D.D.H. GEOMANAGEMENT LTD.

PROSPECTUS
Feb. 10, 1989.
062304

REPORT

ON THE

ANTONIUK PROPERTY, MOUNT FREEGOLD AREA,

YUKON TERRITORY

FOR

BIG CREEK RESOURCES LTD.
315 - 850 West Hastings Street
Vancouver, B.C.
V6C 1E1

By

D.A. HOWARD, M. Sc., P. Eng.

Property Location
62° 16' N., 137° 05' W.
N.T.S. 115-I-6
Yukon Territory

January, 1988

422 - 470 Granville Street. Vancouver, B.C. Canada V6C 1V5 • Telephone (604) 681-4413

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SUMMARY

The Big Creek Resources Ltd. Antoniuk property is held under option from Discovery Mines Ltd. The property consisting of 14 leased claims and 38 claims held by staking is located on the southern flank of Mount Freegold which is located 45 kilometres northwest of Carmacks, Yukon Territory (62°16'N, 137°05'W).

The property is underlain by coarse grained hornblende syenite which is successively intruded by a Jurassic unfoliated hornblende granodiorite, a porphyritic granite, a Tertiary complex multi-phase feldspar porphyry stock/breccia pipe and a younger Tertiary andesite dyke set.

Soil geochemical and geophysical surveys conducted over the area during 1974 and 1980-81 by Discovery Mines Ltd. and Arctic Red Resources Corp. respectively outlined a large gold-arsenic anomaly in the vicinity of the complex multi-phase feldspar porphyry stock/breccia pipe. Diamond and percussion drill programs in 1975, 1981 and 1986 totalling 14,919 feet (4,547.3 metres) has defined a gold bearing zone containing a mineral inventory of approximately three million tonnes grading 1.28 grams gold per metric tonne. The zone from which the above mineral inventory was calculated is open and essentially untested outside the drilled area. Geochemistry and/or assay results from trenching south and east of the measured zone suggest that the present mineralized zone is much larger than presently indicated.

Metallurgical testing (leach tests) by Witteck Development Inc. during 1986 and 1987 of surface trench material and drill core achieved heap leach extraction of the gold that ranged from 38.5 to 95.6 percent within time limits that would allow for leaching in the Yukon.

It was concluded that the Antoniuk property has an excellent potential for developing more tonnage and the fact that it has good metallurgical characteristics for leaching makes the property worthy of further exploration. Since the economics of the property is tonnage sensitive, it was recommended that an exploration program consisting of diamond drilling and trenching be instituted to

expand the tonnage potential. The estimated cost of the proposed program is $300,000.00

INTRODUCTION

The firm of D.D.H. Geomanagement Ltd. was retained by Big Creek Resources Ltd. to evaluate the exploration and heap leach potential of the Antoniuk property located in the Mount Freegold area of the Yukon Territory.

This assignment was accomplished by reviewing all of the progress reports to date that were compiled by Archer, Cathro & Associates (1981) Limited, operators of the property, government reports, other consultants reports and past reports by the writer on the subject property for Permian Resources Ltd. and Nordac Mining Corporation. The property was last examined by the writer on August 10 and 11, 1985.

PROPERTY AND TITLE

Big Creek Resources Ltd. Antoniuk property (Figure 1 & 2) consists of 14 surveyed and leased (21 year lease) claims held under an option agreement dated March 12, 1985 and an amending agreement dated September 10, 1987, respectively, with Discovery Mines Ltd. of Toronto, Ontario and 38 adjoining mineral claims which were acquired by staking during 1985 and 1986. The located claims are included in the above agreements. Claim details are listed below.

Name	Number	Lease Grant Number	Expiry Date
Surveyed and Leased Claims			
Mayflower	1	2751	March 19, 2001
Baker	1	2765	March 19, 2001
Connie	1	2766	March 19, 2001
Jim	1	2768	March 19, 2001
Donalda 1-9	9	2773-81	March 19, 2001
Donalda 13	1	2782	March 19, 2001
	14		
Mineral Claims			
Nat 1-29	29	YA86843-YA86871	June 5, 1988
Nat 30F-33F	4	YA93013-YA93016	August 12, 1988
Peggy 1	1	YA95146	July 14, 1992
Peggy 2F-4F	3	YA95147-YA95149	July 14, 1992
Peggy 5F	1	YA96268	September 19, 1992
	38		

FIGURE 1

BIG CREEK RESOURCES LTD.

ANTONIUK PROJECT

YUKON TERRITORY

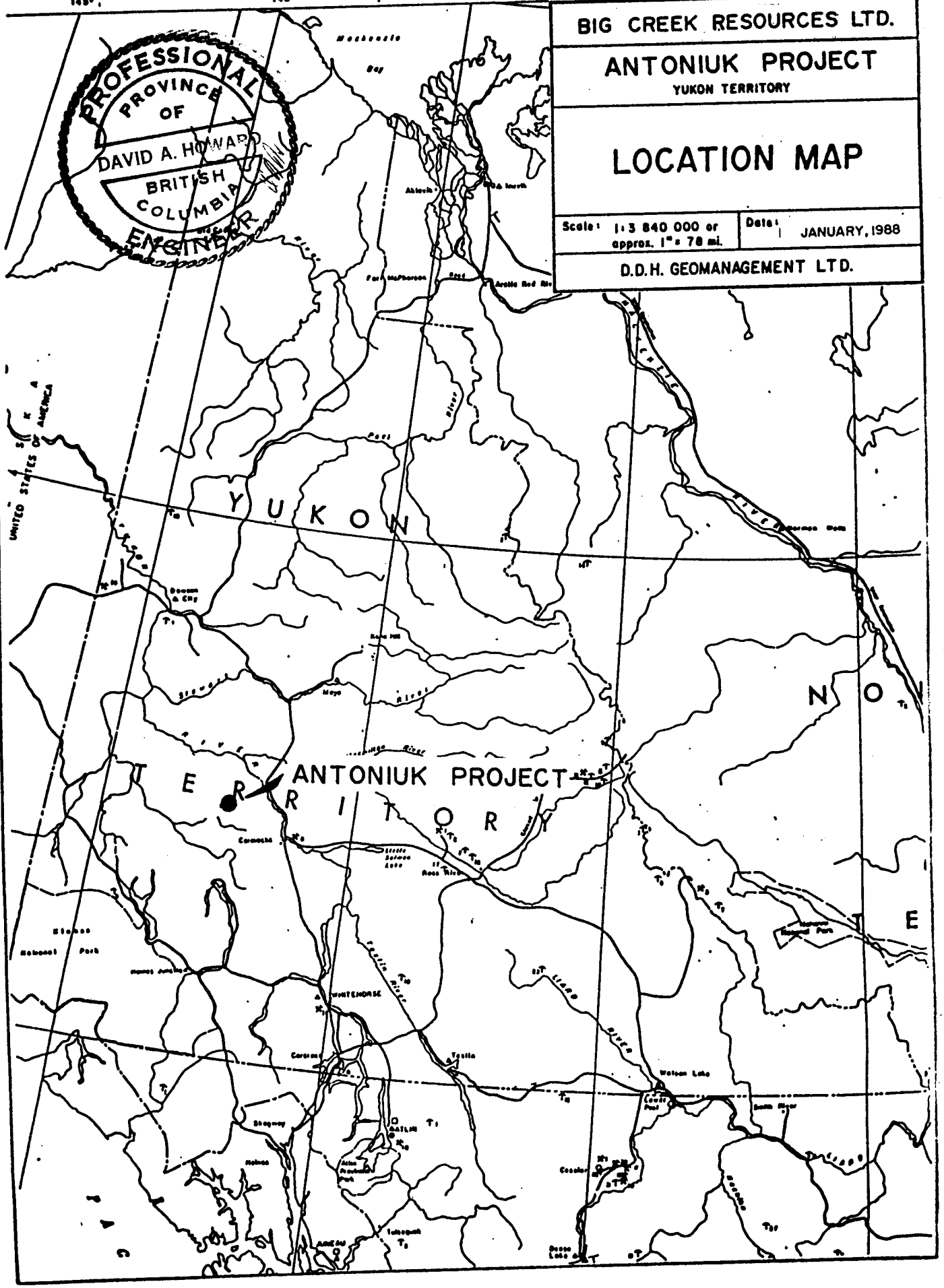
LOCATION MAP

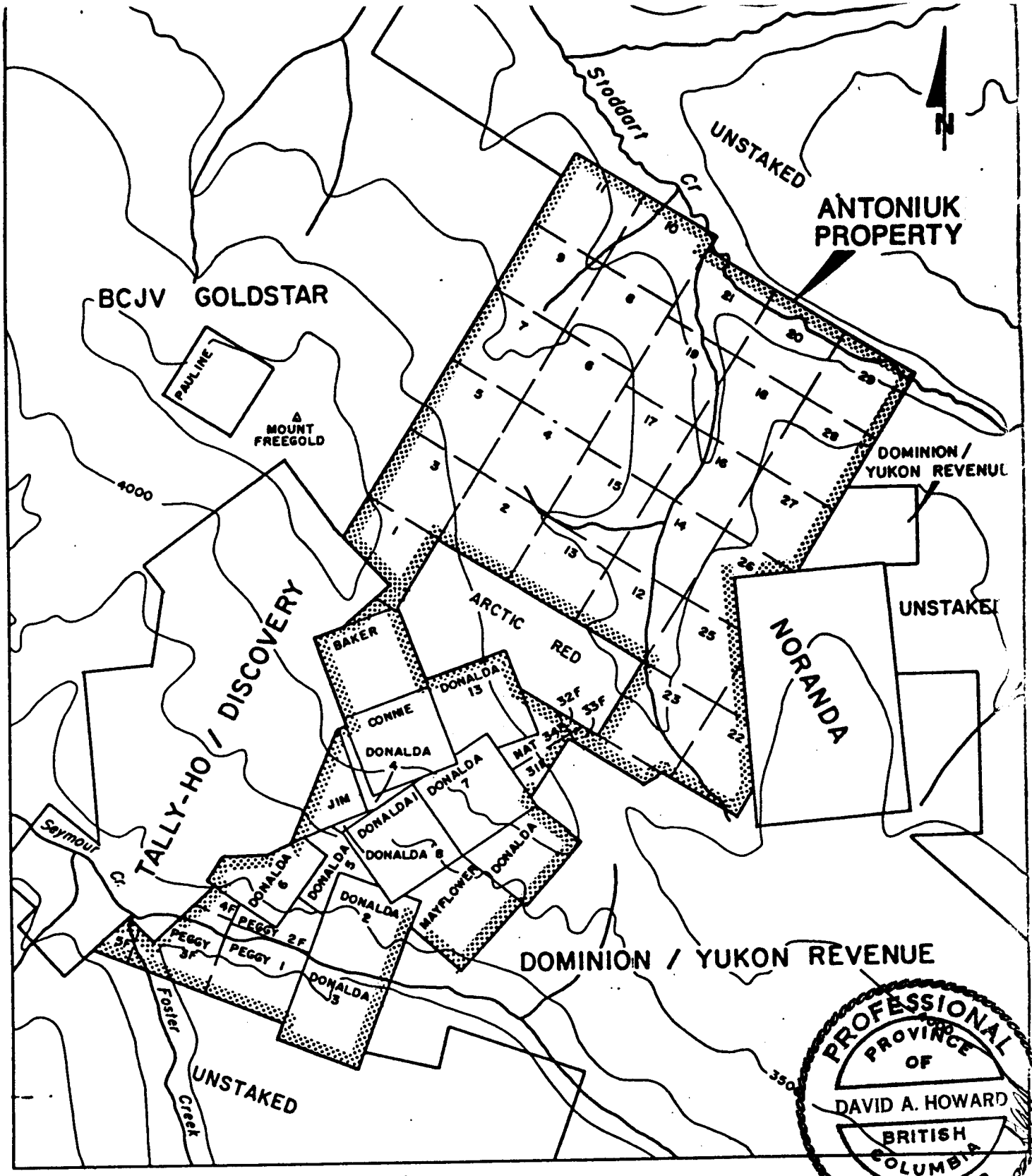
Scale: 1:3 840 000 or approx. 1" = 78 mi.

Date: JANUARY, 1988

D.D.H. GEOMANAGEMENT LTD.

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
DAVID A. HOWARD





Modified from Main (1988)

FIGURE 2

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
DAVID A. HOWARD

CLAIM LOCATION MAP

ANTONIUK PROPERTY

BIG CREEK RESOURCES LTD.

SCALE 1 inch = 1/2 Mile (1:31,680)

0 500 1000 1500 m

0 1500 3000 4500 ft

The Antoniuk property was acquired by Archer, Cathro & Associates (1981) Limited by option and staking. The option was assigned to Big Creek Resources Ltd. (formerly Nordac Mining Corporation). Permian Resources Ltd. and Big Creek Resources Ltd. (Nordac Mining Corporation) formed a joint venture in 1985 to explore the property. Permian presently owns 25 percent of the Discovery option, but is being diluted at a rate of one percent per $7,000.00 of expenditures made by Big Creek Resources and/or its joint venture partners.

In 1987 Big Creek Resources Ltd. formed Big Creek joint venture with Rexford Minerals Ltd. The terms of the joint venture are beyond the scope of this report.

LOCATION, ACCESS, TOPOGRAPHY AND CLIMATE

The Big Creek Resources Ltd. Antoniuk property is located 45 kilometres (28 miles) northwest of Carmacks, Yukon at 62° 16'N, 137° 05'W on the southeast flank of Mount Freegold (claim map sheet 115-I-6).

Access to the property is via a well maintained (summer only) gravel road that starts at Carmacks, Yukon Territory, a distance of 64 kilometres (40 miles).

Topography in the area is subdued with minimum rock exposure, which is typical of the unglaciated Dawson Range. South-facing slopes are normally free of permafrost, at least in areas that are well drained. Elevations on the property range from 944 metres (3,100 feet) to 1,280 metres (4,200 feet).

Temperature and precipitation records are not available for the property, but from records of other locations in the same general area it can be demonstrated that there is at least a 5 month period when the daily mean temperature does not drop below the freezing point.

HISTORY

Mr. P.F. Guder is credited with the discovery of lode gold in the Mount Freegold area in 1930. His discovery caused a stampede during the period 1930-31 when over 100 claims were staked in the area (Bostock, 1936). After the initial flurry, activity waned until the winter of 1934-35 when N.A. Timmins Corporation acquired most of the claims, built a winter road and began development on the Laforma group (portion of adjoining claims to the Big Creek Resources Ltd. property) only to abandon its holdings in the spring of 1935 (Bostock, 1956).

The various properties continued to change hands with minor development and some mining until the main claim group was acquired by Ormsby Mines Ltd. (later reorganized into Discovery Mines Ltd.) (Archer, 1980). Discovery Mines Ltd. developed the G-3 vein, constructed a mill and processed 9,538 tons of "ore" in the period 1963-1966 when the operation was forced to close. No further work was done on the property until 1974 when Discovery Mines Ltd. conducted a soil geochemical survey over the property and discovered the broad gold-arsenic anomaly over the quartz-feldspar porphyry complex which underlies the present Big Creek Resources Ltd. property. In 1975, nine diamond drill holes (4,169 feet) were drilled on the anomaly by Discovery Mines Ltd.

In May 1980, Arctic Red Resources Corp. optioned the entire Discovery Mines Ltd. holdings in the Mount Freegold area. During the period 1980-81, Arctic Red Resources Ltd. conducted a program of soil sampling, geophysical surveying, geologic mapping and diamond drilled nine holes totalling 1,087 (3,568 feet) over the portion of the property now held by Permian Resources Ltd. Arctic Red Resources Corp. dropped its option in 1982 on the block of claims now held by Big Creek Resources Ltd.

"In 1985, the Antoniuk property was optioned by Archer, Cathro & Associates (1981) Limited and assigned to Big Creek Resources Ltd. (formerly Nordac Mining Corporation) and Permian Resources Ltd. Later that year, a program of systematic bulldozer trenching provided exposure for rock sampling and mapping. About 4,950 meters of trench length was excavated within an area approximately 1,300 meters long by 400 meters wide. Some 1,050 channel samples were collected at 4.6 meter intervals and eight rotary percussion holes (607 meters) were drilled within four of the better grade zones outlined by trenching." (Main, 1988)

Preliminary metallurgical testing was undertaken in 1985. Initial column leach tests were performed on surface material and rotary percussion drill cuttings by Coastech Research Inc., North Vancouver, followed by metallurgical and engineering studies later in the year by Witteck Development Inc., Mississauga, Ontario. The 1985 field work was summarized in two reports by R.J. Cathro and J.T. Dennett, dated December 20, 1985 and February, 1986.

In 1986, 22 NQ and 2 HQ holes totalling 2,189 meters (7189) were drilled to test the mineralization previously outlined by bulldozer trenching (see Table 1 for locations).

"In 1987, two bulldozer trenches were cut for geological purposes and two excavator pits were cut to test for supergene surface enrichment. The pits were situated adjacent to vertical 1986 HQ diamond drill holes which had produced the material for metallurgical testing by Witteck. The bulldozer trenches were oriented northeasterly, perpendicular to previous bulldozer trenches and diamond drill sections." (Main, 1988)

REGIONAL GEOLOGY

Geologic mapping (Figure 3) by Bostock (1936) and Tempelman-Kluit (1984) show the Mt. Freegold area to be structurally very complex. The oldest rocks exposed in the area are metasediments and the Jurassic Big Creek Syenite (Tempelman-Kluit, 1984). The coarse grained syenite is gradational with the foliated hornblende granodiorite phase of the Klotassin Batholith which forms the core of the Dawson Range (Tempelman-Kluit, 1974). Locally, the coarse grained syenite appears to cut the foliated hornblende granodiorite (Archer and Carne, 1981).

The foliated hornblende granodiorite is intruded by an unfoliated hornblende granodiorite and other related granitic rocks of Jurassic age. The above intrusive/metamorphic package has been intruded by small dykes and stocks of feldspar porphyry and quartz-feldspar porphyry of Eocene age.

Basalt, andesite and felsite tuffs unconformably overlie the older rocks elsewhere in the district.

TABLE ONE - 1986 DRILL STATISTICS
ANTONIUK DEPOSIT, MT. FREEGOLD, YUKON

DRILL HOLE NUMBER	COORDINATE	SIZE	AZIMUTH (°)	DIP (°)	LENGTH (FEET/M)	OVERBURDEN DEPTH (FEET/M)	1986 DATE STARTED/ COMPLETED
86-1	20N/0E	NQ	Vert.	-90	300/ 91.4	.0/ 0	June 6/ 8
86-2	20N/0E	NQ	330	-50	350/106.7	0.0/ 0	June 6/ 8
86-3	20N/2W	NQ	Vert.	-90	300/ 91.4	6.0/ 1.83	June 8/ 9
86-4	18N/2W	NQ	330	-50	300/ 91.4	6.0/ 1.82	June 9/11
86-5	18N/0E	NQ	330	-50	300/ 91.4	0.0/ 0	June 11/12
86-6	18N/2E	NQ	330	-50	338/103.0	0.0/ 0	June 12/14
86-7	20N/2E	NQ	330	-50	300/ 91.4	6.0/ 1.82	June 14/15
86-8	22N/0E	NQ	330	-50	300/ 91.4	0.0/ 0	June 15/16
86-9	22N/2W	NQ	330	-50	300/ 91.4	6.0/ 1.83	June 17/18
86-10	24N/0E	NQ	330	-50	300/ 91.4	0.0/ 0	June 18/19
86-11	24N/2E	NQ	330	-50	300/ 91.4	0.0/ 0	June 19/21
86-12	24N/5E	NQ	330	-50	348/106.1	35.0/10.50	June 21/23
86-13	26N/2E	NQ	330	-50	300/ 91.4	0.0/ 0	June 23/25
86-14	26N/4E	NQ	330	-50	300/ 91.4	0.0/ 0	June 26/27
86-15	26N/6E	NQ	330	-50	300/ 91.4	0.0/ 0	June 27/28
86-16	16N/4E	NQ	330	-50	300/ 91.4	0.0/ 0	June 29/30
86-17	18N/4E	NQ	330	-50	300/ 91.4	0.0/ 0	June 30/ July 1
86-18	16N/2E	NQ	330	-50	300/ 91.4	0.0/ 0	July 1/ 2
86-19	22N/5E	NQ	330	-50	350/106.7	11.0/ 3.4	July 3/ 4
86-20	22N/2E	NQ	330	-50	300/ 91.4	8.0/ 2.44	July 5/ 6
86-21	18N/1E	HQ	Vert.	-90	200/ 61.0	4.0/ 1.22	July 23/24
86-22	18N/2E	NQ	150	-50	300/ 91.4	0.0/ 0	July 24/25
86-23	24N/4E	NQ	150	-50	296/ 90.2	10.0/ 3.05	July 26/27
86-24	24N/4E	HQ	Vert.	-90	200/ 61.0	11.0/ 3.35	July 27/28
TOTAL					7182/2189.1		

FIGURE 3

BIG CREEK RESOURCES LTD.

ANTONIUK PROJECT

YUKON TERRITORY

REGIONAL GEOLOGY

Modified from Tempelman-Kluit (1984)

Scale:	Date:
1:250,000	JANUARY, 1988

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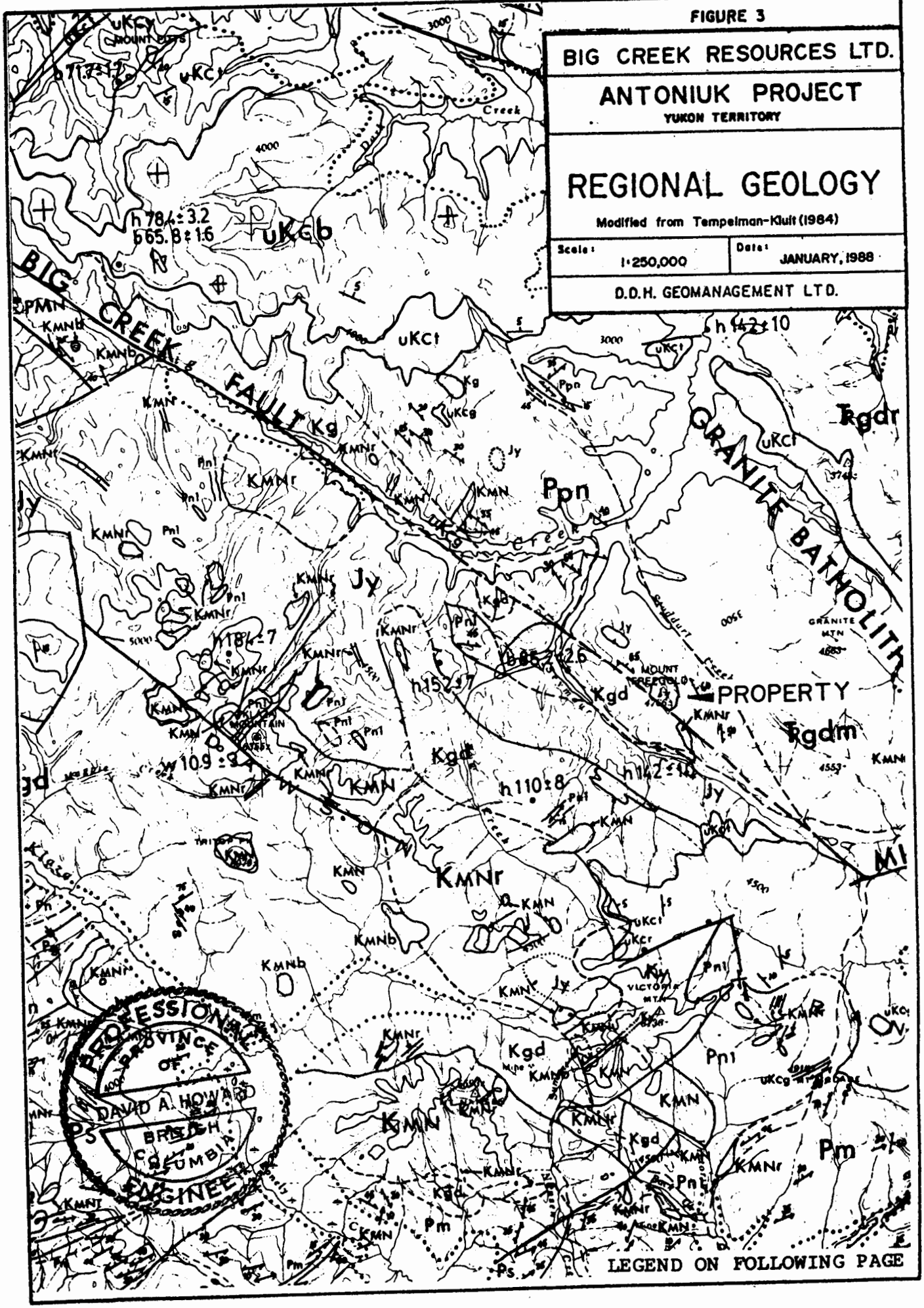


LEGEND ON FOLLOWING PAGE

LEGEND FOR REGIONAL GEOLOGY MAP

(Modified From Tempelman-Kluit, 1984

Paleozoic? - ?Devonian?

?Pelly Gneiss

P_n Moderately resistant, pale buff weathering, medium- to light-grey, muscovite biotite granite to granodiorite gneiss. Carmacks map area.

P_g Moderately resistant, light weathering, foliated biotite leucogranite; gradational to P_n. Carmacks map area.

P_{nl} Recessive weathering, mesocratic, biotite or hornblende granodiorite gneiss: the equivalent of P_n or P_{pn}. Carmacks map area.

P_s Resistant, brownish grey weathering, coarsely schistose quartz mica schist and micaceous quartzite; minor amphibolite. Carmacks map area.

P_c Resistant, white weathering, white sugary marble with a ductile flow fabric. Carmack map area.

P_m Resistant, black weathering amphibolite, amphibolitic gneiss and biotite-amphibole granodiorite gneiss; includes undifferentiated quartz-mica schist; may include minor serpentinite. Carmacks map area.

P_u Dun brown weathering, green to black serpentinite, and serpentinized peridotite and pyroxenite. Carmacks map area.

K_{MNb} Resistant, dark weathering, massive, dark green andesite breccia of pipes and plugs; minor porphyry. Carmacks map area.

K_{MNr} Orange weathering, rhyolite to dacite quartz feldspar porphyry; forms innumerable dykes and small plugs; includes K_{gd}, J_y, P_n K_{MN} undifferentiated. Carmacks map area.

K_{MN1} (Packers Mountain) - Recessive, rusty weathering, aphanitic, flow banded dacite and rhyolite; minor porphyritic dacite. Laberge map area.

K_{MN2} (Teslin Mountain) - Resistant, dark green, massive andesite, greenstone and volcanic breccia. Laberge map area.

K_{gdp} Homogeneous, massive, fine grained, pale mauve porphyritic hornblende biotite granodiorite to syenite; subvolcanic to Mount Nansen Group on Teslin Mountain; gradational to K_{gd}. Laberge map area.

Jurassic

Big Creek Syenite

J_y Resistant, dark weathering, massive, coarse- to very coarse-grained and porphyritic, mesocratic hornblende syenite; locally sheared, commonly fractured and saussuritized; locally has well developed layering of aligned pink K-feldspar tablets; contains screens of undifferentiated gneiss. Carmacks map area.

uK_{Ct} On Prospector Mountain, green, recessive, medium bedded, sandy tuff with interbedded andesitic basalt flows in the top, minor red tuff; north of Big Creek, andesitic basalt flows with minor tuff; includes granite boulder conglomerate west of Minto. Carmacks map area.

uK_{Ctl} Moderately resistant, light weathering, thick bedded, immature volcanic sandstone and conglomerate; minor volcanic flows and dacitic ash flow tuff. Carmacks map area.

uK_{Cy} Resistant, homogeneous, massive, pale mauve, medium grained, equigranular, hornblende syenite to granite commonly with crowded porphyry texture; forms a laccolith on Prospector Mountain and a plug on Mount Pitts. Carmack map area.

uK_{Cr} Pink to white flow banded rhyolite to dacite and felsic breccia, forms small plugs or domes near Braeburn; includes pink welded felsic tuff under uK_{Ca} north of Victoria Mountain. Carmacks map area.

uk_{Cg} Coarsely crystalline gabbro and diorite; forms a small plug in the Carmacks Group east of Victoria Mountain. Carmacks map area.

uk_{Ca} Lower part: thick, green, hornblende feldspar porphyry andesite flows with interbedded greywacke and breccia;
Upper part: brownish purple, thick, vesicular, porphyritic augite andesite and trachyte, minor sandy airfall tuff. Laberge map area.

uK_{Cal} Resistant, dark grey to black weathering, thick feldspar porphyry andesite flows. Laberge map area.

uK_{Cyl} Resistant, dark weathering, massive, homogeneous, coarse grained porphyritic hornblende syenite; a small plug. Laberge map area.

uK_{Cp} Resistant, homogeneous, fine grained, biotite hornblende granodiorite to quartz diorite; small plugs in the Laberge Group east of the Miners Range. Laberge map area.

Mid-Cretaceous

Mount Nansen Group

K_{MN} Resistant, dark weathering, dark green, massive andesitic plagioclase porphyry and andesite breccia: forms plugs, pipes and dykes. Carmacks map area.

YUKON CRYSTALLINE TERRANE

Pleistocene and Recent

Selkirk Volcanics

Q_S Resistant, brown weathering, columnar jointed, vesicular to massive basalt flows; minor pillow basalt; Q_{Sl}- basaltic tuff and breccia at Volcano Mountain and opposite the mouth of Wolverine Creek; Q_{Sa}-oldest to youngest -Q_{Sd} lava flows from Selkirk volcano. Carmacks map area.

Upper Cretaceous

Carmacks Group

uK_{Cb} Brown weathering, resistant, brown basalt flows. Carmacks map area.

WHITEHORSE TROUGH

Upper Jurassic and/or Cretaceous

Tantalus Formation

JK_T Thickbedded, resistant, chert-pebble conglomerate, minor interbedded gritty chert-grain sandstone: JK_{T1}- massive to thickbedded, gritty sandstone with quartz, chert and feldspar grains: JK_{TV}- red weathering, dacite to andesite flows beneath Tantalus strata near Hootalinqua. Laberge map area and Carmacks map area.

Middle Jurassic (Bajocian to Bathonian)

Teslin Crossing Stock and Dykes

Jpp Medium to fine grained, equigranular, leucocratic monzonite, syenite and granite (Teslin Crossing Stock); dykes of dacite to andesite porphyry with euhedral andesine, hornblende and locally quartz in aphanitic greenish, or grey groundmass. Laberge map area.

Lower and Middle Jurassic

Laberge Group

J_L Undifferentiated Laberge and Lewes River Group shale, greywacke and conglomerate betwdeen Open Creek and Teslin River; J_{Lcg}-conglomerate, like that of the Conglomerate Formation, but with some chert clasts; TR_H?- limestone, probably part of the Lewes River Group. Laberge map area.

Upper Triassic

Minto Pluton and Granite Mountain Batholith

TR_{gdm} Massive, medium- to coarse- grained, heterogeneous, equigranular, mesocratic, foliated biotite-hornblende granodiorite; locally strongly foliated (Tr_{gdm1}); locally contains biotite rich screens and gneiss schlieren and screens, probably Ppn; locally porphyritic with pink K-feldspar phenocrysts (TR_{gdm2}). Carmacks map area.

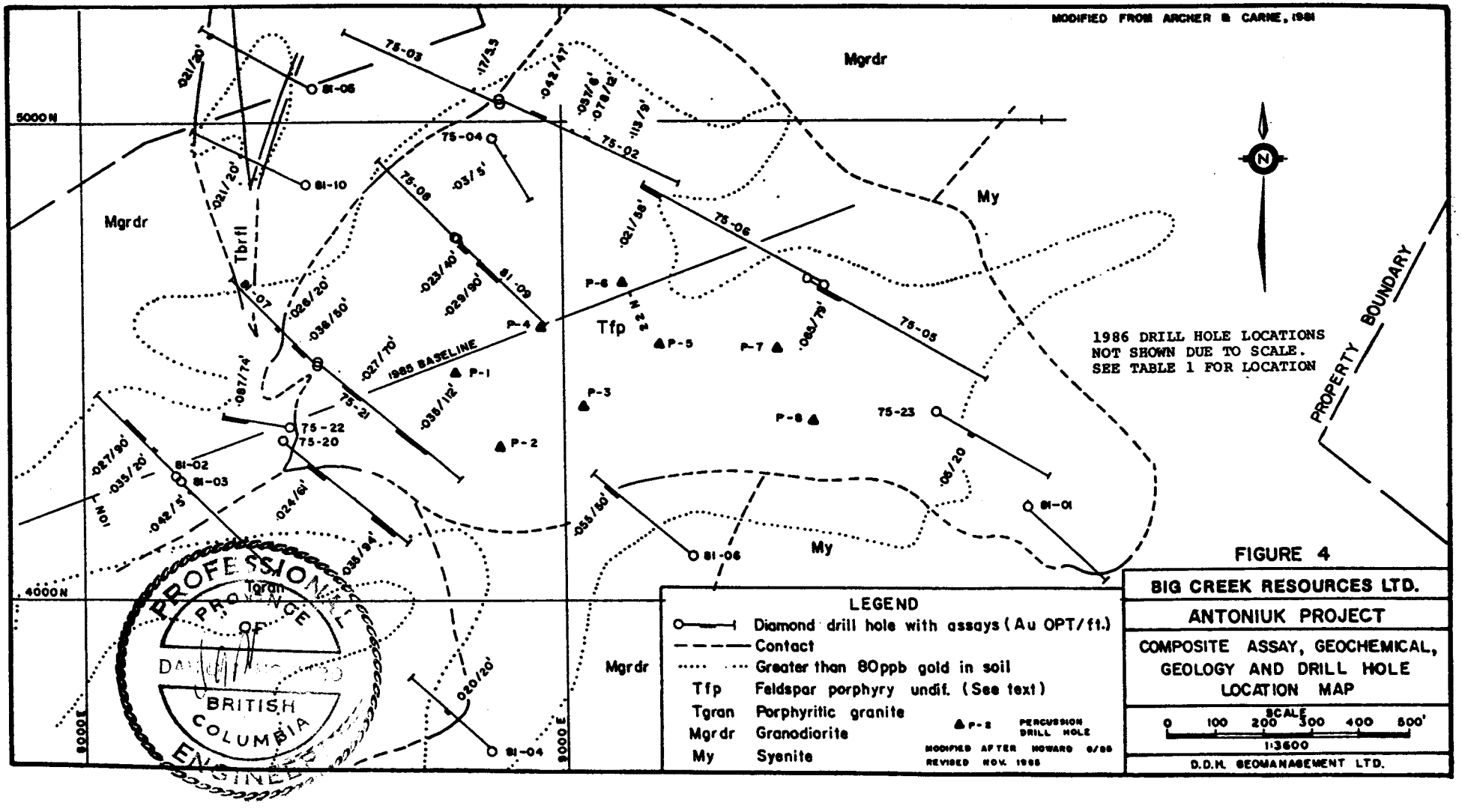
Gold mineralization appears for the most part to be associated with the early Tertiary intrusive event including the gold mineralization on the Permian Resources Ltd. option.

Major northwest trending faults with associated north to northeast trending cross faults are fairly common in the district. The main gold veins in the Freegold district (G-3, Rambler) are associated with the cross structures.

PROPERTY GEOLOGY

The Big Creek Resources Ltd. Antoniuk property is underlain by coarse grained hornblende syenite (My) which is intruded by Jurassic unfoliated hornblende granodiorite (Mgrdr). Both rock types have been intruded by a Tertiary complex multi-phase feldspar porphyry (Tfp) intrusions (both dykes and a stock) of varying compositions (Figure 4). The main Tertiary intrusion forms a triangular shaped stock/breccia with maximum north-south dimension of 396 metres (1,300 feet) and maximum east-west dimension of 488 metres (1,600 feet). Contacts between the porphyry/breccia complex and the enclosing rocks are brecciated as are the contacts between varying phases within the complex.

Detailed geologic mapping by Archer, Cathro (1981) Limited, based on limited outcrops, float and drill hole projections, identified 6 distinct rock types associated with the Tertiary porphyry/breccia complex. The internal rock type details are omitted from the writer's geologic map (Figure 4) because of scale, but are listed as follows because there appear to a positive correlation between rock types and the presence of gold mineralization. The composition of the Tertiary feldspar porphyry stock breccia pipe (Figure 4) is as follows:


MODIFIED FROM ARCHER & CARNE, 1981
5000N
4000N
8000E
9000E
Mgrdr
My
Tfp
Tgran
Tbrfl
1985 BASELINE
1986 DRILL HOLE LOCATIONS NOT SHOWN DUE TO SCALE. SEE TABLE 1 FOR LOCATION
PROPERTY BOUNDARY
LEGEND
Diamond drill hole with assays (Au OPT/ft.)
Contact
Greater than 80ppb gold in soil
Feldspar porphyry undif. (See text)
Porphyritic granite
Granodiorite
Syenite
PERCUSSION DRILL HOLE
MODIFIED AFTER HOWARD 6/86
REVISED NOV. 1986
FIGURE 4
BIG CREEK RESOURCES LTD.
ANTONIUK PROJECT
COMPOSITE ASSAY, GEOCHEMICAL, GEOLOGY AND DRILL HOLE LOCATION MAP
SCALE
0 100 200 300 400 500'
1:3600
D.D.H. GEOMANAGEMENT LTD.
PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA

1) Felsite (TFLST) - Light coloured, very fine grained
2) Brecciated Felsite (TBRFL) - Developed near contacts, contains fragments of granodiorite and syenite
3) Quartz Porphyry (TPPQZ) - 1-5% fine grained quartz phenocrysts
4) Feldspar Porphyry (TPPFL) - fine grained euhedral feldspar phenocrysts
5) Feldspar-Quartz Porphyry (TPPFQ) - dark green colour
6) Quartz-Feldspar-Biotite-Hornblende Porphyry (TQFBX) - quartz feldspar, biotite, and hornblende phenocrysts

"The breccia body is composed of fragments of porphyry, porphyritic rhyodacite, syenite and granodiorite, all of which outcrop around the diatreme. The breccia is interpreted as having an intrusive, subvolcanic origin, and it is roughly coeval with Mid-Cretaceous Mount Nansen Group porphyry dykes and fine-grained intrusive rocks of rhyodacite composition that are common on Mt. Freegold. It is noteworthy that this is the only such breccia body presently identified on Mt. Freegold.

The gold-bearing zones at Antoniuk occur within or adjacent to the diatreme, in altered and brecciated rock that is usually heterolithic but occasionally homoclastic (porphyry or granodiorite). Fracturing is pervasive although no well defined mineralized vein structures have been recognized.

Weathering has removed most traces of sulphide mineralization from surface rocks, leaving only occasional disseminated pyrite and limonitic staining. Mineralogical studies of the oxide material by Witteck have shown that gold occurs as free particles within limonite and, hence, probably had an original affinity with sulphides. Although quartz veins are not abundant, thin limonitic fractures are common and some of these contain quartz or carbonate veinlets.

Within the oxidized hypogene zone, the principal sulphide material is pyrite which occurs both as disseminations and within thin quartz veinlets. Heterolithic breccia is the main host, containing up to 3% pyrite. Small amounts of arsenopyrite and trace amounts of chalcopyrite are also present. Logs of 1975 and 1981 drill holes noted the presence of occasional patches of other sulphides, including stibnite, bornite, galena, sphalerite, and molybdenite. Of these, bornite was the only sulphide observed in 1986. Silver values increased toward the southeast end of the breccia body ("Southeast Area" on Figure 7 on the following page) and may be related to rare occurrences of copper sulphides (chalcocite and bornite) and sulfosalts (tetrahedrite?). Individual drill intersections from this area assay up to 90 g/t silver over 1.5 meters." (Main, 1988)

MINERALIZATION AND MINERAL INVENTORY

Gold mineralization occurs as free gold (at least locally) in association with pyrite, stibnite, jamesonite, arsenopyrite and unidentified sulfosalt minerals, although this combination of sulphides is not always present at every location. Based on the diamond drilling, the high sulphide (2-5 percent) zones which contain gold, are usually associated with quartz vein stockworks in breccia zones near or at contacts between the various rock units. The vein quartz is commonly vuggy and range in width from less than 5 mm to 300 mm (12 inches). Disseminated pyrite and/or arsenopyrite is common in most rock types, but does not always contain gold mineralization. Limonite is commonly developed to below 200 feet (61 metres) throughout the area although sulphides are usually always present.

Hydrothermal alteration varies in intensity and type in the mineralized zones and does not appear to be a good guide to gold content. Table 2 summarizes the various correlations between rock types, alteration, structure and gold content in the 1975 and 1981 diamond drill holes. Based on the 1975 and 1981 diamond drilling, the average weighted gold content of all mineralized intersections is 0.042 ounces per short ton (1.440 gram/mt) with a range of values from 0.005 ounces per short ton (0.171 gram/mt) to 0.77 ounces per short ton (26.400 gram/mt). The mineralized intervals ranged from 5 to 120 feet (1.52 to 36.6 metres) and have an average width of 43 feet (13.1 metres).

During 1985, a total of eight 250 foot (76.2 metre) percussion holes (see Figures 5, 6, 7, 8 & 9) were drilled in the trenched area. The weighted average gold assay grade of all the higher grade mineralized sections (10 feet sample interval - 3 metres) greater than 0.020 ounces per ton is 0.048 ounces per ton. (Based on 76 out of 200 samples.) The weighted average grade of all drill samples is 0.029 ounces gold per ton.

TABLE 2
MINERALIZED INTERVAL SUMMARY

Hole No.	Colar Elev.	Bearing	Dip	From	To	Major Rock Type	Alteration	Structure	Gold oz/t	Silver oz/t	Gold Range From	Gold Range To	Inverval Width	Au x W
R-75-02	4220	115	-45	101	148	TBRFL	mod. koal	slight breccia	0.042	0.125	0.005	0.18	47	1.974
				230	236	TBRFL	minor argillic	breccia	0.057	0.09	0.035	0.08	6	0.342
				279.5	292	TBRFL	minor kaolin	breccia/fault	0.078	0.113	0.03	0.145	12.5	0.975
				398	407	TFLST	mod. argillic	—	0.113	0.10	0.10	0.13	9	1.017
R-75-03	4219	295	-45	93.9	99.4	TPPQZ	intense kaol.	breccia	0.17	0.15	0.02	0.39	5.5	0.935
R-75-04	4211	150	-55	70	75	TBRFL	minor kaol	breccia	0.03	0.10	0.03	0.03	5	0.15
R-75-05	4187	130	-45	40	119	TBRSY/ANDS	intense kaol	breccia	0.065	0.33	0.015	0.26	79	5.135
R-75-06	4185	310	-45	550	608	TBRFL	intense kaol	breccia	0.021	0.13	0.01	0.065	58	1.218
R-75-20	3833	130	-50	116	177	KGRAN	mod. kaol	—	0.024	0.057	0.005	0.16	61	1.464
				383	477	KGRAN	intense kaol	—	0.035	0.09	0.05	0.18	94	3.290
R-75-21	3906	130	-45	95	165	TBRGD	mod. kaol	breccia	0.027	.06	.005	.05	70	1.89
	3906	130	-45	300	412	TBRGD/BREL	mod. prop.	breccia	0.035	.089	0.01	0.155	112	3.920
R-75-22	3833	280	-45	128	202	TGRDR	minor argillic	near contact	0.087	0.116	0.005	0.77	74	6.438
R-75-23	4169	120	-45	13	133	TBRFL	mod. kaol	breccia	0.05	1.39	0.01	0.14	20	3.500
AR81-01	4172	135	-50			TFLST		no values >0.015					-	
AR81-02	3829	315	-50	100	120	TGRDR	minor argillic	—	0.035	0.03	0.005	0.091	20	0.700
				155	245	TGRDR	mod. inten. kaol	—	0.027	0.038	0.007	0.046	90	2.43
AR81-03	3829	135	-50	40	45	TGRDR	minor kaol		0.042	0.007	0.42	0.042	5	0.201
AR81-04	4202	315	-50	180	200	TGRAN	mod. kaol		0.020	0.016	0.013	0.027	20	0.400
AR81-05	4020	295	-50	360	380	TBRFL	mod. kaol	breccia/contact	0.021	0.084	0.014	0.028	20	0.420
AR81-06	3987	315	-50	310	360	TBRFL	prop/argillic	breccia/fault	0.055	0.079	0.016	0.118	50	2.750
AR81-07	3907	315	-50	50	100	TBRFL/FLST	mod. kaol	breccia/contact	0.036	0.032	0.012	0.100	50	1.800
				160	180	TBRGD	minor kaol	breccia/contact	0.026	0.033	0.023	0.029	20	0.520
AR81-08	4127	315	-50					no values >0.015					-	
AR81-09	4127	135	-50	20	60	TBRFL	mod. kaol	breccia	0.023	0.042	0.01	0.041	40	0.920
				100	190	TBRFL	minor kaol	breccia	0.029	0.075	0.005	0.074	90	2.610
AR81-10	4125	295	-50	230	250	TBRFL	intense argillic	breccia	0.021	0.004	0.01	0.033	20	0.420
													1078	45.428

Weighted Average Gold Content - 0.042 OPT

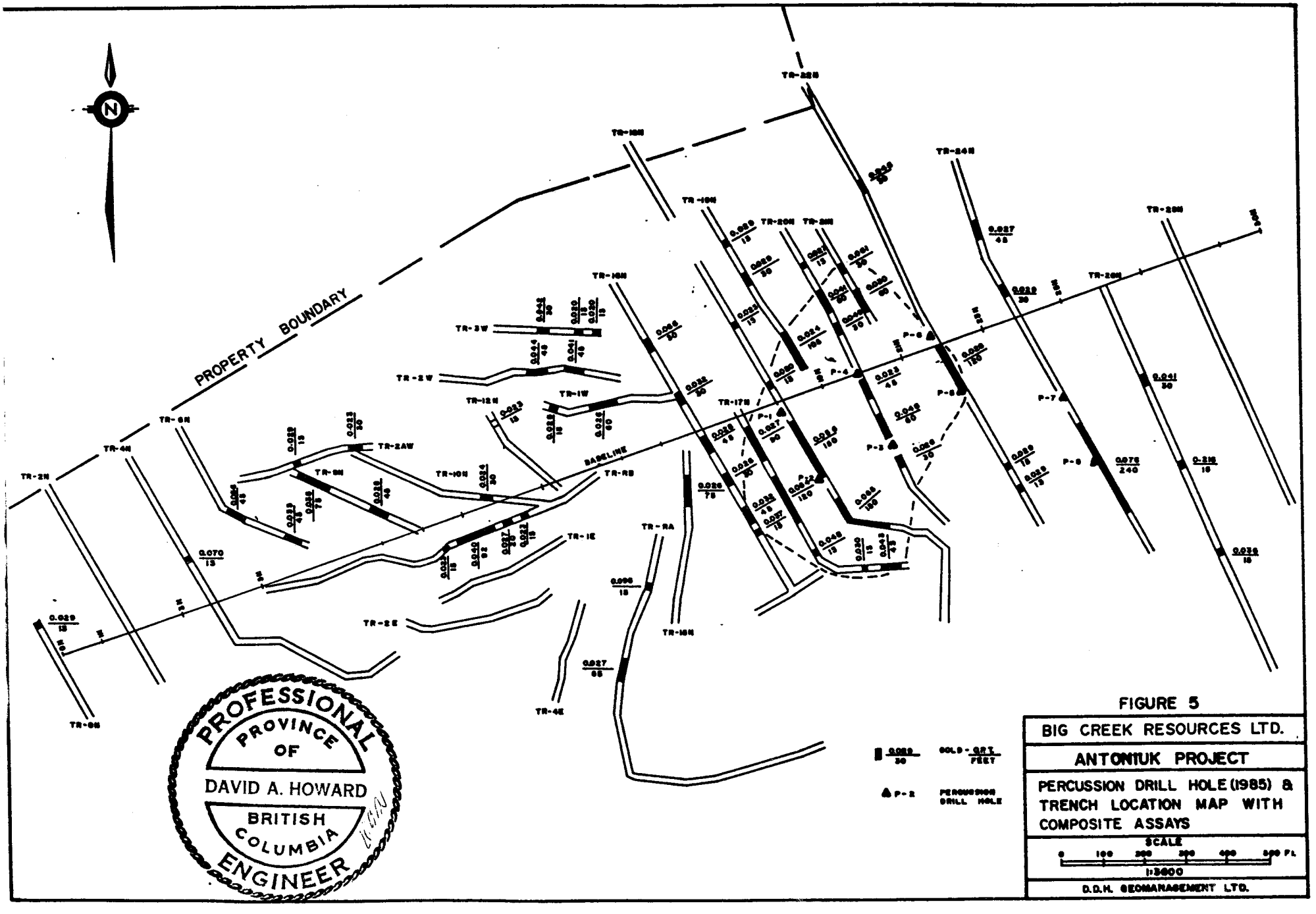


FIGURE 5

BIG CREEK RESOURCES LTD.

ANTONIUK PROJECT

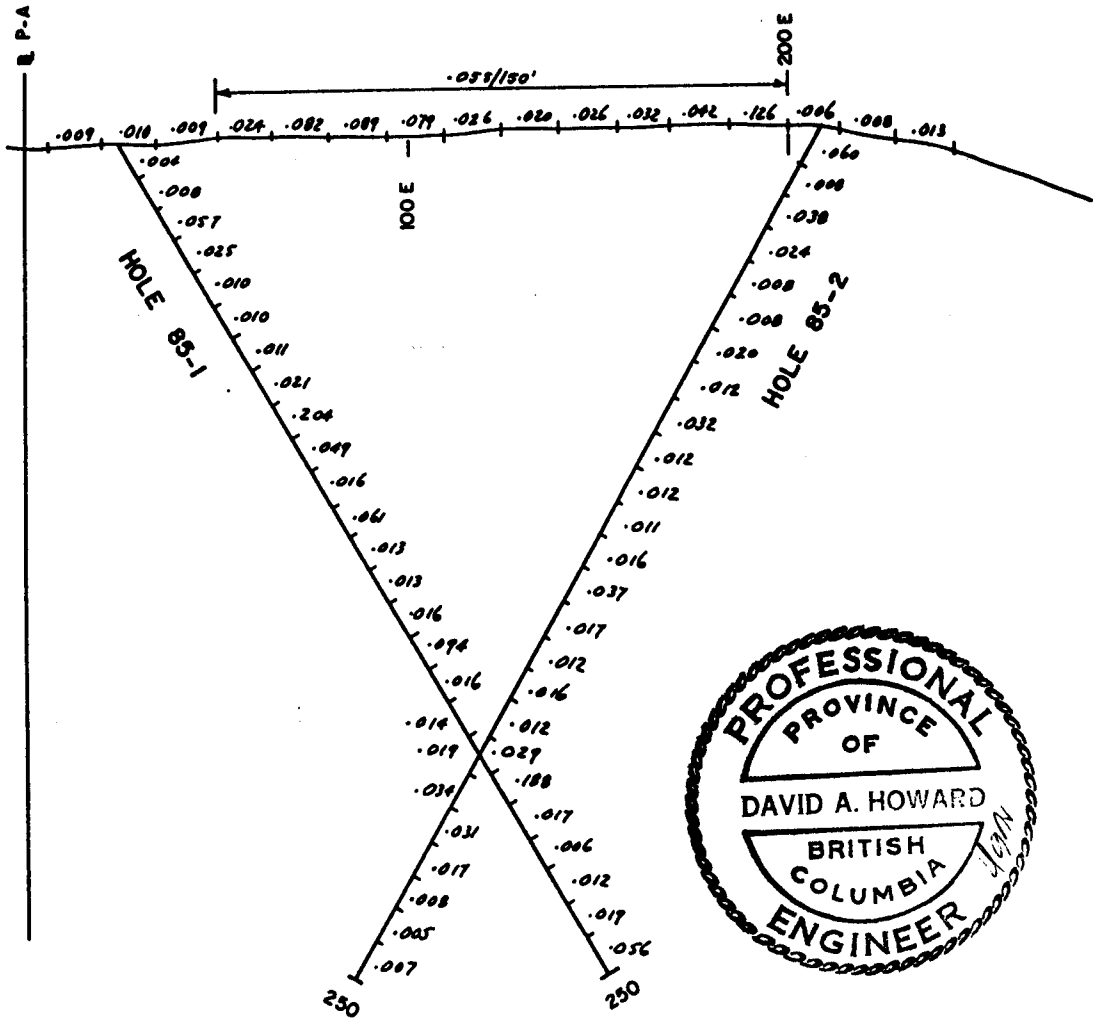
PERCUSSION DRILL HOLE (1985) & TRENCH LOCATION MAP WITH COMPOSITE ASSAYS

SCALE

0 100 200 300 400 500 Ft.

1:3600

D.D.H. GEOMANAGEMENT LTD.



PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
DAVID A. HOWARD
N/A

0 50 100 Feet

SCALE 1:600
MODIFIED FROM ORIGINIAL

Assays in oz/ton Au by fire assay
at Chemex Labs, Vancouver

INCLUDED BY PERMISSION

FIGURE 6

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

VERTICAL CROSS-SECTION

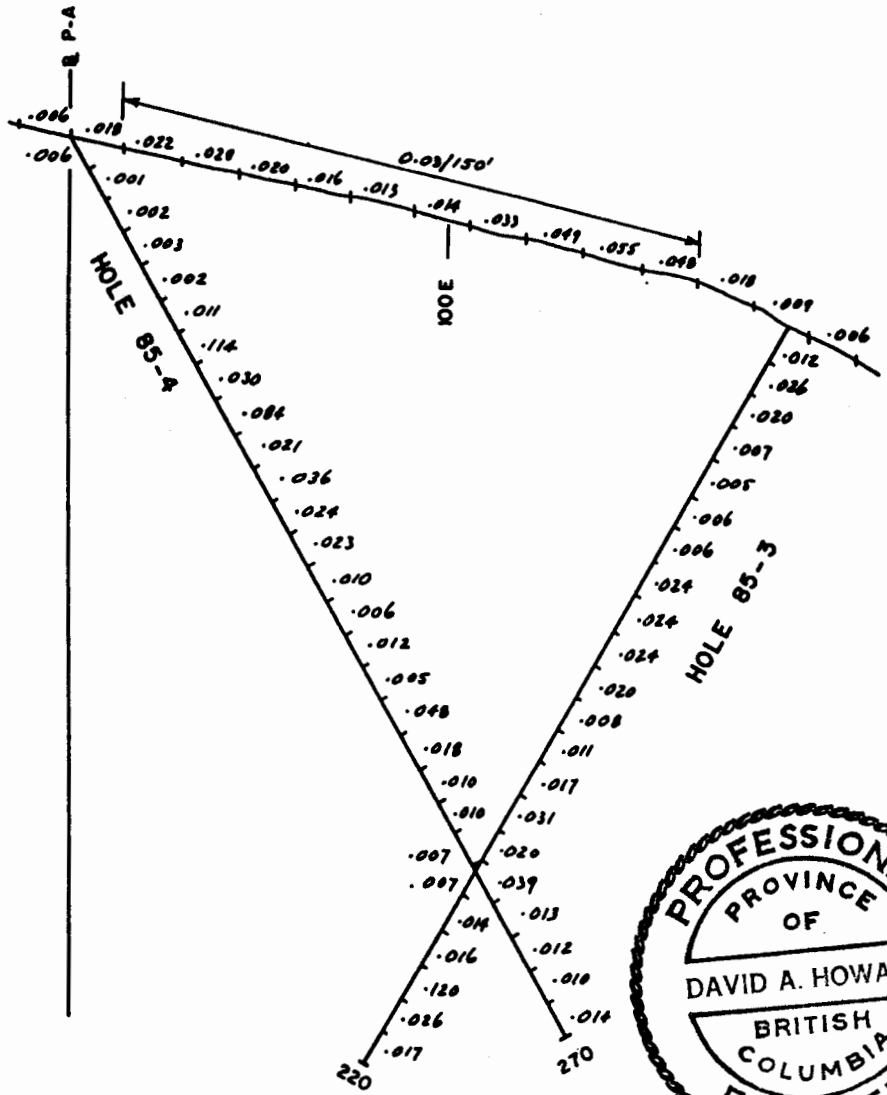
HOLES 85-1 & 85-2

TRENCH 18N

ANTONIUK DEPOSIT- MT. FREEGOLD, Y.T.

BIG CREEK RESOURCES LTD.

August, 1985



0 50 100 Feet

SCALE 1:600
MODIFIED FROM ORIGINIAL

Assays in oz/ton Au by fire assay
at Chemex Labs, Vancouver

INCLUDED BY PERMISSION

FIGURE 7

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

VERTICAL CROSS-SECTION

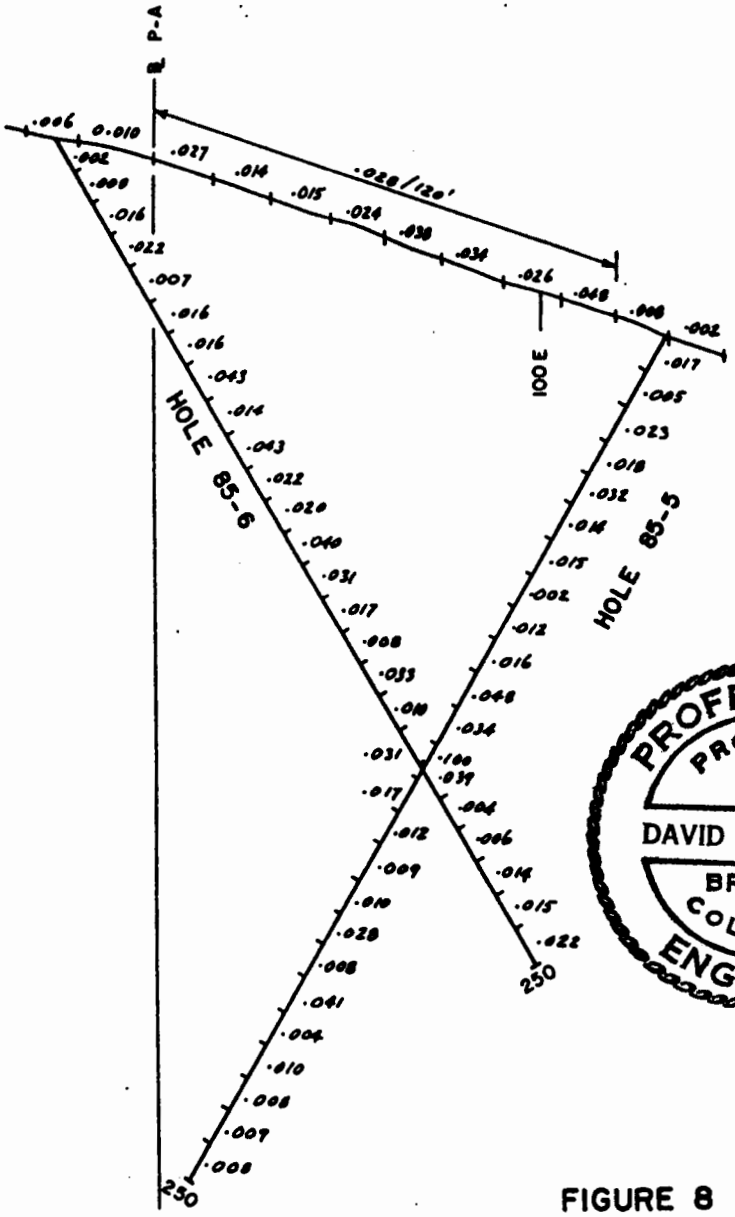
HOLES 85-3 & 85-4

TRENCH 20N

ANTONIUK DEPOSIT - MT. FREEGOLD, Y.T.

BIG CREEK RESOURCES LTD.

August, 1985



0 50 100 Feet

SCALE 1:600
MODIFIED FROM ORIGINIAL

Assays in oz/ton. Au by fire assay
at Chemex Labs, Vancouver

INCLUDED BY PERMISSION

FIGURE 8

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

VERTICAL CROSS-SECTION

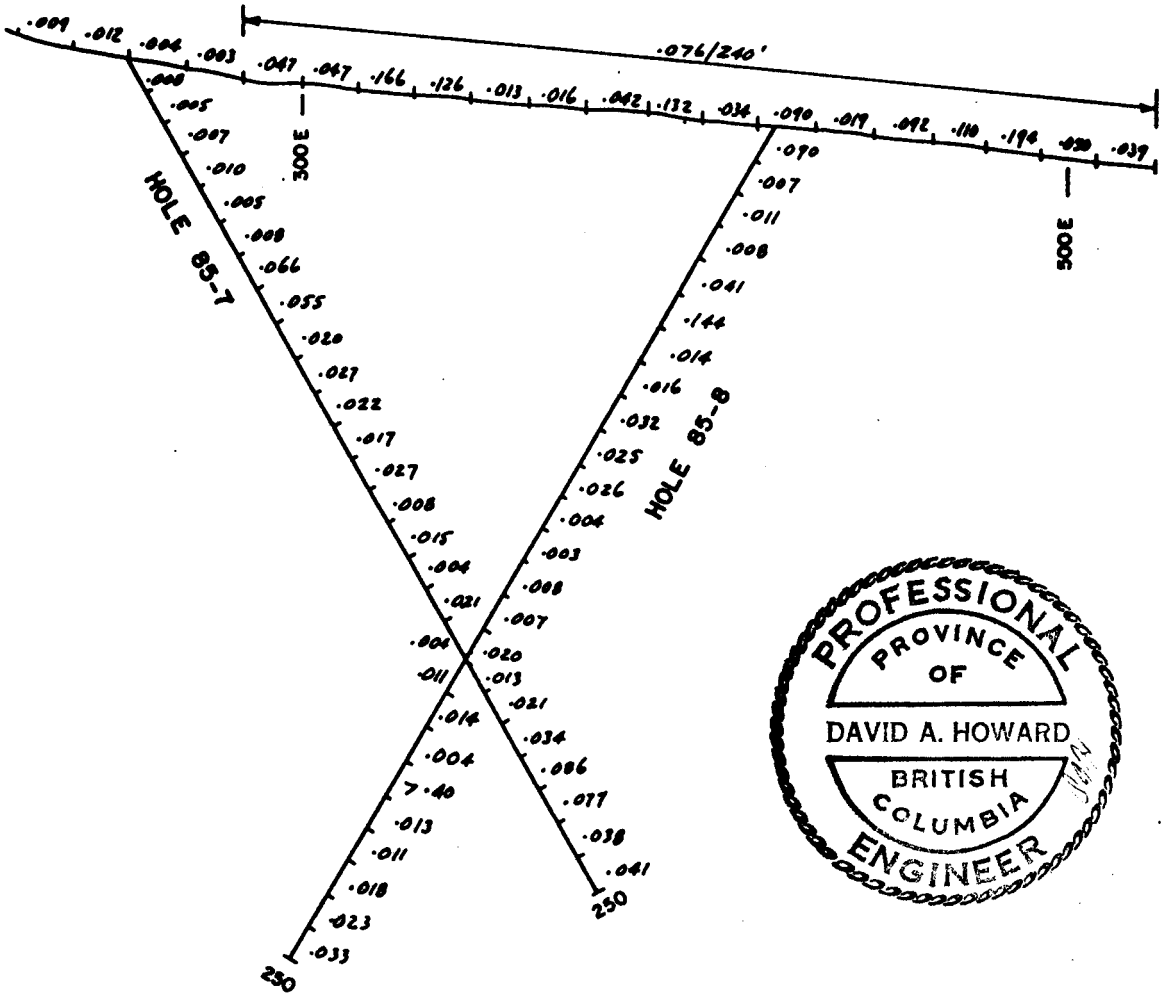
HOLES 85-5 & 85-6

TRENCH 22N

ANTONIUK DEPOSIT - MT FREEGOLD, Y.T.

BIG CREEK RESOURCES LTD.

August, 1985



0 50 100 Feet

SCALE 1:600
MODIFIED FROM ORIGINIAL

Assays in oz/ton Au by fire assay
at Chemex Labs, Vancouver
INCLUDED BY PERMISSION

FIGURE 9

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

VERTICAL CROSS-SECTION

HOLES 85-7 & 85-8

TRENCH 24N

ANTONIUK DEPOSIT - MT. FREEGOLD, Y.T.

BIG CREEK RESOURCES LTD.

August, 1985

A total of 27 trenches dug during the 1985 field season were sampled at 15 foot (4.6 metre) intervals. The composited gold assay results greater than 0.020 ounces per short ton (0.686 grams/mt) are shown on Figure 5. The weighted average grade of all mineralized intersections greater than 0.020 ounces per short ton (1.474 grams/mt) from all the trenches is 0.043 ounces gold per ton.

A comparison of the trench assays, percussion drill hole assays and 1975 & 1981 diamond drill assays from the plus 0.020 ounce per ton gold mineralized zones show that the average values obtained are very close, i.e. 0.043 vs. 0.048 vs. 0.042 respectively.

A comparison of Figures 4 and 5 will show the high correlation between the higher grade gold intersections in the diamond drilling and in the trenches. It also confirms the vertical nature and northerly trend of the mineralization.

The 24 hole - 2,189 metre, 1986 diamond drill program further refined the above grade estimates. Mr. E.S. Holt, P.Eng. calculated a mineral reserve (Holt, 1986) using the results of the 1986 drill program along with all of the previous drill and trench assay data. The text of Mr. Holt's report is included in this report as Appendix A. Due to the fact that all of the 1986 drill data is included in Mr. Holt's report (Holt, 1986) it will not be repeated in the text of this report, but the averaged assay intervals and areas of influence used by Mr. Holt are shown on the cross-sections (see Figures 10, 11, 12, 13, 14 and 15).

Table 3 is a summary of tonnage and grade estimates for Holt (1986) which shows that the estimates of the average grade from previous calculations (Howard, 1985) are comparable (0.042 vs. 0.037 at same cut-off grade).

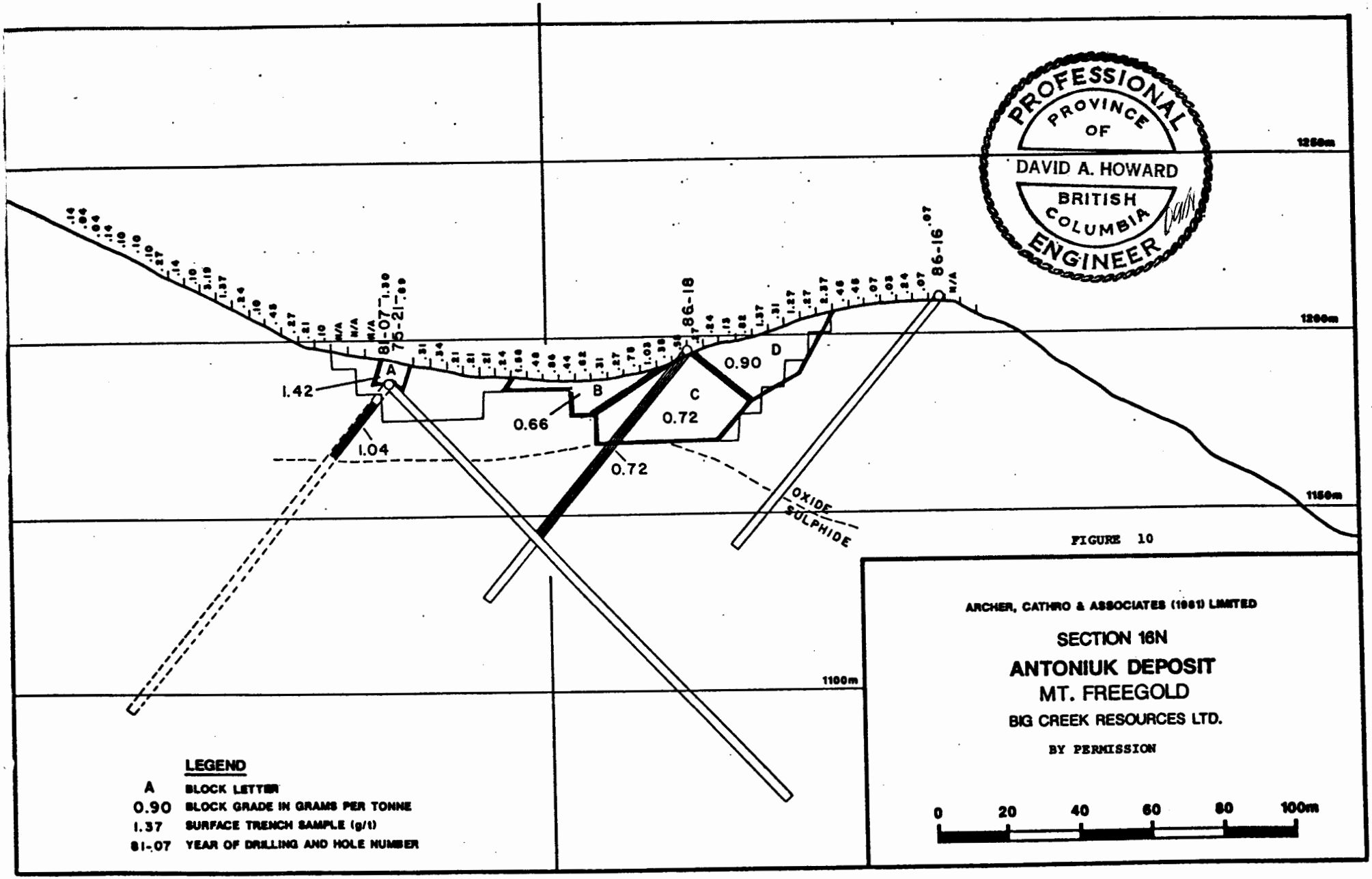
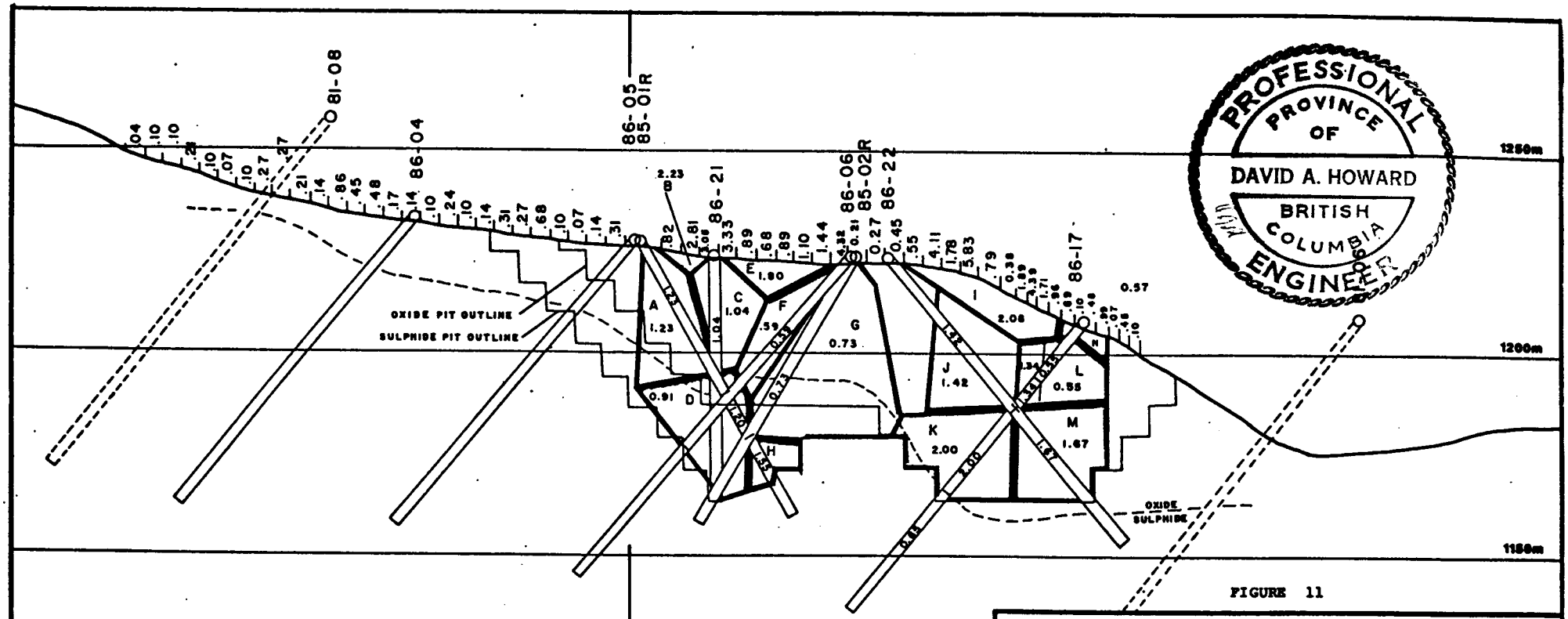

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
DAVID A. HOWARD
1250m
1200m
1150m
1100m
81-07
75-21
86-18
86-16
A
B
C
D
1.42
0.66
1.04
0.72
0.90
OXIDE
SULPHIDE
LEGEND
A BLOCK LETTER
0.90 BLOCK GRADE IN GRAMS PER TONNE
1.37 SURFACE TRENCH SAMPLE (g/t)
81-07 YEAR OF DRILLING AND HOLE NUMBER
ARCHER, CATHRO & ASSOCIATES (1981) LIMITED
SECTION 16N
ANTONIUK DEPOSIT
MT. FREEGOLD
BIG CREEK RESOURCES LTD.
BY PERMISSION
0 20 40 60 80 100m

FIGURE 10



PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
DAVID A. HOWARD

FIGURE 11

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

SECTION 18N

ANTONIUK DEPOSIT

MT. FREEGOLD

BIG CREEK RESOURCES LTD.

BY PERMISSION

0 20 40 60 80 100m

LEGEND

A	BLOCK LETTER
1.04	BLOCK GRADE IN GRAMS PER TONNE
.86	SURFACE TRENCH SAMPLE (g/t)
81-08	YEAR OF DRILLING AND HOLE NUMBER

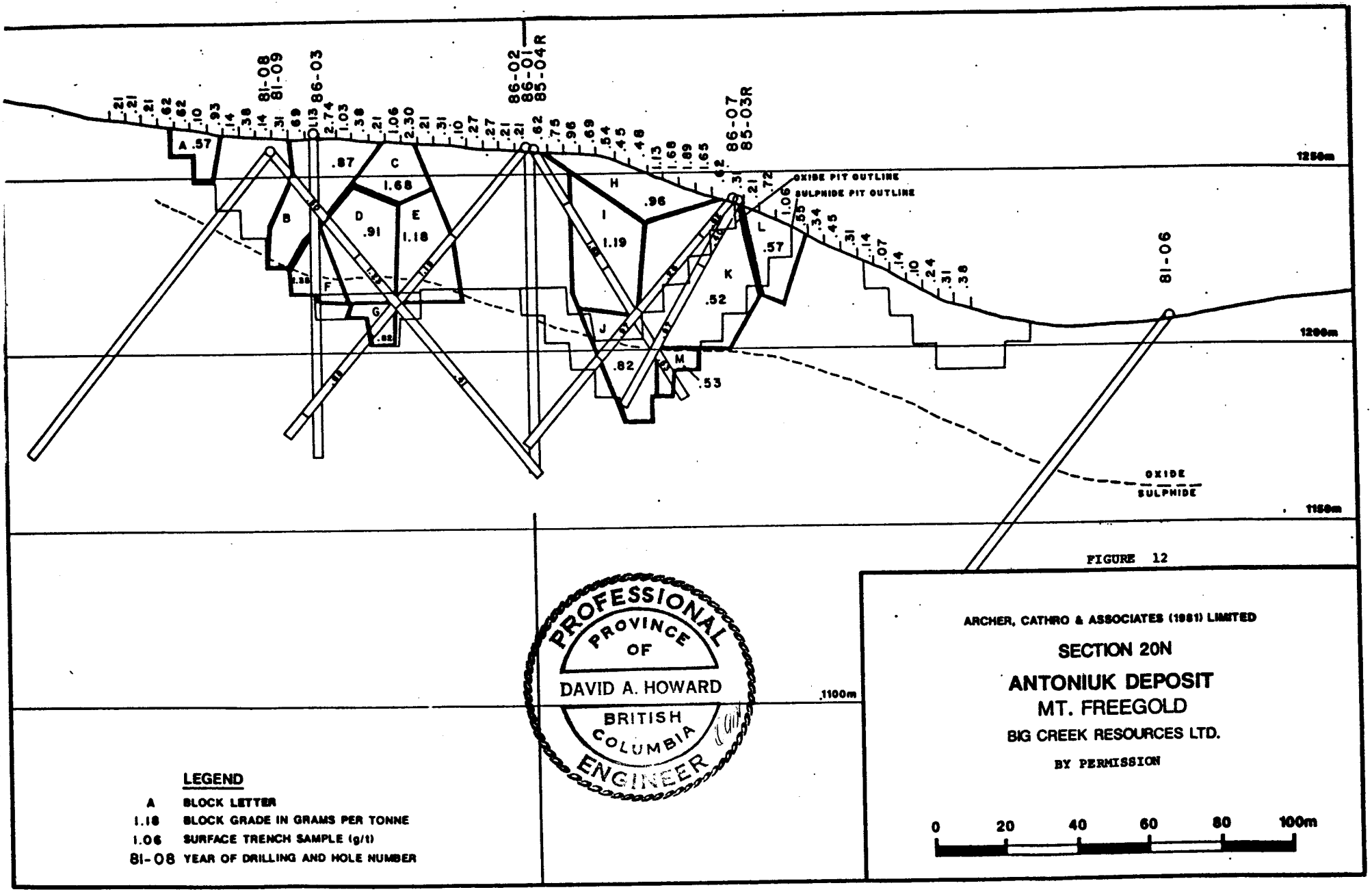


FIGURE 12

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PROVINCE OF BRITISH COLUMBIA
DAVID A. HOWARD

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

SECTION 20N

ANTONIUK DEPOSIT

MT. FREEGOLD

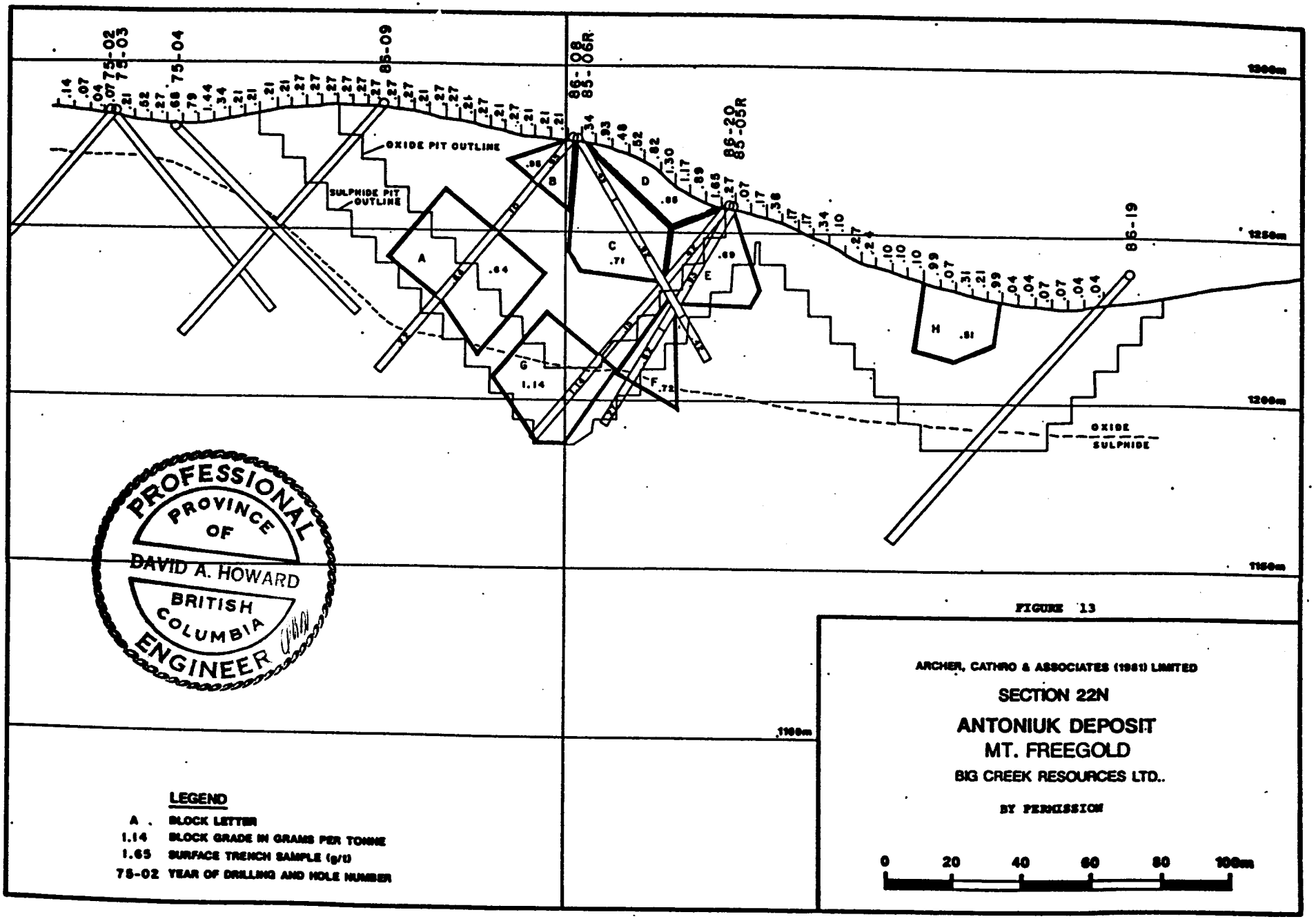
BIG CREEK RESOURCES LTD.

BY PERMISSION

0 20 40 60 80 100m

LEGEND

A	BLOCK LETTER
1.18	BLOCK GRADE IN GRAMS PER TONNE
1.06	SURFACE TRENCH SAMPLE (g/t)
81-08	YEAR OF DRILLING AND HOLE NUMBER



PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
DAVID A. HOWARD

FIGURE 13

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

SECTION 22N

ANTONIUK DEPOSIT

MT. FREEGOLD

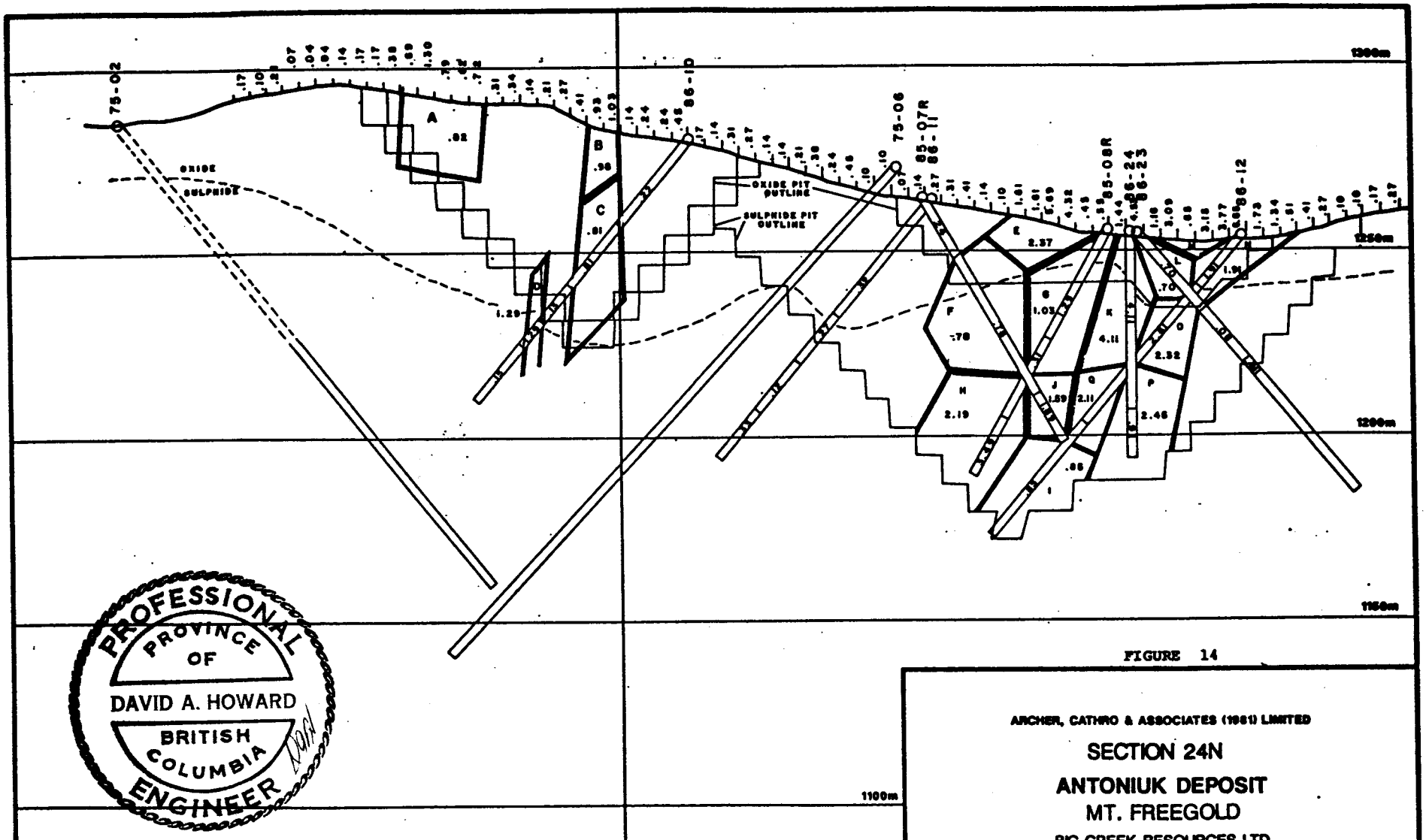
BIG CREEK RESOURCES LTD..

BY PERMISSION

0 20 40 60 80 100m

LEGEND

A . BLOCK LETTER
1.14 BLOCK GRADE IN GRAMS PER TONNE
1.65 SURFACE TRENCH SAMPLE (g/t)
75-02 YEAR OF DRILLING AND HOLE NUMBER



PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
DAVID A. HOWARD

FIGURE 14

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

SECTION 24N
ANTONIUK DEPOSIT
MT. FREEGOLD

BIG CREEK RESOURCES LTD.

BY PERMISSION

0 20 40 60 80 100m

LEGEND

- A — BLOCK LETTER
- 1.03 — BLOCK GRADE IN GRAMS PER TONNE
- 3.49 — SURFACE TRENCH SAMPLE (g/t)
- 75-02 — YEAR OF DRILLING AND HOLE NUMBER

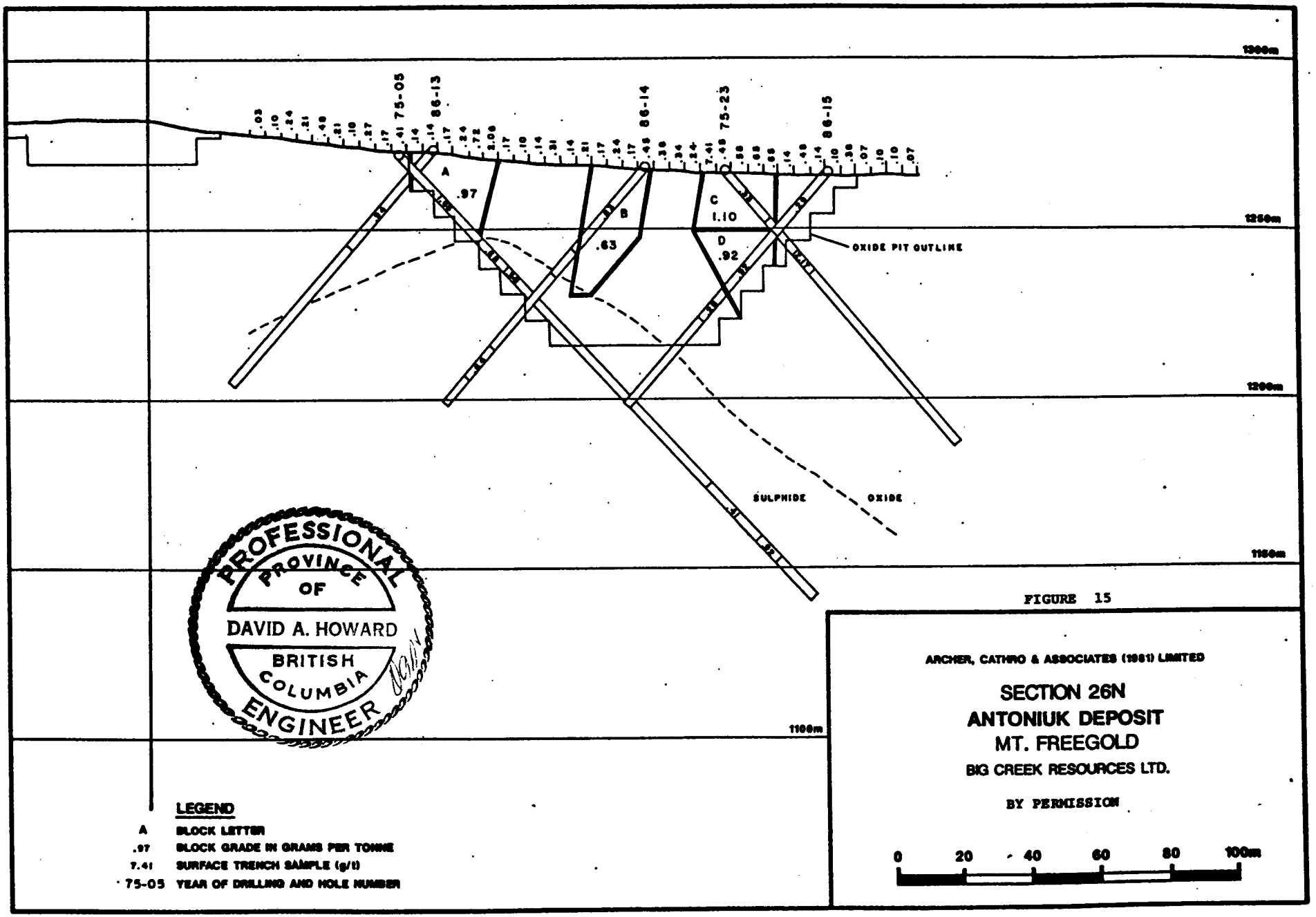


FIGURE 15

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

SECTION 26N
ANTONIUK DEPOSIT
MT. FREEGOLD

BIG CREEK RESOURCES LTD.

BY PERMISSION

0 20 40 60 80 100m

LEGEND

A	BLOCK LETTER
.97	BLOCK GRADE IN GRAMS PER TONNE
7.41	SURFACE TRENCH SAMPLE (g/t)
75-05	YEAR OF DRILLING AND HOLE NUMBER

TABLE 3

Mineral Reserves Modified After Holt, 1986

Category	Tonnes "Ore"	Grams Au/Mt	Troy Ounces Au/short/T	Tonnes Waste
Oxide .7 Cutoff	1,892,400	1.14	0.033	1,470,900
Oxide .5 Cutoff	2,621,900	0.99	0.029	741,400
Sulphide .7 Cutoff	1,068,000	1.52	0.044	2,430,300
Sulphide .5 Cutoff	1,093,600	1.50	0.044	2,405,300
Combined .7 Cutoff	2,961,000	1.28	0.037	3,901,200
Combined .5 Cutoff	3,715,500	1.14	0.033	3,146,700
Diluted .7 Cutoff	3,260,000	1.19	0.035	3,602,200
Diluted .5 Cutoff	4,087,000	1.06	0.031	2,775,200

Figure 16 outlines the areas used in Mr. Holt's reserve calculation as well as showing areas with future exploration potential.

METALLURGICAL TESTING

A metallurgical study of the Antoniuk deposit was financed by the Canada Centre for Mining and Energy Technology (CANMET), a branch within the Federal Department of Energy, Mines and Resources. Metallurgical testing was performed by Witteck Development Inc., Mississauga, Ontario. The study was conducted in two phases from 1985 to 1987 on four uncrushed bulk samples of surface material weighing a total of approximately five tonnes and uncrushed HQ core from two diamond drill holes (Main, 1988).

The summary by Main (1988) of the Witteck Development Inc. is as follows:

"Phase One of the study consisted of agitated leach tests and large scale pilot plant tests of oxidized surface material from the following sites:

Location	Interval	Sample Name (wt)	Trench Assay	Calc. Head Agitated	Calc. Head Bulk Test
Tr 20N	5 m	TR20 (835 kg)	2.74 g/t Au	2.25 g/t	0.69 g/t
Tr 24N	10 m	TR24 (980 kg)	5.01 g/t Au	1.67 g/t	1.65 g/t

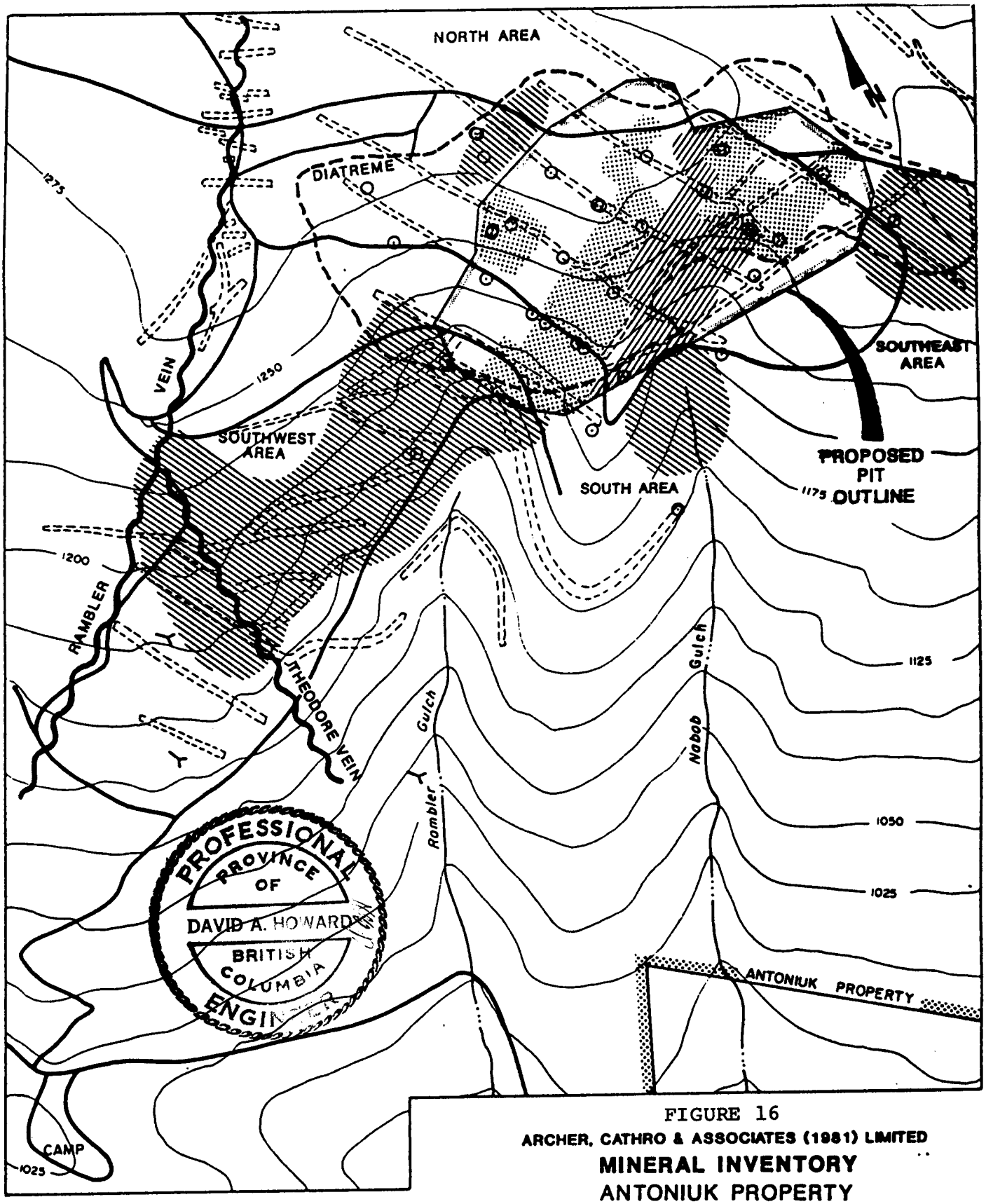


FIGURE 16

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

MINERAL INVENTORY

ANTONIUK PROPERTY

Mt. Freegold, Y.T.

BIG CREEK RESOURCES LTD.

BY PERMISSION

SCALE 1:5000

0 50 100 200 300 m

0 500 1000 ft

DRILL INDICATED RESERVES

DRILL INFERRED RESERVES

EXPLORATION POTENTIAL

DRILL COLLAR

ROAD

ADIT

TRENCH

DIATREME BOUNDARY

The detailed procedures for the test work is not included but is on file in the offices of Archer, Cathro & Associates (1981) Limited. The results of the agitated leach tests, an indication of ultimate recovery, indicated that 99% and 88% of the gold from Trench 20N and 24N samples, respectively, could be leached. The more definitive bulk tests confirmed a recovery of 95% in 15 days from Trench 20N but Trench 24N material only had recovery up to 50% within 44 days. This poorer recovery appears to be due to either the coarseness of the gold particles, a high variation of grade and/or mineralization in the sample material, or due to mineralogical complications. Petrological studies identified most of the gold occurs as coarse free particles associated with iron oxides located along fractures.

The first part of Phase Two, as described in Appendix Three, consisted of an in depth study of the leachability of a bulk sample from a 150 m wide zone of Trench 21N which was chosen because this interval appeared to have an average grade similar to that calculated for the deposit as a whole.

Location	Sample Name	Trench Assay	Calc. Head
Trench 21N	TR 21	1.03 g/t Au	0.77 g/t Au

However, the calculated grade of the sample is significantly lower (0.77 vs. 0.99 g/t Au) than the average grade of the deposit and it is certainly not representative of the better grade material which would be mined initially. As described in the Witteck report, the bulk sample was split into 31 separate samples which were tested by altering variables in a controlled fashion. The results of these tests, eight of which were by agitated leaching, eight by bottle roll testing and nine by column heap leaching indicate:

1. while crushing does not alter the final leachability, a crushing to minus 4.0 cm produces the optimum recovery rate;
2. leach extraction varies up to 87% in column tests over 69 days; and,
3. cyanide consumption can be reached to 2.0 kg/tonne by using relatively high lime quantities of 3.9 kg/tonne.

The second part of Phase Two consisted of column leach tests of HQ drill core from two diamond drill holes which had been sited to test both higher grade material (Trench 24N) and average to low grade material (Trench 18N).

This core contained material from unoxidized, as well as oxidized ore. The results of this testing is somewhat inconclusive as no head sample was taken for each test and hence no comparison of calculated head grade to sampled head grade could be made. Although cyanide consumption was high (ranging from 3.6 to 5.1 kg/tonne), Witteck consider that conditions should improve, as they did for Trench 21 samples, when the pH is lowered. The recovery from the hypogene (sulphide) core samples ranged from 28 to 40% but Witteck states that recoveries up to 60% can be expected from the sulphide ore under mining conditions.

1987 Metallurgical Test Pits

The two test pits were cut adjacent to the two vertical HQ diamond drill holes on Trenches 24N and 18N used for the second part of Phase Two (described above). The pits were dug to a depth of 8 m, as deep as the excavator could reach. The geology and gold content of these pits is shown in profile on Figures A6a and A6b, on the following page. A more meter wide zone on the side of each pit closest to the diamond drill hole was panel sampled by chipping approximately 15 kg of outcropping material from each depth interval of one metre, measured from the original trench surface. The top one metre of each pit consisted of rubble and was not sampled. Samples were sent to Chemex Labs, North Vancouver, for assaying and bottle roll cyanide leach testing. The head grade referred to on the table of assays below is the fire assayed grade of a 500 gram segment of the sample after being jaw crushed twice to 100% passing 1.0 cm. The leach grade is the calculated grade of a 500 gram sample of the same material after it has been bottle roll leach tested for 24 hours in a pH adjusted cyanide solution. Leach grades of over 100% (i.e. leach grade/head grade) indicate that gold is probably present in a coarse form which is causing sampling problems (resulting in head grades of lower than average value). The results of this sampling is shown below:

PIT A87-1 LEACH TEST - TRENCH 24N

(all grades in g/t gold)

Depth (m)	Head Grade	Head Grade (check assay)	Leach Grade	% Extracted
1-2	1.54	nss	0.93	60
2-3	8.67	9.94	9.81	113
3-4	11.66	13.51	12.75	109
4-5	6.82	9.70	7.27	142
5-6	3.57	3.91	5.52	155
6-7	2.85	3.60	4.35	153
7-8	1.23	1.40	2.54	206

nss = not sufficient sample

Corresponding metallurgical hole (86-24), about 1 m to the north:

0	to	3.35m	-	no sample
3.35	to	4.57m	-	0.01 g/t gold
4.57	to	5.49m	-	0.58
5.49	to	6.10m	-	0.34
6.10	to	7.33m	-	11.79
7.33	to	8.30m	-	5.49
8.30	to	8.84	-	14.98
8.84	to	10.36m	-	20.26

These sample present many difficulties in assessment. Although the leachability seems to improve downward, most of the results are greater than 100% and hence unreliable. The grade of the corresponding drill core to a depth of 6 m is much lower than the bulk sample and undoubtedly some gold was lost

in coring this surface material (core recovery from this interval was an average of 53%). A recalculation of grade over the interval with comparable samples indicates that the core samples have at least a 9% lower grade than the bulk samples and hence ore reserves should be increased accordingly. There is no evidence to suggest an increase of grade toward surface due to enrichment.

PIT A87-2 LEACH TEST - TRENCH 18N

(all grades in g/t gold)

Depth (m)	Head Grade	Leach Grade	% Extracted
1-2	2.43	1.85	76
2-3	0.82	0.92	109
3-4	0.31	0.38	126
4-5	0.24	0.17	67
5-6	0.27	0.24	88
6-7	0.45	0.24	52
7-8	1.20	0.51	43

Corresponding metallurgical hole (86-21), about 2 m to the south:

1.22	to	2.13m	-	0.27 g/t gold
2.13	to	3.60	-	1.92
3.60	to	4.27	-	0.69
4.27	to	5.79	-	0.24
5.79	to	6.71	-	0.58
6.71	to	7.62	-	5.00
7.62	to	8.53	-	4.46

Again, there is no clear trend to the results. The leachability is highly variable and the best recovery seems to be due to the presence of a small fault which occurs between 2 and 5 m depth. The grades at the top of the pit and drill hole correspond well but the two samples below 7 m do not, possibly due to the variability of mineralization or perhaps because the "top" of the pit has not been correlated with the top of the drill hole accurately. A comparison of corresponding samples indicate that the ore grade in this area should be decreased 8%. Because this result contradicts the increase of 9% indicated by Pit A87-1, no attempt was made to systematically alter any ore reserves away from the these pits."

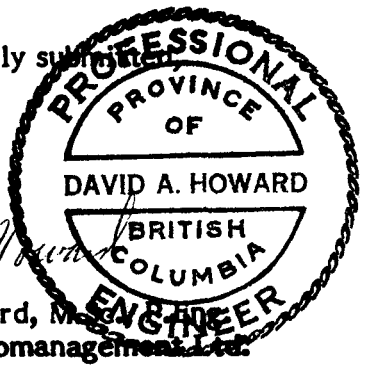
CONCLUSIONS AND RECOMMENDATIONS

Diamond and percussion drilling, trenching, and metallurgical test work has been successful in defining a mineral inventory of approximately 3 million tons of material grading 1.28 grams gold per metric tonne at a cut-off grade at 0.7 grams gold per metric tonne. In the writer's opinion, the above tonnage figure must be increased in order to make the Antoniuk deposit a mine. It is also the writer's opinion that the property contains an excellent potential for increasing the above tonnage. The proposed exploration areas outside the pit boundary as shown on Figure 16 all contain mineralized showings or occurrences which for various reasons have not been explored in great detail. It is therefore recommended that Big Creek Resources Ltd. conduct further exploration consisting of trenching, diamond drilling and bulk sampling of those areas with potential that lie outside the pit boundary as shown as Figure 16.

ESTIMATED COST OF PROPOSED EXPLORATION PROGRAM

Diamond drilling 1500 meters at $75.00 per meter	$ 112,500.00
Mob/demob	5,000.00
Bulldozer trenching / road building D7 - D8 150 hours at $100.00 per hour	15,000.00
Excavator trenching 200 hours at $150.00 per hour	30,000.00
Assaying 3000 samples at $15.00 per sample	45,000.00
Camp operation includes cook's wages	35,000.00
Transportation / expediting	5,000.00
Supervision / geology / labour	28,000.00
Sub-total	275,500.00
10% Contingency	27,550.00
Total	303,050.00
Say	$ 300,000.00

Respectfully submitted,



D.A. Howard, M.Sc., P.Eng.
D.D.H. Geomanagement Ltd.

D.D.H. GEOMANAGEMENT LTD.

CERTIFICATION

I, David A. Howard, of the City of Vancouver, Province of British Columbia, hereby certify as follows:

1. I am a geologist residing at 9040 Glenallan Gate, Richmond, B.C., with an office at 422 - 470 Granville Street, Vancouver, B.C.

2. I am a registered Professional Engineer of the Province of British Columbia. I graduated from Montana State University in 1964 and from the University of Washington in 1967.

3. I have practised my profession continuously since June, 1966.

4. The information contained in this report is derived from data contained in company files, government publications, a field examination of the property on August 10 and 11, 1985, and a total of 3 field seasons working on adjoining properties (1966, 1967, 1981).

5. I have no interest, direct or indirect in Big Creek Resources Ltd. property or in the securities of Big Creek Resources Ltd. or its affiliates, nor do I expect to receive any.

6. I consent to the use of this report in or in connection with the Prospectus, or in a statement of material facts relating to the raising of funds for this project.

Dated at Vancouver, B.C., this 18th day of January, 1988.

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
DAVID A. HOWARD

David A. Howard, M.Sc., P.Eng.

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APPENDIX A

RESERVE ESTIMATE - ANTONIUK GOLD DEPOSIT

prepared by

Holt Engineering Ltd.

RESERVE ESTIMATE

ANTONIUK GOLD DEPOSIT

MOUNT FREEGOLD, YUKON TERRITORY

September 25, 1986

Prepared by

HOLT ENGINEERING LTD.

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RESERVE ESTIMATE

INTRODUCTION

At the request of Mr. Robert Cathro, President of Archer, Cathro & Associates (1981) Limited, Holt Engineering Ltd. has carried out an independent reserve calculation for the Antoniuk gold deposit.

The project is at an intermediate stage of development following surface exploration, rotary and diamond drilling on sections spaced at 61 metre intervals and extensive surface trenching and sampling, but preceeding advanced metallurgical testing, definition drilling and related grade confirmation bulk sampling.

The reserve estimate is based on exploration results provided by Archer, Cathro & Associates which included drill sections, assay data, geologic interpretations and a preliminary pit design. Our assignment did not include certification of the accuracy of the information provided. However, it can be stated that the assaying, exploration work and preliminary engineering were all carried out by reputable firms or individuals experienced in the mineral industry. During the course of our independent calculation we did not encounter data which we considered to be unreliable.

The reserve estimate is based on our analysis of the data provided to us. We have not examined the property, however we are familiar with comparable bulk type gold deposits and with open pit reserve calculations in particular.

SUMMARY

The presently outlined reserves are based on 4950 metres of surface sampling, 607 metres of rotary drilling and 4775 metres of diamond drilling. The reserves are categorized into oxide and sulphide components which correspond with a geologic transformation resulting in different metallurgical characteristics.

The tonnage and grade estimates at two selected cut-off grades are summarized as follows:

Category	Tonnes Ore	Grams Au/t	Tonnes Waste	Strip Ratio
Oxide .7 Cutoff	1,892,400	1.14	1,470,900	0.78:1
Oxide .5 Cutoff	2,621,900	0.99	741,400	0.28:1
Sulphide .7 Cutoff	1,068,000	1.52	2,430,300	2.28:1
Sulphide .5 Cutoff	1,093,600	1.50	2,405,300	2.20:1
Combined .7 Cutoff	2,961,000	1.28	3,901,200	1.32:1
Combined .5 Cutoff	3,715,500	1.14	3,146,700	0.85:1
Diluted .7 Cutoff	3,260,000	1.19	3,602,200	1.11:1
Diluted .5 Cutoff	4,087,000	1.06	2,775,200	0.68:1

The reserves outlined above are MINEABLE RESERVES; that is, they are contained within the limits of trial open pits which adhere to normal open pit mining constraints.

It must be emphasized that the reserve grade and tonnage should not be regarded as precise measurements of quantity, tenor and distribution. They are, however, our best estimate based on the presently available data. Sufficient continuity has been demonstrated to afford a reasonably high level of confidence in

the overall metal content, but the position of those reserves within the deposit is less precise. In this regard, the estimates are considered appropriate for preliminary economic projections, but require refinement before detailed production scheduling can proceed with the desired level of confidence.

The present uncertainty regarding the configuration of the ore/waste boundaries effectively dictates that the bulk of the reserves be categorized as PROBABLE. The primary ingredient required to conform with a proven classification is increased geologic evidence that specific projections can be made with confidence.

The potential to expand the reserves and/or locate a high grade core are both realistic possibilities. Further potential to enhance the reserves lies in the need for a careful analysis of some conflicting results. They could represent a bias which has failed to reflect the true gold content of the ore.

RESERVE CALCULATION DETAIL

The reserve estimates presented in this report were compiled from six drill sections spaced at 61 metre intervals through the deposit. Geologic and assay data were available from a number of sources:

Trenching

1985 - 4950 lineal metres (1050 samples)

Diamond Drilling

1975 - 10 BQ holes - 1393 metres
1981 - 10 NQ holes - 1193 metres
1986 - 22 NQ holes - 2067 metres
1986 - 2 HQ holes - 122 metres

Rotary Drilling

1985 - 8 holes - 607 metres

Geochemical Soil Sampling

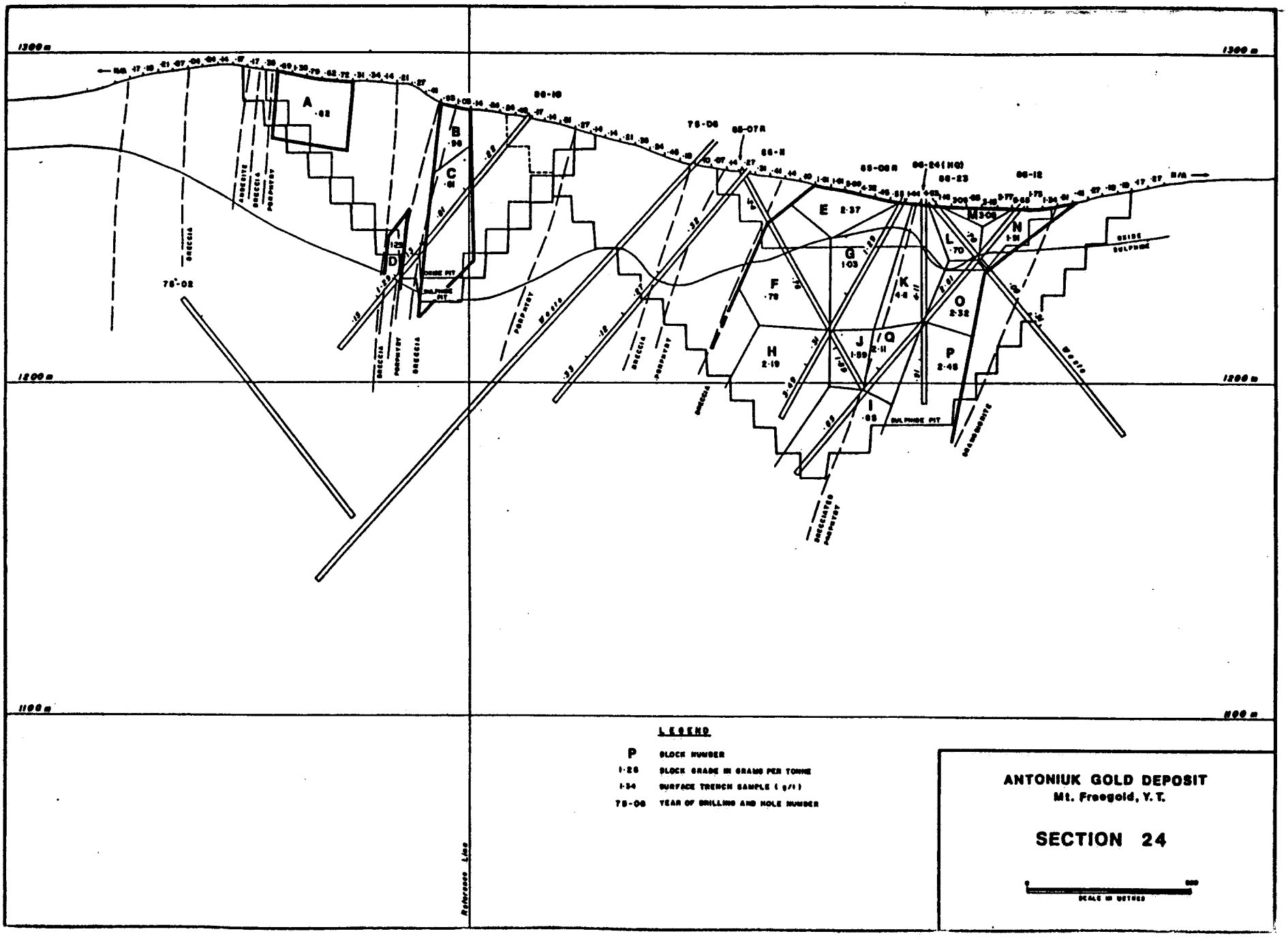
500 x 300m anomaly

The reserve estimates were prepared using standard engineering techniques. Reserve blocks were outlined on drill sections by using all available assay data, together with known geologic trends. Tonnages were calculated by planimetering block areas on section and projecting the ore or waste one half the distance to adjacent sections. Tonnage factors used were:

Oxide ore & Waste: 2.4 tonnes per cubic metre
Sulphide ore & Waste: 2.6 tonnes per cubic metre

Ore projections on section were limited to a maximum of 20 metres from assay data points.

Trial pit designs provided by Archer, Cathro were utilized to determine mineable reserves and waste to ore strip ratios. The



LEGEND

P	BLOCK NUMBER
1.26	BLOCK GRADE IN GRAMS PER TONNE
1.34	SURFACE TRENCH SAMPLE (g/t)
76-06	YEAR OF DRILLING AND HOLE NUMBER

ANTONIUK GOLD DEPOSIT
Mt. Freegold, Y. T.

SECTION 24

SCALE IN METRES

designs were based on an assumed 45 degree pit slope. Detailed pit planning to provide for ramps, minimum working areas and end slopes invariably increase the waste to ore ratios by a modest amount.

The geologic contact between oxide and sulphide material was also provided by Archer, Cathro. Their project geologist utilized drill core data and related geologic knowledge to determine the anticipated boundary. Although the contact is currently difficult to predict, the two rock types have characteristic visual features which should result in reasonably accurate pit sorting.

The drawing on the preceeding page is a reasonably typical drill section and is provided to illustrate the general geologic setting, the sampling density, the drill hole orientations, the individual reserve block grades, the pit outlines and the general reserve calculation method. Individual block tonnage and grade data is provided as an appendix to this report.

As will be noted, the estimated metal content of adjacent reserve blocks often varies considerably. While this lack of continuity indicates a need for some caution with regard to individual block grades, the somewhat erratic metal distribution found in the Antoniuk deposit is not untypical of gold deposits in general and, in fact, is less erratic than some exploited deposits which lived up to overall reserve estimate expectations.

RESERVE CLASSIFICATION

The terminology used in classifying mineral reserves is often a controversial subject. Many authorities have, however, agreed upon and defined certain terminology that provides relative dependibility of information. Based on these guidelines, the reserve classifications are defined as follows:

(1) Proven

Proven reserves are material for which tonnage is computed from dimensions revealed in outcrops, trenches, workings and drill holes and for which the grade is computed from the results of detailed sampling. The sites of inspection, sampling and measurements are so closely spaced and the geologic character is defined so well that the size, shape, and mineral content are well established. The computed tonnage and grade are judged to be accurate within limits which are stated, and no such limit is judged to differ from the computed tonnage or grade by more than 20 percent.

(2) Probable Reserves

Probable reserves or resources are material for which tonnage and grade are computed partly from specific measurements, samples or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, measurement, and sampling are too widely spaced or otherwise inappropriately spaced to outline the ore completely or establish its grade throughout.

Based on the somewhat rigid reserve classification outlined above, a significant portion of the reserves currently classed as "Probable" border on qualifying as "Proven". This is particularly true in the immediate vicinity of the surface trenches where geologic observations, measurements and detailed sampling have all

been undertakien and, to a lesser extent, in the central area of the east pit where the deposit's continuity is more predictable.

Fill-in drilling is not expected to significantly alter the overall tonnage and grade estimates; however, it is expected to modify the configuration of individual ore shoots. Based in part on this factor, it is proposed that a reasonably high degree of confidence is appropriate for the total predicted tonnage and grade within the pit; however, due to erratic metal distribution known to occur in the Antoniuk deposit, local discrepancies should be expected. In this regard, the estimates for reserve blocks on an individual basis should not be regarded as precise. Any discrepancies would, however, be compensating, with the result that confidence in the grade estimate increases as the population of reserve blocks increase.

The reserves primarily require improved geologic knowledge of the deposit, particularly regarding metal distribution patterns and geologic controls, to improve their status to the "Proven" category. A carefully executed check sampling program would also contribute to an increased confidence level.

The Antoniuk deposit is a potential heap leach project and as such may not warrant the detailed documentation of reserves normally required for Canadian gold deposits. Some heap leach projects commence production with very modest capital outlays and consequently do not justify major expenditures on the reserves.

In my opinion, some improvement in the confidence level is required, particularly in the vicinity of the initial pit. However, a major definition drilling program is probably not warranted unless major capital expenditures are anticipated.

Although the mineralization is somewhat erratic, examination of continuity both along strike and down dip has revealed that there is a reasonable degree of consistency. Although boundaries

between ore and waste blocks are not accurately defined at present, there is justification for assuming that pit sorting will meet with some success.

Mining dilution is provided at two stages of the reserve calculation:

1. Internal low grade assays were not excluded from block grades, and
2. A 10% dilution factor was added to the overall reserves.

The grade applied to the 10% dilution factor was 0.23 grams per tonne which is the average grade of all of the waste samples adjacent to ore blocks.

High grade assays have not been cut. With few exceptions, the assays are within the range expected to occur in a bulk type gold deposit.

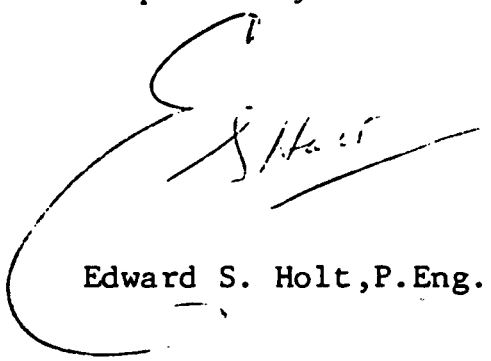
Careful analysis of the sampling data indicates that a possible bias on the low side may exist. Higher quality samples generally tend to indicate higher grade than adjacent low quality samples. Cases in point are:

1. The upper, low recovery, sections of many diamond drill holes indicated lower grade than adjacent surface sampling,
2. Rotary drilling results tend to be lower grade than diamond drilling, and
3. The large size HQ diamond drill holes, on average provided higher grade samples than the adjacent smaller size core and rotary drill holes.

It must be emphasized that the above mentioned observations are

not entirely consistent and within an erratic deposit could be simply coincidental. However, if a modest improvement in reserve grade is critical to the success of the project, then this is an area that warrants detailed investigation.

Respectfully submitted,



Edward S. Holt, P.Eng.

APPENDIX

OXIDE RESERVE SUMMARY

Category	Section	Tonnes	Grams Au/tonne
Oxide >.7 cutoff	16N	192,500	0.83
	18N	441,600	1.44
	20N	441,400	1.08
	22N	292,100	0.82
	24N	342,900	1.33
	26N	181,900	1.00
	TOTAL	1,892,400	1.14
Oxide <.7>.5	16N	35,500	0.66
	18N	86,000	0.57
	20N	213,400	0.53
	22N	307,900	0.62
	24N	-	-
	26N	86,700	0.63
	TOTAL	729,500	0.59
TOTAL >.5		2,621,900	0.99
Oxide Pit	16N	276,000	
	18N	775,200	
	20N	854,600	
	22N	686,300	
	24N	669,000	
	26N	102,200	
	TOTAL	3,363,300	

SULPHIDE RESERVE SUMMARY

Category	Section	Tonnes	Grams Au/tonne
Sulphide >.7 cutoff	16N	800	0.72
	18N	179,200	0.96
	20N	111,700	0.95
	22N	130,900	1.13
	24N	646,000	1.85
	26N	-	-
	TOTAL	1,068,600	1.52
Sulphide <.7 >.5	16N	-	-
	18N	4,000	0.59
	20N	17,800	0.53
	22N	1,200	0.64
	24N	-	-
	26N	2,000	0.63
	TOTAL	25,000	0.55
Total >.5		1,093,600	1.50
Sulphide Pit	16N	-	
	18N	232,000	
	20N	87,400	
	22N	1,115,000	
	24N	1,287,400	
	26N	777,100	
	TOTAL	3,498,900	

SECTION 16N

CATEGORY	BLOCK	PLANIMETER READING	METRIC TONNES	GRAMS AU
Oxide +.7	A	3.7	13,500	1.42
	C	31.0	113,500	0.72
	D	17.9	65,500	0.90
			192,500	0.83
Oxide +.5	B	9.7	35,500	0.66
Sulphide +.7	C	0.2	800	0.72
Oxide Pit		75.4	276,000	

SECTION 18N

Category	Block	Planimeter Reading	Metric Tonnes	Grams Au
Oxide +.7	A	16.5	60,400	1.23
	B	1.4	5,100	2.23
	C	11.5	42,100	1.04
	D	1.2	4,400	0.91
	E	6.2	22,700	1.80
	G	24.0	87,900	0.73
	I	10.9	39,900	2.08
	J	2.0	7,300	1.42
	K	19.1	69,900	2.00
	M	20.2	73,900	1.67
	O	4.1	13,800	1.34
			441,600	1.44
Oxide +.5	E	11.4	41,700	0.59
	L	9.5	34,800	0.55
	N	2.6	9,500	0.57
			86,000	0.57
Sulphide +.7	A	2.3	9,100	1.23
	D	18.7	74,100	0.91
	G	17.8	70,600	0.73
	H	5.4	21,400	1.55
	K	1.0	4,000	2.00
			179,200	0.96
Sulphide +.5	F	1.0	4,000	0.59
Oxide Pit		211.8	775,200	
Sulphide Pit		58.5	232,000	

SECTION 20N

Category	Block	Planimeter Reading	Metric Tonnes	Grams Au
Oxide +.7	B	23.9	87,500	0.87
	C	10.5	38,400	1.68
	D	18.8	68,800	0.91
	E	15.4	56,400	1.18
	F	4.7	17,200	1.35
	H	15.8	57,800	0.96
	I	26.0	95,200	1.19
	J	5.5	20,100	0.82
			441,400	1.08
Oxide +.5	A	6.7	24,500	0.57
	K	44.1	161,400	0.52
	L	7.5	27,500	0.57
			213,400	0.53
Sulphide +.7	D	4.1	16,200	0.91
	E	2.5	9,900	1.18
	F	4.6	18,200	1.35
	G	5.1	20,200	0.82
	J	11.4	45,200	0.82
	B	0.5	2,000	0.87
			111,700	0.95
Sulphide +.5	M	4.5	17,800	0.53
Oxide Pit		233.5	854,600	
Sulphide Pit		97.7	87,400	

SECTION 22N

Category	Block	Planimeter Reading	Metric Tonnes	Grams Au
Oxide +.7	B	10.2	37,300	0.95
	C	38.7	141,600	0.71
	D	12.7	46,500	0.85
	F	7.5	27,500	0.72
	G	10.7	39,200	1.14
			292,100	0.82
Oxide +.5	A	44.5	162,900	0.64
	E	21.0	76,900	0.69
	H	18.6	68,100	0.51
			307,900	0.62
Sulphide +.7	F	0.5	2,000	0.72
	G	32.5	128,900	1.14
			130,900	1.13
Sulphide +.5	A	0.3	1,200	0.64
Oxide Pit		187.5	686,300	
Sulphide Pit		281.2	1,115,000	

SECTION 24N

Category	Block	Planimeter Reading	Metric Tonnes	Grams Au
Oxide +.7gm	A	18.5	67,700	0.82
	B	6.6	24,200	0.98
	C	20.8	76,100	0.81
	D	1.8	6,600	1.29
	E	9.6	35,100	2.39
	F	9.2	33,700	0.78
	G	3.8	13,900	1.03
	K	4.9	17,900	4.11
	L	7.0	25,600	0.70
	M	2.3	8,400	3.08
	N	9.2	33,700	1.91
			342,900	1.33
Oxide +.5	Nil			
Sulphide +.7	C	0.2	800	0.81
	F	23.2	92,000	0.78
	G	21.2	84,100	1.03
	H	30.7	121,800	2.19
	I	19.1	75,700	0.85
	J	8.6	34,100	1.59
	K	14.0	55,500	4.11
	N	0.6	2,400	1.91
	O	13.0	51,500	2.32
	P	21.4	84,900	2.45
	Q	10.9	43,200	2.11
			646,000	1.85
Oxide Pit		182.8	669,000	
Sulphide Pit		324.7	1,287,400	

SECTION 26N

Category	Block	Planimeter Reading	Metric Tonnes	Grams Au
Oxide +.7 gm	A	19.4	71,000	0.97
	C	16.3	59,700	1.10
	D	14.0	51,200	0.92
			181,900	1.00
Oxide +.5	B	23.7	86,700	0.63
Sulphide +.5	B	0.5	2,000	0.63
Oxide Pit		25.8	102,200	
Sulphide Pit		196.0	777,100	