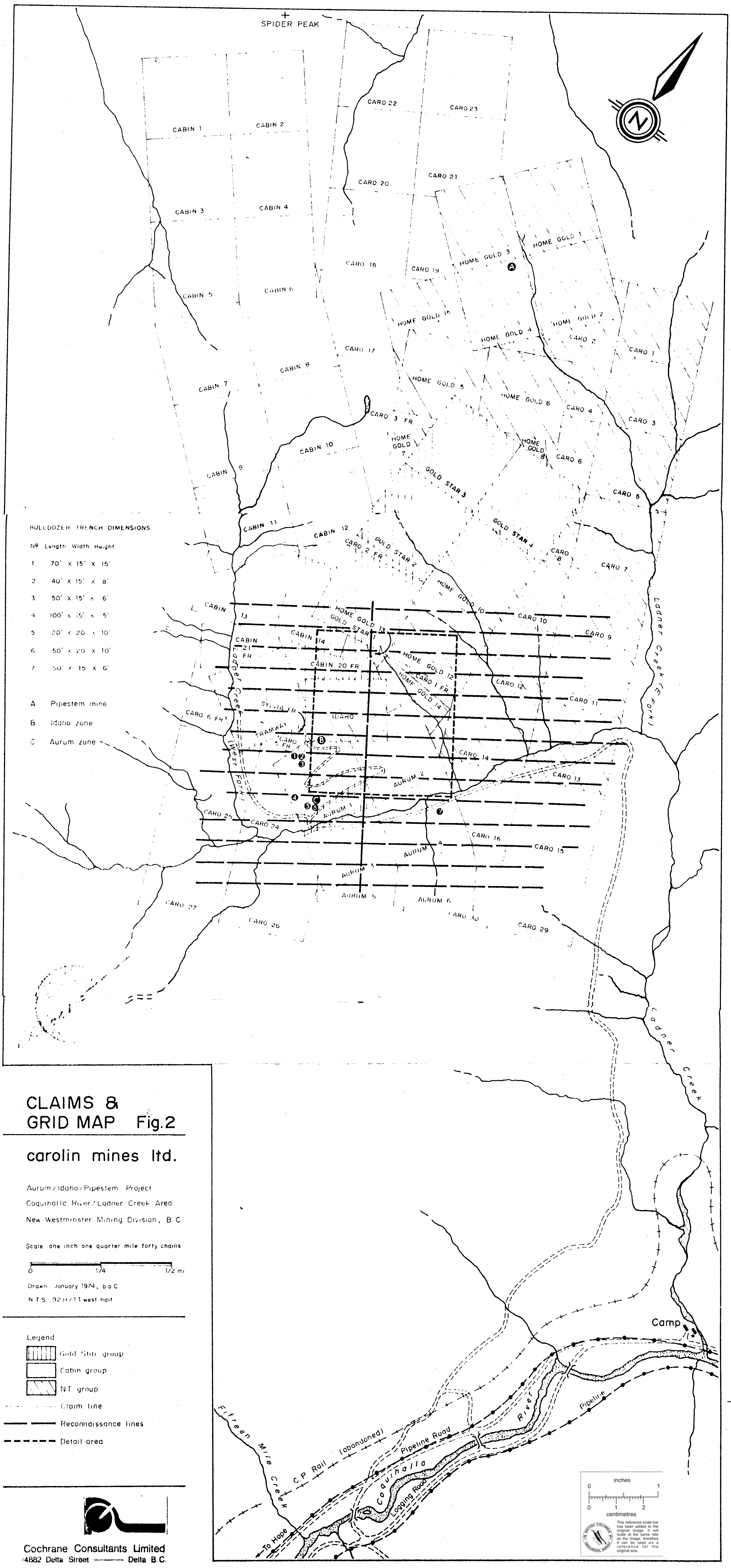
Name(s)	Record No. (s)	Expiry Date
CARO #1 to #27	28614-28640	June 29, 1980
CARO #29 to #30	28641-28642	" "

CARO #1, #2, #3 FR	28643-28645	June 29, 1980
CARO #5 FR	28646	" "
CARO #6 FR	28647	" "

B-3 SETTING

The claims are situated in the Cascade Mountains of southwestern British Columbia, a region characterized by rugged peaks and dense, heavy forest cover. Elevations on the property vary from just over 2500 feet to over 4500 feet above sea level, and the flank of the unnamed peak on which the Aurum and Idaho Zones are located rises some 2000 feet in a little over half a mile. The creeks and tributaries are deeply incised, and in general, there is only scattered outcrops, since much of the area is covered by a widespread but relatively thin mantle of drift.

The west side of the claim group covers a narrow serpentine band which extends north-northwesterly from Mt. Dewdney (20 miles north of the Canada-U.S.A. border) to Boston Bar, a distance of 40 miles. Parallel to, and on the east side of the serpentine band lies the Jurassic/Cretaceous Ladner Group, composed chiefly of slate, greywacke, schist and conglomerate. The Ladner Group and adjacent serpentine band is host to numerous gold occurrences, most of which are quartz veins carrying free gold and/or auriferous arsenide-sulphide zones.



CLAIMS & GRID MAP Fig.2

carolin mines ltd.

Aurum/Idaho/Pipestem Project
Coquihalla River/Ladner Creek Area
New Westminster Mining Division, B.C.

Scale one inch one quarter mile forty chains

0 — 1/4 — 1/2 mi

Drawn January 1974, b.a.C.
N.T.S. 92 H/11 west half

BULLDOZER TRENCH DIMENSIONS

Nº	Length	Width	Height
1.	70'	15'	15'
2.	40'	15'	8'
3.	50'	15'	6'
4.	100'	15'	5'
5.	20'	20'	10'
6.	50'	20'	10'
7.	50'	15'	6'

A Pipestem mine
B Idaho zone
C Aurum zone

Legend
- Gold Star group
- Cabin group
- NT group
- Claim line
- Reconnaissance lines
- Detail area

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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.

is host to numerous gold occurrences, most of which are quartz veins carrying free gold and/or auriferous arsenide-sulphide zones.

B-4 HISTORY

The discovery of the Ladner Slate gold belt followed the intense placer activity along the Fraser River in the 1850s, and intermittent placer work was conducted in the Coquihalla region until the early 1900s. In 1915, considerable work was being conducted on quartz veins and stringers in the Ladner slates; however, early in 1928 rich gold ore was discovered at the "Aurum" in a talc sheer zone along the northeast contact of the serpentine body. Spectacular hand specimens were recovered at the Aurum and this precipitated a staking rush that culminated in the discovery of a number of additional gold bearing zones along the serpentine band. None of the latter occurrences produced the free gold specimens as spectacular as the Aurum specimens however. Between 1916 and 1943, five properties in the Coquihalla Gold belt produced ore amounting to 3102 tons containing 3912 oz. of gold, an average of 1.2 oz. of gold per ton. (B.C. Dept. of Mines Records).

Two former producers are now covered by the Carolin Mines claims and these are the Aurum and Pipestem, the latter also known as the "Home Gold". The Aurum Mine covered much of the ground originally occupied by both the Idaho and Snowstorm Groups, and these were consolidated in 1926.

Recorded Production

Mine	Dates	Tons	oz. Au	Average Grade
Aurum	1930-42	545	533	0.98 oz. Au/ton
Pipestem (Home Gold)	1935-37	1650	272	0.165 oz. Au/ton

Both the Aurum and Pipestem also produced small quantities of silver.

Surface and underground work on the Aurum and adjacent Idaho claims proceeded intermittently until the second world war at which time there existed some 2500 lineal feet of underground workings in seven adits. Several of the adit levels on the Aurum were interconnected by stopes and raises. In 1945 and 1946 eight shallow diamond drill holes were collared in the Idaho zone and a report by E. E. Mason described the results as including ..."fifteen intersections... in widths from 9 to 46 feet or an arithmetical average of 17.7 feet. These yielded an average grade weighted by core length of 0.171 oz. gold per ton."

Work on the Pipestem Mine situated about 2½ "trail" miles north of the Aurum continued intermittently from early in the 1920s until suspension of mining in 1937. The 1936 Minister of Mines Report describes quite extensive underground workings including a total of four adit levels, a shaft and several stopes and raises. The Pipestem ore was reported to average $15/ton (1935 prices) and ore consisted of irregular quartz stringers containing pyrite and arsenopyrite. To the author's knowledge no work has been conducted on the Pipestem since the second world war.

The Idaho-Aurum zones were again tested, this time by trenching by Summit Mining in 1966 and W.M. Sharp reports on 8000 lineal feet of continuous side-hill earth and rock cut. Sharp states "The average grade of the Idaho zone mineralization, as computed from the aforementioned drilling program (i.e. Mason, 1947), and excluding four indeterminate holes, is 0.176 oz. Au/ton. The currently delineated strike length is 250 feet; the dip length is also in the 250 foot range. The writers (i.e. Sharp) sampling generally confirms this grade estimate."

B-5 1973 EXPLORATION PROGRAM

Early in August a camp consisting of two large trailers was assembled near the confluence of Ladner Creek and the Coquihalla River and this camp served as the field exploration base for the ensuing work on the Aurum and Idaho zones. Shortly thereafter, approximately 23 miles of grid lines were completed by blazing and flagging stations at 50 foot intervals. Cross lines run northeasterly from a central base line and within the detailed coverage area lines are spaced 100 feet apart. In the reconnaissance coverage area cross lines are spaced 400 feet apart. Figure 2 shows the grid layout of lines in relation to claims and topographic features.

Geochemical orientation work revealed that the "B" soil horizon provided good geochemical contrast, and soil sampling over the grid area at 50 foot intervals was completed in September. The soil samples were dried at the field camp and then shipped to Min-En Labs of North Vancouver where they were analyzed by an atomic absorption method for their content in gold (parts per million). The results are shown on maps accompanying previous reports.

Running concurrently with the geochemical program, was a magnetometer survey, and an MF-1 (fluxgate) was used as a permanent base recorder, and an MF-2 was utilized in the field. Magnetometer stations were established at 50 foot intervals and all readings were time drift corrected for diernal fluctuations.

Early in September, Mr. D. Griffith (B.Sc. Geology) commenced geological mapping and sampling and particular emphasis was placed on the Idaho zone. During this same period, several trenches were excavated for geological and assay information, and the underground workings on the Aurum were opened, renovated, walls washed and accessible workings were geologically inspected.

The final phase of the program is the diamond drilling which commenced late in September and has continued with only a few short interruptions to the present. Drill core is logged at camp by Mr. D. Griffith, and split, numbered and crated for shipment and assay to Bondar Clegg in Vancouver, B. C. Most recently, every tenth sample is crushed, pulverized and split at camp, one-half going to Bondar Clegg and the remainder to Loring Labs in Calgary for control assay.

PART C GEOLOGY AND MINERALIZATION-IDAHO ZONE

C-1 GENERAL GEOLOGY

The geology of the area has been described in various G.S.C. publications including Summary Report, 1919 Part B; 1920 Part A; Memoir 139 "Coquihalla River Area" (1924); and Summary Report, 1929 Part A.

The Carolin claims straddle the north trending Coquihalla Serpentine Belt, which lies between the Paleozoic Cache Creek series on the west, and the upper Jurassic Ladner slates on the east. Cache Creek rocks consist chiefly of interbedded cherts and shales and the Ladner Group is predominantly slates and argillites. The Idaho zone lies within what is believed to be the Lower Ladner Group and the regional trend of the host rocks is northwesterly. Rock units dip at steep to moderate angles northeasterly. The serpentine contact is some 500 to 750 feet west of the Idaho Zone, and this contact is nearly vertical. The host rocks of the Idaho Zones are greywackes, argillites and slates which have been variously albitized, carbonatized and silicified. The upper Idaho mineralized band appears to conform to the general bedding attitude.

C-2 MINERALIZATION

In the Idaho Zone anomalously high gold values are found in silicified albite-carbonate bands which contain up to 20% by volume sulphides and arsenides. Normally pyrrhotite and pyrite are the most abundant metallic minerals followed by arsenopyrite and chalcopyrite. Visible native gold is not common.

The sulphides and arsenides occur in streaks and bands within the altered fractured host rocks as well as in disseminations for some distance on each side of the well mineralized zones. Gold values are moderately uniform and the gold appears to be associated with pyrite (see report by A. J. Sinclair).

A generalized and simplified vertical geological section through the Idaho may be described as follows:

(a) non mineralized argillite

(b) grey green greywacke, becoming fractured and chloritic

(c) upper mineralized Idaho Zone of albite-carbonite rock

(d) black to grey (graphitic) slates

(e) lower mineralized Idaho Zone

(f) greywacke to conglomerate

Mineralization is best classified as A. H. Lang's "E" type of ore, that is "sulphide replacement---consisting chiefly of auriferous arsenopyrite, pyrite and pyrrhotite."

Silver values are generally low (i.e., less than 1 oz. Ag/ton) and copper is often present in the 0.01 to 0.03 % range within the auriferous zones.

PART D DRILLING RESULTS AND DISCUSSION

D-1 DIAMOND DRILLING

A total of nineteen holes have been collared to date for a lineal total of just over 7000 feet. Drilling has been conducted on a modified 50 foot step out grid system. All drilling is B.Q.W.L. and recovery of core is excellent. The drill holes complete with their respective attitudes and locations are listed in table A following.

TABLE A

IDAHO ZONE '73-74 DRILLING
(as of June 15, 1974)

	LOCATION*	BEARING(T.)	INCLINATION	EL.COLLAR	DEPTH
OG 1	33+55N; 0+98E	234	-35	3302	228.3
OG 2	" "	234	-60	3302	267.0
OG 3	33+00N; 2+35E	235	-40	3229	301.0
OG 4	" "	235	-65	3229	305.0
OG 5	32+35N; 2+85E	235	-40	3173	297.0
OG 6	34+30N; 0+80E	220	-55	3379	336.0
OG 7	abandoned	234	-30	3270	30.0
OG 8	34+80N; 1+40E	90	-75	3381	267.3
OG 9	33+80N; 1+50E	320	-70	3290	155.6
OG 10	34+30N; 0+80E	--	-90	3379**	223.0
OG 11	34+80N; 1+40E	270	-80	3388	438.0
OG 12	34+82N; 0+40E	--	-90	3420	266.0
OG 13	34+82N; 0+40E	270°	-75	3420	338.0
OG 14	35+00N; 1+95E	--	-90	3390**	445.0
OG 15	35+00N; 1+95E	90	-77	3390**	620.0
OG 16	35+00N; 1+95E	270°	-77	3390**	638.0
OG 17	35+75N; 0+70E	--	-90	3488	557.0
OG 18	35+75N; 0+70E	270	-81	3488	528.0
OG 19	36+20N; 0+95E	--	-90	3474	945.
					7,185.2

* Relative to 1973 Grid

** Estimated due to snow

D-2 SUMMARY OF ASSAY RESULTS

Analysis of the split core samples was conducted by Bondar Clegg and Co. Ltd., in Vancouver. They use a standard fire assay technique with a final atomic absorption determination at the end of the assay sequence. Check samples of the first two holes were run by Warnock Hersey of Vancouver. Recent check assaying of every tenth sample has been done by Loring Labs in Calgary (control assays). The checks are listed in the assay sheets and in general the correlation may be described as moderate to good.

Frequency analysis of the assay data was conducted in order to determine if a statistical "cut off grade" could be established. The assays were normalized in order to eliminate sampling bias in the mineralized zones. The results showed that a "cut off grade" of 0.050 would be statistically sound. (Note: cut off of 0.050 is herein defined as assays "at" and "below" this threshold value not included in the average.) Consequently core length weighted average were calculated for

(a) cut off grade of 0.050

(b) averages of entire mineralized band
(i.e., Upper Idaho and Lower Idaho)

Table B (following) gives averages at 0.050 cut off and Table C lists averages of entire mineralized sequences.

Table B, presented below shows the drill intersections using 0.050 as assay cut off. (i.e. no values of 0.050 or below used in averaging.)

TABLE B

HOLE NO.	FROM	TO	WIDTH	Au (oz./ton)
OG 1	12.4	27.0	14.6	0.103
"	31.0	95.0	64.0	0.182
"	99.0	101.5	2.5	0.150
"	113.7	128.2	14.5	0.115
"	133.8	137.2	3.4	0.170
"	141.5	146.0	4.5	0.065
OG 2	11.0	86.8	75.8	0.246
"	90.6	107.7	17.1	0.094
"	113.7	116.0	2.3	0.330
"	133.5	135.4	1.9	0.070
OG 3	69.8	72.0	2.2	0.070
"	112.0	114.3	2.3	0.140
"	181.9	186.2	4.3	0.110
OG 4	67.0	72.5	5.5	0.065
"	262.4	265.0	2.6	0.055
OG 5	187.2	192.8	5.6	0.060
OG 6	87.0	89.0	2.0	0.055
"	93.1	107.1	14.0	0.208
"	113.5	126.4	12.9	0.081
"	129.9	135.9	6.0	0.054
"	140.6	143.1	2.5	0.160
"	159.5	164.4	4.9	0.060
"	177.2	178.8	1.6	0.100
"	194.8	214.9	20.1	0.074
"	248.3	253.4	5.1	0.060
"	258.2	260.1	1.9	0.055
"				

TABLE B (con't)

HOLE NO.	FROM	TO	WIDTH	Au (oz./ton)
OG 7	abandoned			
OG 8	177.3	189.0	11.3	0.198
"	194.2	206.1	11.9	0.215
"	213.5	215.5	2.0	0.120
"	236.6	241.2	4.6	0.081
"	245.9	252.1	6.2	0.111
OG 9	18.1	27.5	9.4	0.127
"	35.4	40.9	5.5	0.070
"	49.1	53.0	3.9	0.070
"	61.6	64.7	3.1	0.150
"	81.0	87.8	6.8	0.131
"	92.0	97.0	5.0	0.098
"	109.5	111.8	2.3	0.380
"	122.1	147.1	25.0	0.173
OG 10	86.0	166.0	80.0	0.164
"	191.7	213.4	21.7	0.147
OG 11	139.4	140.4	1.0	0.110
"	177.8	185.3	7.5	0.123
"	206.0	210.9	4.9	0.140
"	225.9	241.8	15.9	0.195
OG 12	67.8	68.6	0.8	0.100
"	161.8	205.9	44.1	0.075
"	213.2	215.9	2.7	0.150
"	228.8	230.4	1.6	0.180
"	241.0	249.7	8.7	0.181
OG 13	85.0	86.2	1.2	0.150
"	169.3	180.6	11.3	0.255
"	185.4	196.7	11.3	0.062
"	202.1	213.3	11.2	0.071
"	228.6	232.6	4.0	0.118
"	281.3	301.6	20.3	0.148
"	306.6	309.3	2.7	0.330
"	318.8	327.4	8.6	0.220

TABLE B (con't)

HOLE NO.	FROM	TO	WIDTH	Au (oz./ton)
OG 14	176.0	180.2	4.2	0.055
"	202.3	220.1	17.8	0.152
"	225.1	230.7	5.6	0.060
"	234.0	241.7	7.7	0.104
"	246.8	259.8	13.0	0.177
"	264.9	272.0	7.1	0.186
"	281.3	285.7	4.4	0.212
"	393.0	414.5	21.5	0.117
"	421.8	435.0	13.2	0.083
OG 15	219.0	242.3	23.3	0.148
"	261.6	269.8	8.2	0.059
"	285.4	291.9	6.5	0.310
"	442.7	443.2	20.5	0.158
"	472.0	485.9	13.9	0.060
"	494.5	498.4	3.9	0.060
"	526.6	531.2	4.6	0.080
OG 16	227.4	266.0	38.6	0.139
"	353.2	366.4	13.2	0.152
OG 17	172.0	175.0	3.0	0.065
"	329.9	360.0	30.1	0.208
"	413.1	418.8	5.7	0.171
"	422.7	427.4	4.7	0.060
"	476.9	483.3	6.4	0.070
OG 18	334.3	375.0	40.7	0.147
"	426.5	451.6	25.1	0.188
"	464.5	469.6	5.1	0.080
"	504.1	508.8	4.7	0.130
OG 19	303.4	316.6	13.2	0.136
"	343.4	359.4	16.0	0.139
"	363.7	374.3	10.6	0.094
"	391.5	405.8	14.3	0.077
"	510.8	522.7	11.9	0.070
"	572.5	578.7	6.2	0.070
"	596.5	603.4	6.9	0.065
"	623.0	630.2	7.2	0.125
"	641.4	651.9	10.5	0.063
"	662.2	690.2	28.0	0.102
"	838.9	852.8	13.9	0.090

TABLE C

INTERSECTIONS OVER ENTIRE ZONES

HOLE NO.	FROM	TO	WIDTH	AV/AU
OG 1	12.4	151.0	138.6	0.120
OG 2	11.0	135.4	124.4	0.172
OG 3	106.5	114.3	7.8	0.051
"	181.9	188.6	6.7	0.083
OG 4	67.0	77.2	10.5	0.051
OG 5	187.2	194.3	7.1	0.047
OG 6	87.0	214.9	127.9	0.067
"	87.0	260.1	173.1	0.057
"	214.9	260.1	45.2	0.030
OG 7	abandoned			
OG 8	173.0	252.1	79.1	0.088
OG 9	Entire hole		151.2	0.070
OG 10	86.0	213.4	127.4	0.129
OG 11	177.8	241.8	64.0	0.082
OG 12	161.8	249.7	87.9	0.071
OG 13	169.3	232.6	63.3	0.085
"	281.3	327.4	46.1	0.128
OG 14	176.0	285.7	109.7	0.086
"	393.0	435.0	42.0	0.087
OG 15	219.0	291.9	72.9	0.089
"	422.7	503.5	80.8	0.065
OG 16	~~204.2~~ 221.4 one	266.0	33.6	0.138
"	353.2	366.4	13.2	0.152
OG 17	312.2	360.0	47.8	0.143
"	413.1	427.4	14.3	0.091
OG 18	315.8	375.0	59.2	0.113
"	423.3	476.4	53.1	0.106
"	504.1	508.8	4.7	0.130
OG 19	303.4	421.0	117.6	0.076
"	510.8	534.1	23.3	0.057
"	572.5	651.9	79.4	0.048
"	623.0	690.2	67.2	0.072
"	825.0	856.0	31.0	0.058

D-3 GRADE AND TONNAGE ESTIMATES

Grade and tonnage estimates have been made at periodic intervals during the course of the program, and the most recent includes all data to and including drill hole OG 19.

Calculations were conducted with a Wang 600 programmable calculator with programs prepared by D. R. Cochrane and W. Watkins (B.Sc. P.Eng.). Diamond drill intersections were length weighted and averaged and a "revised" whole zone average table similar to table "C" was utilized in calculation. The areas of each drill hole cross section were calculated (Heron's Formula) and an average "area weighted" grade determined for each section. The tonnage and overall grade was calculated by the standard cross section method using the frustrum-of-a-pyramid formula. The specific gravity factor used was 11.5 cubic feet per ton and the grade of the block is assumed to be the average of the two sections weighted by their areas.

Diamond drill hole data was assumed to extend 50 feet in any "open" direction and several projections were necessary due to inclined and off section holes.

Table D follows.

TABLE D

CRADE AND TONNAGE ESTIMATES

A. Upper Idaho Zone

BLOCK	FROM	TO	A_1	G_1	A_2	G_2	H	TONNAGE	GRADE
1	0	DH 3&4	0	0	4367	0.100	50	6330	0.100
2	DH 3&4	DH 1&2	4367	0.100	14690	0.146	44	34520	0.135
3	Esd 1&2	Esd 9	1435	0.172	13308	0.072	44	24380	0.081
4	Wsd 1&2	Wsd 6&10	13255	0.144	18250	0.091	55	75020	0.113
5	Wsd 6&10	Wsd 11&12	18250	0.091	7400	0.079	60	64820	0.087
6	13/12/9	8/11	24054	0.074	13274	0.085	66	105590	0.077
7	8/11	14/15/16/17	13274	0.085	22356	0.102	44	67410	0.095
8	14/15/16/17	19	22356	0.102	11760	0.075	90	131330	0.092
9	19	50'N	11760	0.075	11760	0.075	50	51130	0.075
TOTALS								560,530	0.092

B. Lower Idaho Zone

BLOCK	FROM	TO	A_1	G_1	A_2	G_2	H	TONNAGE	GRADE
1	0	13	0	0	4610	0.128	50	6681	0.128
2	13	11	4610	0.128	0	0	66	8819	0.128
3	11	14/15/16/17	0	0	16687	0.088	44	21281	0.088
4	14/15/16/17	19	16687	0.088	6720	0.072	90	88686	0.083
5	19	50'N	6720	0.072	6720	0.072	50	29217	0.072
TOTALS								154,686	0.086

Notes: A = area
G = average grade of area
H = perpendicular distance between cross sections
Frustrum-of-a-pyramid formula used with SG factor = 11.5 cu'/ton

D-4 DISCUSSION

The southern portion of the Upper ore zone has been trenched on surface (Idaho Trench) and the sample collected for mill testing was selected from a benched section of this trench. This southern section is then easily amenable to open pitting.

The Lower zone has as yet not been uncovered on surface and may in fact merge with the Upper zone in the Idaho Trench area. Both zones are open to the west but are believed to be terminated by the serpentine band and Hozameen Fault which lie several hundred feet west of the drilling area. It was along this latter structure that the "high grade" gold was mined in extensive underground workings of the Aurum Mine. Both zones are also open to the north and east and there is no geological evidence at this point to indicate abrupt termination of these zones. The two mineralized zones have variable attitudes but in general trend down to the north and west at shallow to moderate angles. Since surface elevations increase to the north, an increasing amount of "dead" rock has been encountered in drilling prior to intersecting the Upper zone. There is, however, fair evidence to suggest an additional zone lies above the Upper Idaho Zone in the north area since a chip sample across an outcrop located 300 feet north of hole OG 19 ran 0.43 oz. Au/ton across approximately 15 feet. In addition, drill holes 18 and 19 both intersected anomalous gold values (0.30 and .025 respectively) some 140 feet "above" the Upper zone. The possibility of an additional zone above the Upper

Idaho would, of course, alter the economic picture considerably since it may be possible to proceed for some distance to the north with an open pit.

The key geologic features of the Idaho appear to be:

(a) an alternating sequence of rocks of differing competency (argillites-greywackes).

(b) the presence of the north trending Hozameen Fault and a splay (?) fault which is represented in the north drill sections by the chlorite biotite Schist above the Upper zone.

(c) the replacement nature of the mineralization.

It is the author's opinion that considerable potential exists as far as additional reserves are concerned, and that drilling should continue in order to block out additional tonnage. Early drill holes should be deepened in order to intersect the lower zone. Surface exploration should also continue to the north and should include a thorough examination of the former producing Pipestem deposit situated over one mile north of the Idaho. Constant recalculation of grade and tonnage should be included as drilling continues on the Idaho, and this should be conducted with sufficient flexability to allow for the possibility of additional increases in the price of gold.

Respectfully submitted,

D. R. Cochrane, P. Eng.
June 21, 1974
Delta

PROFESSIONAL PROVINCE OF D. R. COCHRANE BRITISH COLUMBIA ENGINEER

APPENDIX I

Certificate:

I, Donald Robert Cochrane, of the Municipality of Delta, British Columbia, do hereby certify that:

1. I am a consulting geological engineer with an office at 4882 Delta Street, Delta, B. C.

2. I am a graduate of the University of Toronto (1962) with a degree in Applied Geology (B.A.Sc.) and a graduate of Queen's University (1964) with a degree in Economic Geology (M.Sc. Eng.)

3. I have practiced my profession continuously since graduation and while being employed by such companies as Noranda Exploration Co. Ltd., Quebec Cartier Mines, and Meridian Exploration Syndicate. During the last four years I have consulted on an independent basis.

4. I have no interest, either direct or indirect in the properties or securities of Carolin Mines Limited, nor do I expect to acquire any such interest.

5. That I have personally examined the Idaho, Aurum and Pipestem properties, and inspected claim posts and claim lines which appear to be staked in accordance with the regulations set out in the Minerals Act, Province of British Columbia. I have personally supervised the exploration program during 1973 and 1974.

6. This report, in its entirety may be used by Carolin Mines Ltd. in any official or unofficial communications they may have, but excerpts from this report to be used for any communications whatsoever must be reviewed by the author.

7. I am a member in good standing of the Association of Professional Engineers of the Province of British Columbia, and also a member of the A.P.E. in the provinces of Ontario, Saskatchewan and the Yukon Territories.

June 21, 1974
Delta, B. C.

(signed) D. R. Cochrane, P. Eng.

PROFESSIONAL
PROVINCE OF
D. R. COCHRANE
BRITISH COLUMBIA
ENGINEER

APPENDIX II

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METALLURGICAL TESTS

ON A SAMPLE OF GOLD ORE

submitted by

CAROLIN MINES LTD. (N. P. L.)

Progress Report No. 1

Project No.: B391

Date: May 28, 1974

BRITTON RESEARCH LIMITED

1612 WEST THIRD AVENUE

VANCOUVER 9, B.C.

CANADA

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President

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Mr O. E. Gillespie, President,
Carolin Mines Ltd. (N.P.L.),
Suite 811, 850 West Hastings Street,
Vancouver 1, B.C.

May 28, 1974

Dear Mr Gillespie,

Re: Metallurgical tests on gold ore

We give below the results of our tests on the sample of arsenical gold ore which we received from Cochrane Consultants on April 22, 1974:

1. Samples received

The following samples of broken ore were received for investigation:

2 bags marked No. 1 - Gross weight 246 pounds
2 bags marked No. 2 - Gross weight 233 pounds
1 bag marked "Extra" - Gross weight 51 pounds

Instructions were received from Mr Don. Cochrane to combine samples No. 1 and No. 2 before assay and test-work. Some of the lumps were badly weathered and were discarded. The remainder was crushed to minus $\frac{1}{4}$" and sampled for assay. The assay rejects were riffled and one-eighth was crushed to minus 10 mesh, mixed and riffled to give samples suitable for assay and metallurgical tests. The assay sample was pulverised until a small proportion remained on a 100-mesh screen; the plus and minus 100 mesh fractions were weighed and were assayed separately in case coarse gold was present in the sample.

2. Assay of head sample

(a) Assays of first sample ("metallics" not assayed separately):

Gold	0.187 oz/ton	Arsenic	(As)	0.74%
Silver	0.05 oz/ton	Sulphur	(S)	2.73%

(cont.)

Assay of head sample (cont.)

(b) Assay of second head sample ("metallics" assayed separately):

Plus 100 mesh ("metallics") (1.14 grams): Au 6.24 oz/ton
Minus 100 mesh (311.5 grams): Au 0.208 oz/ton
Overall assay (calculated): Au 0.230 oz/ton

Proportion of coarse gold (plus 100 mesh) = 10% of total gold

(c) Additional assays: Cu 0.012%; Pb 0.006%; Zn 0.006%; Fe 6.18%.

3. Specific gravity of ore

2.78, equivalent to 11.5 cubic feet per short ton.

4. Grindability of ore

For grinding from 100% minus 10 mesh to flotation feed size (test 391-1), i.e. 92% minus 200 mesh:

Work index (Bond): 13 kwhr per short ton.

5. Flotation test

A 4000-gram sample of minus 10 mesh ore was ground in a Denver laboratory ball mill for 90 minutes at 65% solids. This was followed by flotation of a bulk gold-arsenic-sulphur concentrate in two stages. The first (rougher) concentrate was cleaned once; the second (scavenger) concentrate was assayed without cleaning. Test conditions and results are shown in tables 2 and 3; screen analyses of the ball mill feed and flotation feed are given in table 4.

Comments: 93.4% of the gold, 94.5% of the arsenic and 87.2% of the sulphur were recovered in the cleaner concentrate, which assayed 2.72 oz/ton gold, 9.46% As and 32.23% S and weighed 7.39% of the feed. An additional 2.0% of the gold was present in the middling products (cleaner tailing and scavenger concentrate), which in practice would be recirculated and part of their gold content would ultimately be recovered. It is therefore expected that at least 94% of the gold would be recovered if similar ore is treated by flotation in a full-scale mill. The grade of the concentrate could almost certainly be improved, and the weight reduced, by a second cleaning operation, but there would be some additional loss of gold.

(cont.)

6. Cyanidation of flotation concentrate

The high-arsenic concentrate produced in the above test would be difficult to market. The possibility of extracting the gold by cyanidation was therefore investigated.

(a) Direct cyanidation of concentrate

A 100-gram sample of the cleaner concentrate was cyanided for 48 hours at 20% solids with sodium cyanide equivalent to 16 pounds and calcium hydroxide equivalent to 8 pounds per ton of concentrate. The pulp was filtered and the washed residue was retreated under similar conditions for a further 24 hours. 91.8% of the gold was extracted in the first 48 hours and only 0.2% was extracted in the next 24 hours, giving an overall extraction of 92.0%, equivalent to 85.9% of the gold in the original flotation feed.

(b) Cyanidation of reground concentrate

The residue from the second cyanidation stage was reground in a ball mill to 99.6% minus 325 mesh and cyanided for a further 24 hours. An additional 2.1% of the gold was extracted, raising the total extraction to 94.1%, equivalent to 87.9% of the gold in the original flotation feed; the final residue assayed 0.16 ounce of gold per ton.

Cyanide and lime consumptions were as follows:

	Lb/ton of concentrate		Lb/ton of original ore	
Stage	NaCN	$Ca(OH)_2$	NaCN	$Ca(OH)_2$
First cyanidation	7.9	7.8	0.6	0.6
Second cyanidation	1.1	2.1	0.1	0.2
Third cyanidation	14.6	4.0	1.1	0.3

Note: The high cyanide consumption during the cyanidation of the reground concentrate was due to lack of protective alkalinity, caused by the reaction between the finely-divided pyrrhotite and the lime added. The cyanide consumption could probably be reduced considerably by increasing the lime addition.

(cont.)

(c) Cyanidation of roasted concentrate

A 100-gram sample of the concentrate was roasted in order to eliminate arsenic and sulphur and render the gold more amenable to cyanidation The calcine, which weighed 76.2% of the feed to roasting, was cyanided under the same conditions as the unroasted concentrate, except that the regrinding and third cyanidation stages were omitted.

3.5% of the gold was lost in roasting the concentrate. 97.6% of the gold in the calcine was extracted in the first 48 hours and an additional 0.5% was extracted in the next 24 hours, giving an overall extraction of 98.1% of the gold in the calcine or 94.7% of the gold in the concentrate. The residue assayed only 0.067 ounce of gold per ton.

Comparison of these results with those obtained on the unroasted concentrate showed that some of the gold was refractory to cyanidation; this gold was probably intimately associated with arsenopyrite. However, in the case of the present sample, the proportion of refractory gold was too low to make it worth considering the inclusion of a roasting step in the treatment, especially considering the problems associated with disposal of the sulphur and arsenic present in the concentrate.

Cyanide and lime consumptions on the calcine were low (1.6 and 4.7 lb/ton respectively in the first cyanidation period, based on the weight of concentrate used).

7. Cyanidation of ore

Two cyanidation tests were carried out on the ore, after grinding 1000-gram samples for 16 and 24 minutes respectively (for screen analyses see table 4). Cyanidation was carried out for 72 hours at 40% solids; results are summarised in the following table (table 1).

(cont.)

Cyanidation of ore (cont.)

Table 1	Test C1	Test C2
Weight of sample taken - Grams	1000	1000
Grinding time - Minutes	16	24
% solids in grinding	65	65
Fineness of grind - % -200 mesh	86	95
Cyanidation period - Hours	72	72
Initial cyanide strength - % NaCN	0.1	0.1
Cyanide consumed - Lb/ton of ore	1.3	1.4
CaO consumed - Lb/ton of ore	3.9	3.8
Assay of feed - Au - oz/ton (direct assay)	0.23	0.23
Assay of residue - Au - oz/ton	0.023	0.018
Gold extraction - %	90.0	92.2
Gold extraction - oz/ton of ore	0.207	0.212

Comments: A high extraction of gold (90%) was obtained in the first test; finer grinding improved the extraction by 2.2%. At the fineness of grind used in the flotation test, an extraction of about 91% would have been obtained. This compares with an overall extraction of 87.9% by flotation, followed by cyanidation of the concentrate (including regrinding). However, the capital and operating costs of an all-cyanidation plant would be appreciably higher than those of a mill using flotation to pre-concentrate the gold, followed by cyanidation of the concentrate, and these extra costs would almost certainly exceed the value of the additional gold recovered by direct cyanidation; in addition, the effluent disposal problem would be more serious in the case of the all-cyanidation plant.

In view of the refractory nature of some of the gold and the possibility that a higher proportion of this refractory gold may be present in different parts of the orebody, it is recommended that additional tests should be carried out on ore from the various zones before final assessment of the deposit is undertaken. For this purpose, a relatively simple cyanidation test, possibly on assay pulps, could be developed.

(cont.)

Since the present tests were only intended to determine the amenability of the ore to various possible treatment methods, no attempt was made to investigate the optimum conditions, including fineness of grind, flotation reagents, conditioning and frothing periods in flotation, as well as solution strengths, contact periods and thickening and filtering characteristics in cyanidation of the ore or concentrate. It would, of course, be necessary to study these factors more thoroughly before completing a feasibility study or undertaking design of a mill.

8. Summary

The results showed that the sample of ore was amenable to direct cyanidation, giving an extraction of 92.2% after grinding to 95% minus 200 mesh. It would, however, almost certainly be more profitable to pre-concentrate the ore by flotation, followed by cyanidation of the reground concentrate, the flotation tailing being discarded. In this way, an overall recovery of about 88% of the gold can be expected, assuming that the present sample is representative.

Yours very truly,

BRITTON RESEARCH LIMITED

John W. Britton

John W. Britton, P.Eng.

Mr O. E. Gillespie (3)

JWB/t

Test 391-1 conditions

Table 2

	STAGE						
	1	2	3	4	5	6	Total
Reagents (a)							
$CuSO_4 \cdot 5H_2O$	-	0.5	-	-	-	-	0.5
H_2SO_4	-	-	-	1.0	-	-	1.0
Pot. amyl xanthate	-	-	0.2	-	0.05	0.03	0.28
Aerofloat 31	-	-	-	-	0.012	-	0.012
Pine oil	-	-	0.054	-	0.009	0.009	0.072
Pulp volume - Ml (b)	-	9000	9000	9000	9000	2600	-
% solids (initial)	65	35	35	32	32	13	-
Time - Minutes	90	5	10	3	5	5	-
pH	-	7.9	8.0	7.2	7.5	8.3	-
Temperature °C	-	22	23	25	26	22	-

Notes: (a) Lb/ton of original ore; (b) Per 4000 grams of original ore

Stages:
1. Grinding (92% -200 mesh)
2. Conditioning
3. Roughing
4. Conditioning
5. Scavenging
6. Cleaning of rougher concentrate

Table 3

Test 391-1 flotation results

(Grind: 92% minus 200 mesh)

Individual results

#	Product	Weight %	Assays Au oz/ton	As %	S %	Distribution % Au	As	S
1	Cleaner concentrate	7.39	2.72	9.46	32.23	93.4	94.5	87.2
2	" tailing	1.64	0.11			0.8		
3	Scavenger concentrate	1.35	0.19			1.2		
4	" tailing	89.62	0.011			4.6		
5	Head (calculated)	100.00	0.22			100.0	100.0	100.0
5	Head (direct assays)		0.23	0.74	2.73			

Cumulative results

1	Cleaner concentrate	7.39	2.72	9.46	32.23	93.4	94.5	87.2
1 + 2	Rougher "	9.03	2.25			94.2		
1 to 3	Combined concentrates	10.38	1.98			95.4		

Additional assays: Head sample 0.012% Cu, 0.006% Pb, 0.006% Zn, 6.18% Fe, 0.05 oz/ton Ag.

Screen analyses

Table 4

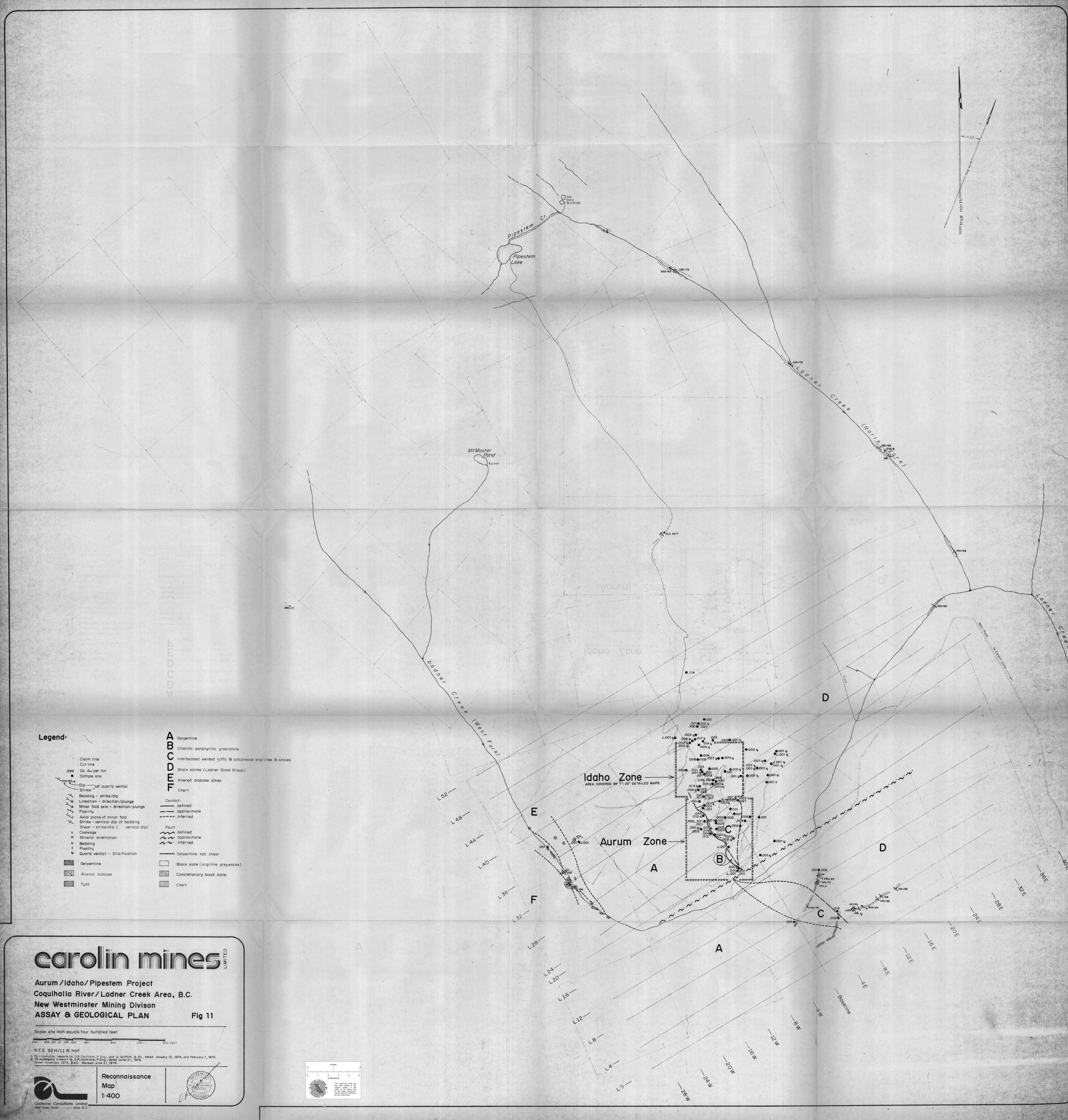
Mesh (Tyler)	Feed to ball mill: % retained Ind.	% retained Cum.	% passing Cum.	Feed to flotation, Test 391-1: % retained Ind.	% retained Cum.	% passing Cum.
10	-	-	100.0	-	-	100.0
14	20.7	20.7	79.3	-	-	100.0
20	15.7	36.4	63.6	-	-	100.0
28	11.1	47.5	52.5	-	-	100.0
35	8.6	56.1	43.9	-	-	100.0
48	7.4	63.5	36.5	-	-	100.0
65	6.0	69.5	30.5	-	-	100.0
100	3.9	73.4	26.6	0.2	0.2	99.8
150	3.9	77.3	22.7	1.3	1.5	98.5
200	3.4	80.7	19.3	6.8	8.3	91.7
+325	4.3	85.0	15.0	19.2	27.5	72.5
-325	15.0	100.0	-	72.5	100.0	-
Total	100.0	-	-	100.0	-	-

Feed to cyanidation (ore)

Mesh (Tyler)	Test 391-C1			Test 391-C2		
35	-	-	100.0	-	-	100.0
48	0.1	0.1	99.9	0.1	0.1	99.9
65	0.1	0.2	99.8	-	0.1	99.9
100	0.6	0.8	99.2	-	0.1	99.9
150	3.5	4.3	95.7	0.8	0.9	99.1
200	9.5	13.8	86.2	4.1	5.0	95.0
+325	20.3	34.1	65.9	16.7	21.7	78.3
-325	65.9	100.0	-	78.3	100.0	-
Total	100.0	-	-	100.0	-	-

Feed to cyanidation (concentrate)

Mesh (Tyler)	Not reground			Reground		
48	-	-	100.0	-	-	100.0
65	0.1	0.1	99.9	-	-	100.0
100	0.1	0.2	99.8	-	-	100.0
150	0.3	0.5	99.5	0.1	0.1	99.9
200	6.0	6.5	93.5	-	0.1	99.9
+325	17.3	23.8	76.2	0.3	0.4	99.6
-325	76.2	100.0	-	99.6	100.0	-
Total	100.0	-	-	100.0	-	-


true north
Old Mine Buildings
Pipestem Cr.
Pipestem Lake
Ladner Creek (North Fork)
McMaster Pond
OLD ADIT
Ladner Creek (West Fork)
Ladner Creek
Idaho Zone
AREA COVERED BY 1" = 20' DETAILED MAPS.
Aurum Zone
A
B
C
D
E
F
Baseline
Legend
Claim line
Cut line
Oz. Au per ton
Sample site
Dip of quartz vein(s)
Strike
Bedding - strike/dip
Lineation - direction/plunge
Minor fold axis - direction/plunge
Fissility
Axial plane of minor fold
Strike - vertical dip of bedding
Shear - strike/dip (vertical dip)
Cleavage
Mineral orientation
Bedding
Fissility
Quartz vein(s) - Silicification
Serpentine
Altered diabase
Tuff
A Serpentine
B Chloritic porphyritic greenstone
C Interbedded welded tuffs & calcareous argillites & shales
D Black slates (Ladner Slate Group)
E Altered diabase dikes
F Chert
Contact
defined
approximate
inferred
Fault
defined
approximate
inferred
Serpentine talc shear
Black slate (argillite greywacke)
Concretionary black slate
Chert
carolin mines LIMITED
Aurum / Idaho / Pipestem Project
Coquihalla River / Ladner Creek Area, B.C.
New Westminster Mining Divison
ASSAY & GEOLOGICAL PLAN
Fig 11
Scale: one inch equals four hundred feet
400 200 100 0 100 200 400 800 1200 1600 FEET
N.T.S. 92 H/11 W. half
1. To accompany reports by D.R. Cochrane, P. Eng., and D. Griffith, B. Sc., dated January 10, 1974, and February 1, 1974.
2. To accompany a report by D.R. Cochrane, P. Eng., dated June 21, 1974.
Drawn November 1973, B.A.C. Revised June 21, 1974.
Reconnaissance
Map
1:400
PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA D. R. COCHRANE
Cochrane Consultants Limited
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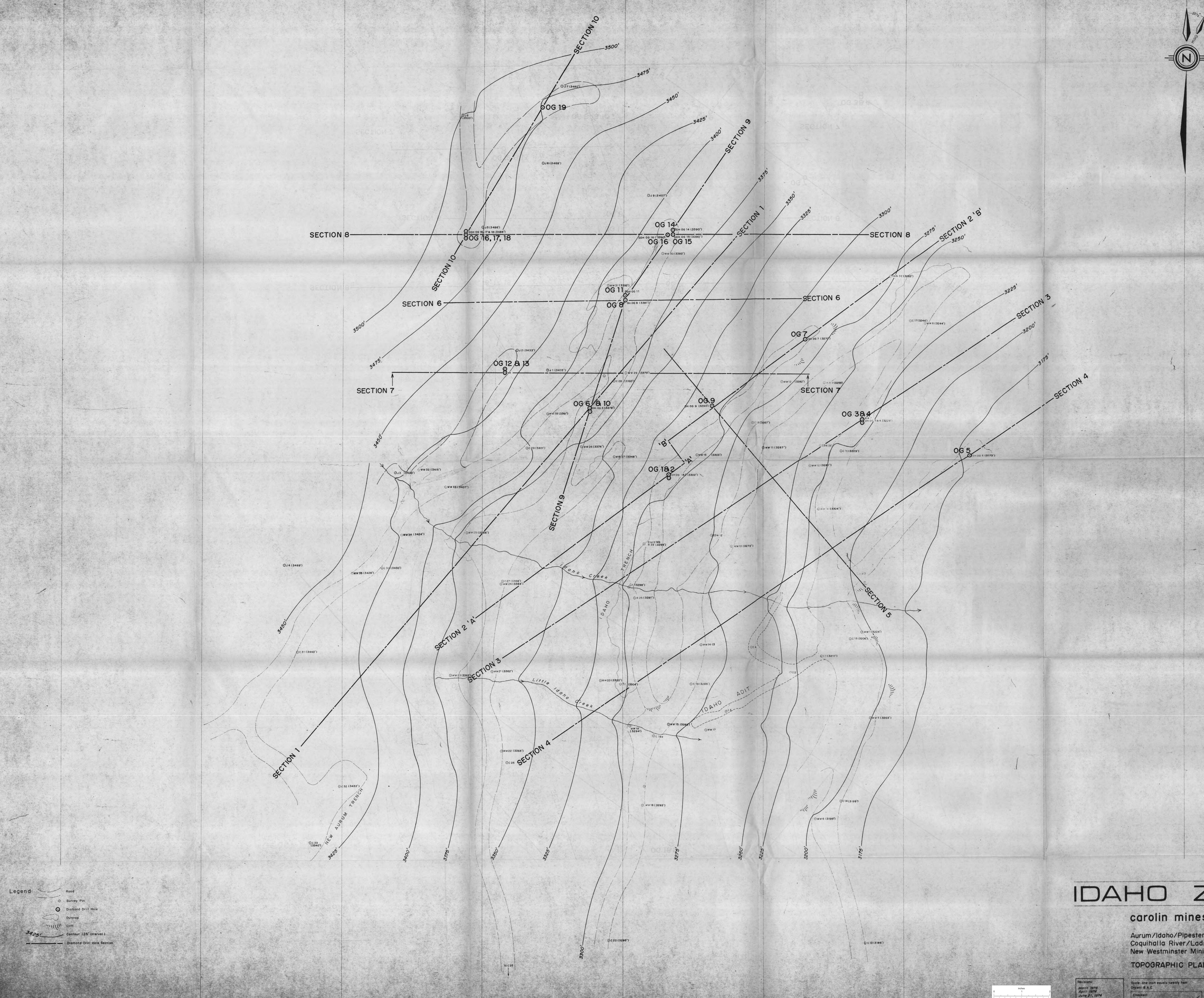


IDAHO ZONE

carolin mines limited

Aurum/Idaho/Pipestem Project
Coquihalla River/Ladner Creek Area
New Westminster Mining Division, B.C.

TOPOGRAPHIC PLAN Fig. 12

Revisions:
March 1974
April 1974
June 21, 1974

Scale: one inch equals twenty feet
Drawn: B.A.C.
Checked:

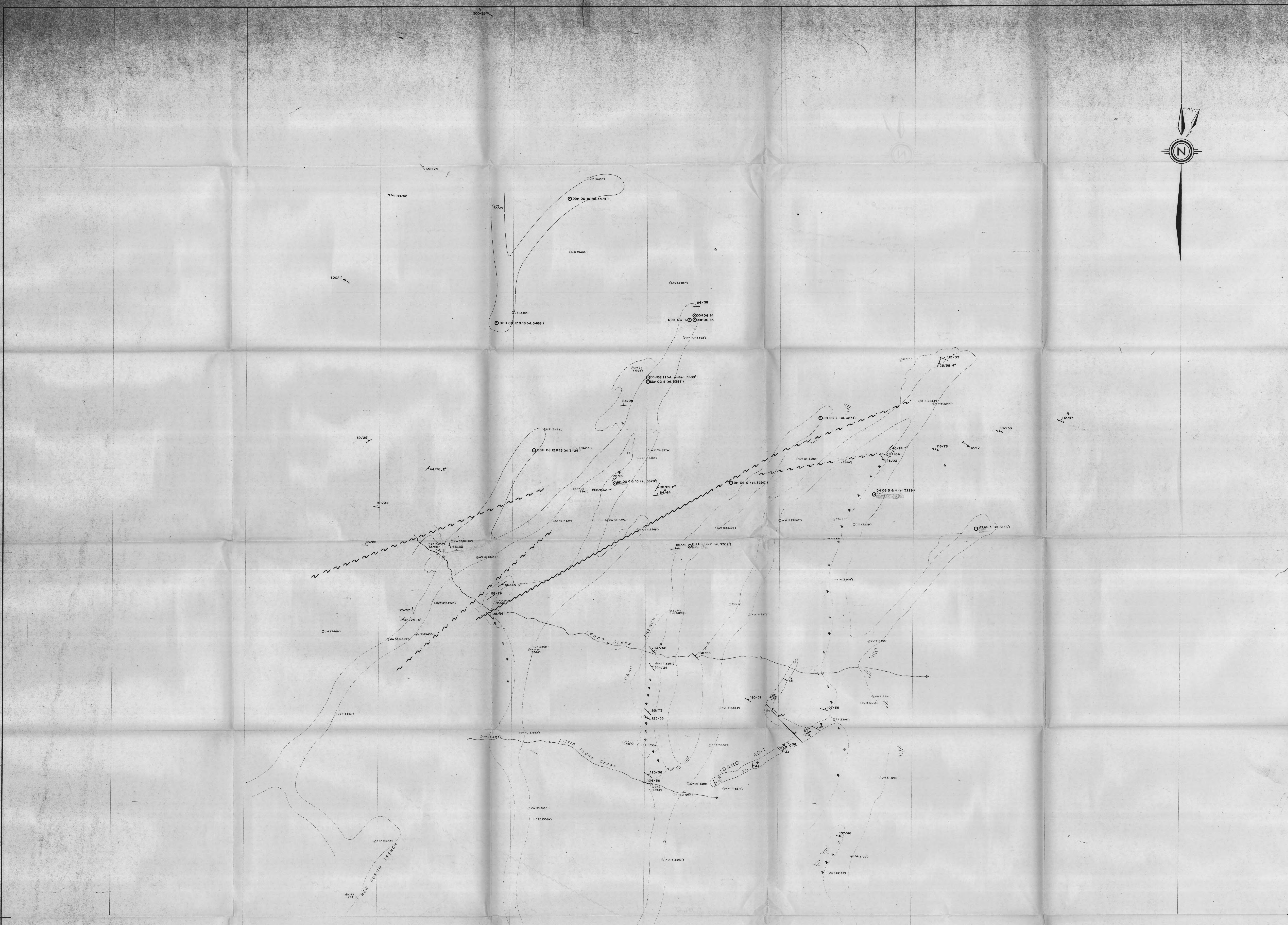
PROFESSIONAL ENGINEER — PROVINCE OF BRITISH COLUMBIA — D. R. COCHRANE

0 10 20 40 60 feet

N.T.S. 92H/11 west half

1. To accompany a report by D.R. COCHRANE, P. ENG., and D. GRIFFITH, B. Sc., dated January 10, 1974.
2. To accompany a report by D.R. COCHRANE, P. ENG., dated June 21, 1974.

Cochrane Consultants Limited
4882 Delta Street —— Delta B.C.



Legend:

- Road
- Survey Pin
- Diamond Drill Hole
- Outcrop
- Cliff
- Bedding - Strike/Dip
- Repeated Fracture - Strike/Dip. Distance noted is approx. period of fracture.
- Fault - Strike/Dip
- Foliation - Strike/Dip
- Shear - Strike/Dip
- Vertical Shear
- c Cleavage - Over bedding measurement indicates paralell cleavage
- q Quartz Vein

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GEOLOGICAL PLAN — Figure 13

Revisions:
March 1974
April 1974
June 21, 1974.

Scale: one inch equals twenty feet
Drawn: B.A.C.
Checked:

PROFESSIONAL ENGINEER — PROVINCE OF BRITISH COLUMBIA — D. R. COCHRANE

0 10 20 40 60 feet

N.T.S. 92H/11 west half

1. To accompany a report by D.R. COCHRANE, P. ENG., and D. GRIFFITH, B. Sc., dated January 10, 1974.
2. To accompany a report by D.R. COCHRANE, P. ENG., dated June 21, 1974.

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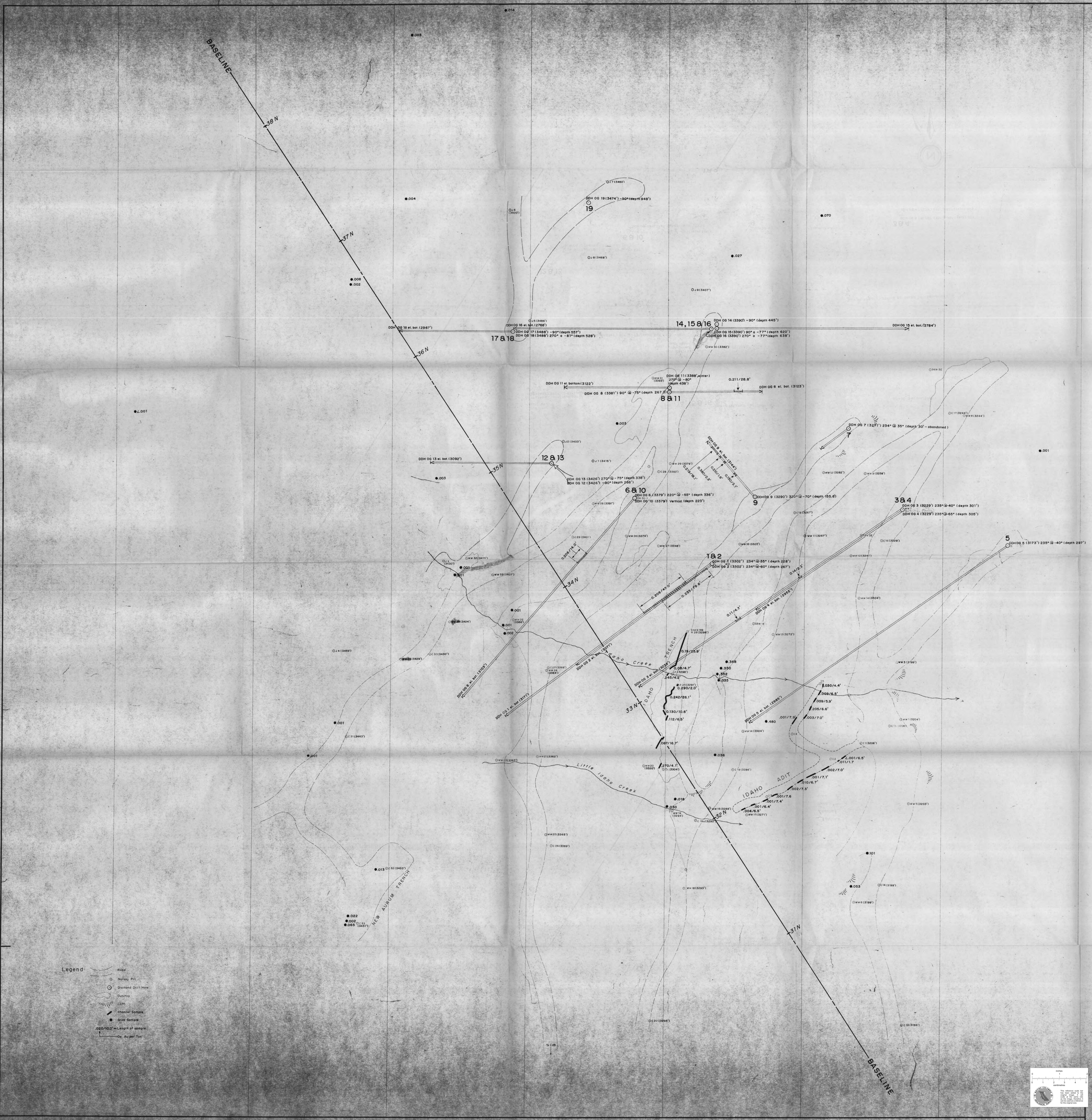

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ASSAY PLAN

Figure 14

Revisions:
March 1974
April 1974
May 1974.
June 1974.
June 21, 1974.

Scale: one inch equals twenty feet
Drawn: B.A.C.

Checked:

PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA D. R. COCHRANE

0 10 20 40 60 feet

N.T.S. 92 H / 11 west half

1. To accompany reports by D.R. COCHRANE, P. ENG., and D. GRIFFITH, B. Sc., dated January 10, 1974, and February 1, 1974
2. To accompany a report by D.R. COCHRANE, P. ENG., dated June 21, 1974.

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