MAP NO.:	ASSESSMENT REPORT X	DOCUMENT NO:	092161
	PROSPECTUS	MINING DISTRICT:	WHITEHORSE
115 I 6	CONFIDENTIAL X	TYPE OF WORK:	TRENCHING, METALLURGICAL TESTING
	OPEN FILE		

REPORT FILED UNDER:	Archer, Cathro and Associates (1981) Ltd.		
DATE PERFORMED:	June 10 - September 7, 1987	DATE FILED:	March 28, 1988
LOCATION: LAT.:	62°16'N	AREA:	Mt. Freegold
LONG.:	137°06'W	VALUE $:	15,000.00
CLAIM NAME & NO.:	NAT 1-29 YA86843-871 NAT 30F-33F YA93013-016		

WORK DONE BY:	C.A. Main
WORK DONE FOR:	Big Creek Join Venture

DATE TO GOOD STANDING:	

REMARKS: #33 ANTONIUK
In 1987, 1200 m of bulldozer trenching was carried out. Bulk samples were taken from two excavator test pits and metallurgical tests were made. Drill-indicated reserves are estimated at 3 123 000 tonnes grading 1.31 g/t Au using a 0.7 g/t Au cutoff.

Indian and Northern Affairs Canada | Affaires indiennes et du Nord Canada

092161

M.R. file no.

R.M.M.R. file no.

Date forwarded 28 March 88

TRANSMITTAL FORM

OFFICE OF THE REGIONAL MANAGER MINERAL RIGHTS MAR 28 1988 WHITEHORSE YUKON TERRITORY

From ▶ Mining Recorder at: Whitehorse

To ▶ Regional Manager, Mineral Rights at Whitehorse, Y.T.

For action are:

- ☐ NEW APPLICATION FOR PLACER LEASE TO PROSPECT — Name
- ☐ RENEWAL APPLICATION PLACER LEASE TO PROSPECT — Name — Lease no.
- ☐ AFFIDAVIT OF EXPENDITURE ON PLACER LEASE — Name — Lease no.
- ☐ SECURITY DEPOSIT
- ☐ FINANCIAL ABILITY
- ☐ ASSIGNMENT OF PLACER LEASE NO. — From — To
- ☐ GROUPING APPLICATION UNDER SEC. 52(2) PLACER MINING ACT. — Owner
- ☐ DIAMOND DRILL LOGS — Claims — Claim sheet no.
- ☐ QUARTZ ASSESSMENT REPORT — Claims Nat 1-29, 30-33 YA86843-87 YA93013-YA93016 — Claim sheet no. 115-I-6

Type of report Trenching Metallurgical Testing. — Work submitted by

Cls. work performed on — $ req. for ren. applicat $15,000

Signature

REPLY ACTION

Date returned 7 April, 1988

Approved for amount required

Signature

802-6027-26 (6-86)

092161

ARCHER, CATHRO
& ASSOCIATES (1961) LIMITED
CONSULTING GEOLOGICAL ENGINEERS

1016 - 510 WEST HASTINGS STREET
VANCOUVER, B.C. V6B 1L8

(604) 688-2568

REPORT ON

TRENCHING PROGRAM AND METALLURGICAL TESTING

ANTONIUK PROPERTY

MT. FREEGOLD, YUKON TERRITORY

Mayflower	Lease 2751
Baker	Lease 2765
Connie	Lease 2766
Jim	Lease 2768
Donalda 1-9	Lease 2773-81
Donalda 13	Lease 2782
Nat 1-29	YA86843-YA86871
Nat 30F-33F	YA93013-YA93016
Peggy 1	YA95146
Peggy 2F-4F	YA95147-YA95149
Peggy 5F	YA96268

Latitude 62°16' Longitude 137°06' NTS 115I/6

092161

Whitehorse Mining District

EIP Designation Number 86-019

BIG CREEK JOINT VENTURE

January, 1988

C.A. Main, B.Sc.

Work done between June 10 and September 7, 1987

This report has been examined by the Geological Evaluation Unit under Section 53 (4) Yukon Quartz Mining Act and is allowed as representation work in the amount of $ 5,000.00.

P. G. Jensen

for Regional Manager, Exploration and Geological Services for Commissioner of Yukon Territory.

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SUMMARY AND RECOMMENDATIONS

Gold mineralization was first discovered about 1931 at the Rambler Vein, which is adjacent to the Antoniuk deposit. A geochemical survey in 1974 led to discovery of the Antoniuk mineralization, which was drilled in 1975, 1981 and 1986, and bulldozer trenched in 1985. This data was used by E.S. Holt, P.Eng, in September, 1986 to calculate ore reserves based on preliminary open pit designs. The 1987 program, discussed in this report, consisted of 1200 m of bulldozer trenching and two excavator test pits. The program was managed by Archer, Cathro & Associates (1981) Limited on behalf of Big Creek Resources Ltd., which has optioned the property from Discovery Mines Ltd.

The 1987 bulldozer trenches were designed to obtain geological information. Exposures in the trenches showed that gold mineralization seems controlled by northeast-trending structures which are offset in a complex fashion by later northwest-trending faults. The trenches were oriented perpendicular to the drill sections used to calculate ore reserves and the assay results do not alter these reserves significantly, except one additional block of ore was discovered in a previously unexplored area. This has increased the ore tonnage and improved the stripping ratio slightly.

The excavator test pits were intended to investigate the presence of surface supergene enrichment which may effect the metallurgical properties of surface mineralization, as well as the gold content. Metallurgical testing of samples from these pits was inconclusive because the gold appears to be present in coarse, free form and assaying of samples was inconsistent. Assaying did show that the average grade of the surface material in one pit is at least 9% higher than indicated by drill core of the same material. The other pit, sited

in much lower grade material, had an average grade 8% lower than the equivalent core. The net effect was to increase the grade of the ore reserves slightly.

The Antoniuk ore reserves, using Holt's data and amended by the 1987 results, are as follows:

Drill indicated, undiluted, open pit mineable reserves:

(A) Using a 0.5 g/t (0.015 opt) gold cutoff

Category	Tonnes (Tons)	Grade g/t (opt)	Cumulative Gold - gm (ozs)	Waste Tonnes (Tons)	Waste/Ore Ratio
OXIDE	2,783,900 (3,039,000)	1.040 (.030)	2,895,000 (92,000)	579,000 (632,000)	0.21
SULPHIDE	1,093,600 (1,194,000)	1.500 (.044)	1,640,000 (52,000)	2,405,000 (2,626,000)	2.20
COMBINED	3,877,900 (4,234,000)	1.160 (0.34)	4,498,000 (143,000)	2,985,000 (3,258,000)	0.77

(B) Using a 0.7 g/t (0.020 opt) gold cutoff

OXIDE	2,054,400 (2,243,000)	1.200 (.035)	2,465,000 (79,000)	1,309,000 (1,429,000)	0.64
SULPHIDE	1,068,000 (1,166,000)	1.520 (.044)	1,623,000 (52,000)	2,430,000 (2,653,000)	2.28
COMBINED	3,123,000 (3,409,000)	1.310 (.038)	4,091,000 (130,000)	3,739,000 (4,082,000)	1.20

In addition to these drill indicated reserves, the property contains inferred reserves of 1,135,000 tonnes that lie within the proposed pit outlines. Exploration potential also exists for additional oxide mineralization peripheral to the proposed pit and for a substantial tonnage of sulphide mineralization below the drill indicated reserves.

During 1987, metallurgical studies and a preliminary feasibility study on the Antoniuk deposit were completed by Witteck Development, Inc. of Mississauga, Ontario. These studies, which were funded by the Canadian Centre for Mines and Energy Technology (CANMET), are the result of sampling and testing conducted during 1985 and 1986.

The initial metallurgical studies indicate that cyanide heap leaching would require about 2.0 kg/tonne of cyanide and 3.9 kg/tonne of lime to achieve optimum leachability for material crushed to minus 4.0 cm. Oxidized material has a leachable recovery ranging from 44% (due to coarse gold particles requiring a longer leach time) to 95% while unoxidized material has a recovery of 40 to 60%. Witteck predicts an overall recovery for the deposit of 75%.

The feasibility study indicates that, while the Antoniuk deposit is not economic at the present time, conditions need change only marginally to make it a profitable venture. The report recommends that to improve the economics there should be:

1. exploration to upgrade the "possible" reserves to "drill indicated" and identify additional reserves. This would reduce mining costs through economy of scale, and may improve ore grade;
2. studies to optimize the pit location to reduce mining costs;
3. further metallurgical testing to lower reagent consumption; and,
4. test mining to calculate the optimum mining parameters.

The feasibility of test mining 80,000 tonnes from the Antoniuk deposit has been reported separately.

Further exploration should be directed toward these objectives:

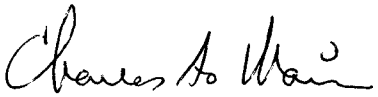
1. continue sampling within the main Antoniuk deposit to identify and increase the indicated reserves. This will require 150 hours of bulldozer trenching and 1500 m of drilling in 17 short holes, at a budgeted cost of $250,000;
2. continue test pitting to determine the accuracy of drill hole assays of surface material. This will require 200 hours of excavator trenching at a budgeted cost of $50,000; and,

3. sample areas peripheral to the known reserves, in particular around the Theodore Vein and to the south and southeast of the main deposit. This will require 150 hours of bulldozer trenching at a budgeted cost of $50,000.

Opportunities for acquiring or developing additional reserves of similar mineralization in close proximity to the Antoniuk Property should be aggressively pursued.

Respectfully submitted,

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED



/mc

C.A. Main, B.Sc.

INTRODUCTION

The Antoniuk Property covers an oxidized heap leach gold target that was acquired by Archer, Cathro & Associates (1981) Limited by option and staking in March, 1985. The option was assigned to Big Creek Resources Ltd. (formerly Nordac Mining Corporation) and was explored with trenching, drill sampling and preliminary metallurgical testing later in the year in a joint venture with Permian Resources Ltd. A program of grid diamond drilling in 1986 served as the basis for mineral reserve calculations and to provide samples for metallurgical testing. Permian presently owns some 25% of the Discovery option but is being diluted at the rate of 1% per $7,000 of expenditure. In 1987, Big Creek Resources Ltd. formed Big Creek Joint Venture (BCJV) with Rexford Minerals Ltd. to continue development. The 1987 program, which cost approximately $45,000, consisted of bulldozer trenching and excavator test pitting. It was planned and managed by Archer, Cathro & Associates (1981) Limited with C.A. Main as project manager. Geologist J.L. Duke mapped the trenches and F. Anderson sampled and mapped the excavator pits.

PROPERTY

The Antoniuk Property consists of fourteen surveyed and leased claims held under an option agreement and amending agreements dated March 12, 1985 and 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 and are part of the same 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, 1988*
Peggy 2F-4F	3	YA95147-YA95149	July 14, 1988*
Peggy 5F	1	YA96268	September 19, 1988*
	38		

*Assessment will be filed to extend these dates to spring, 1993.

LOCATION AND ACCESS

The Antoniuk Property is situated on Mt. Freegold, which lies at the southeast end of the Dawson Range (see Figures 1 to 3 on the following pages), 65 km by all-weather road northwest of Carmacks, Yukon. It is located at latitude 62°16'N and longitude 137°06'W within NTS claim sheet 115I/6.

The 1987 program was conducted from a permanent campsite situated 700 m east of the former LaForma Mine and 900 m west of the Carmacks Road turnoff. Travel between camp and the deposit was by means of four-wheel drive vehicle.

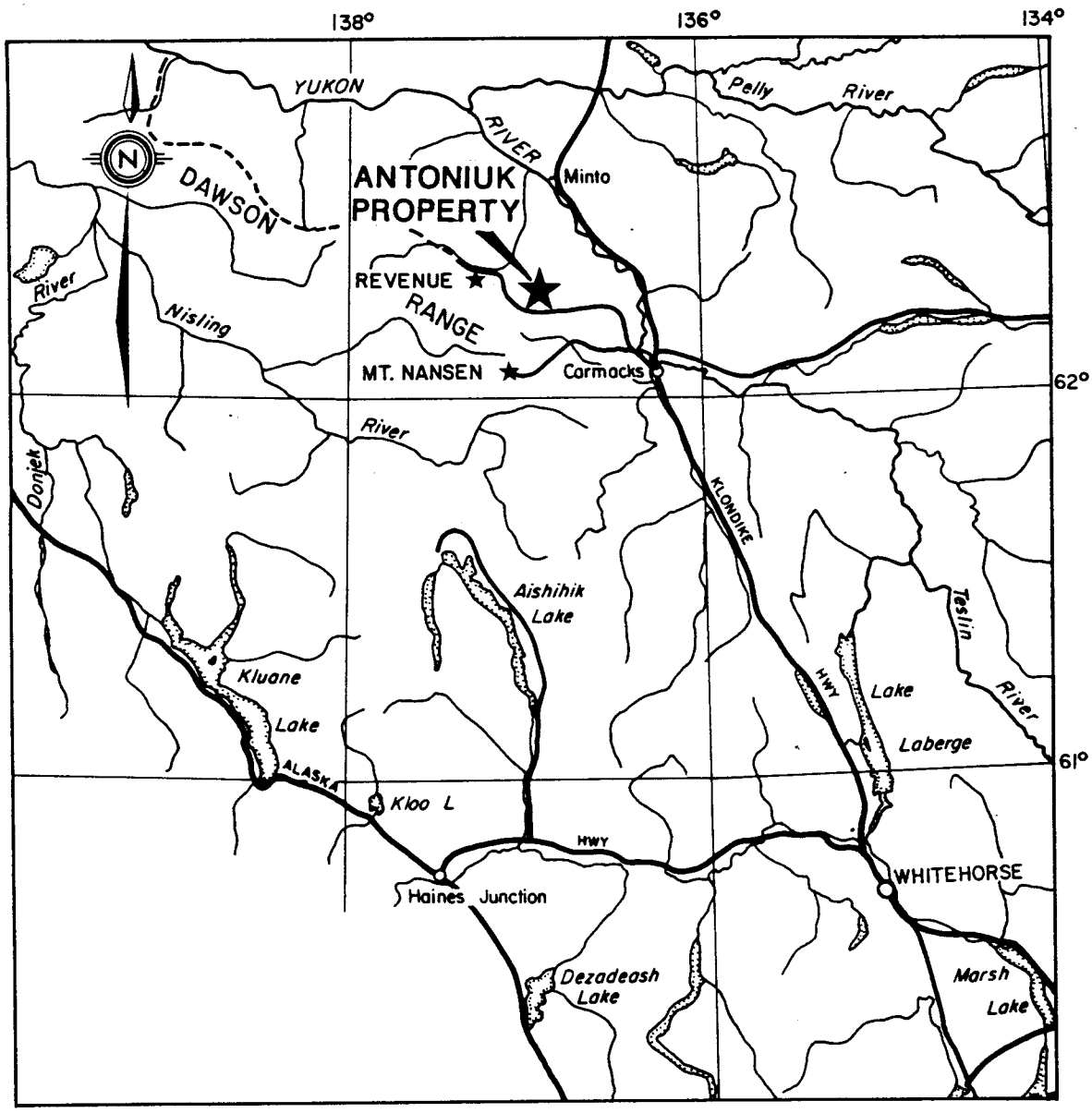


Figure A-1

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

GENERAL LOCATION MAP

ANTONIUK PROPERTY

MOUNT FREEGOLD, Y.T.

BIG CREEK RESOURCES LTD.
REXFORD MINERALS LTD.
PERMIAN RESOURCES LTD.

SCALE 1:2,000,000

0 20 40 60 80 100 Km

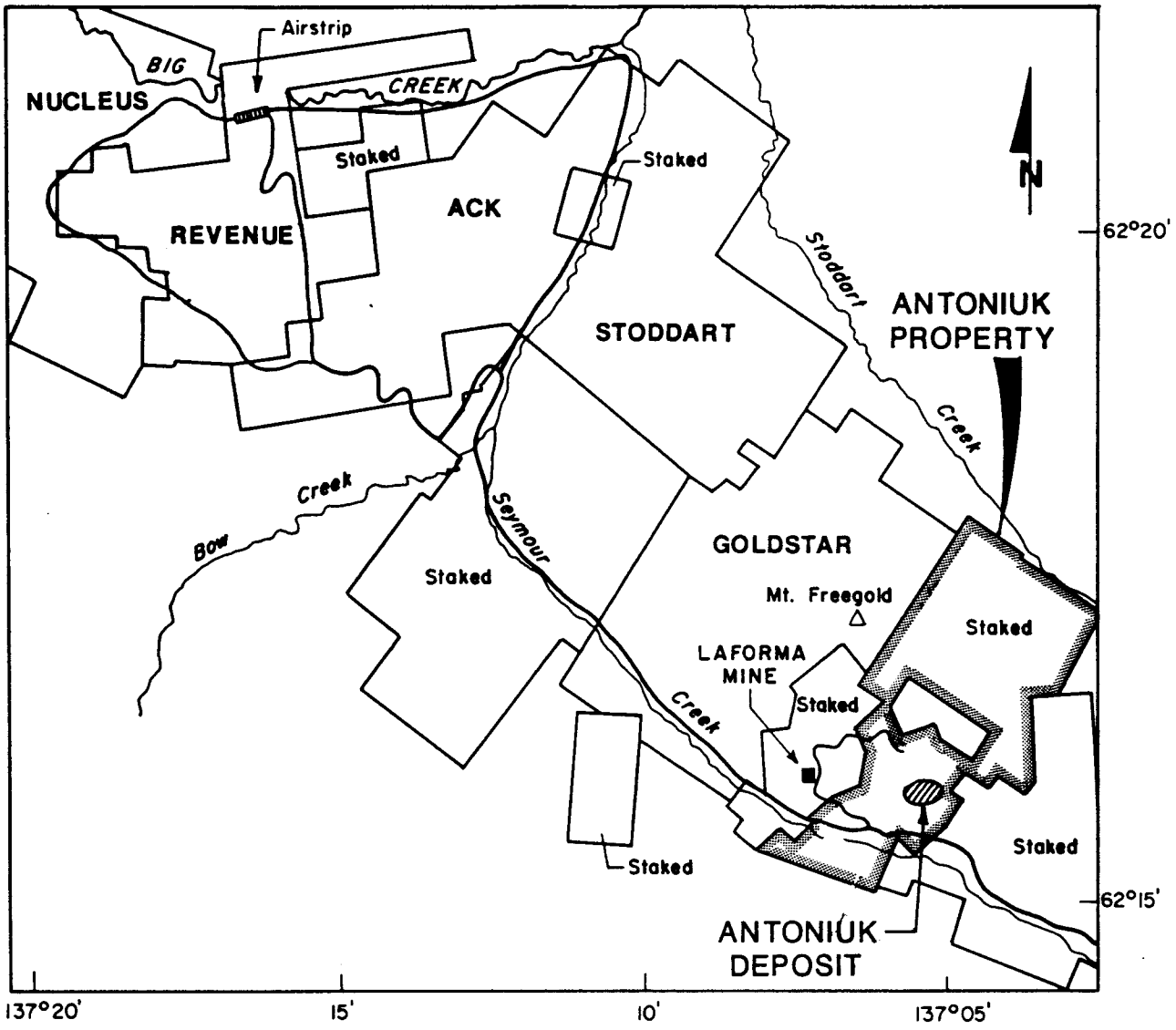


Figure A-2

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

PROPERTY LOCATION MAP

ANTONIUK PROPERTY

MOUNT FREEGOLD, Y.T.

BIG CREEK RESOURCES LTD.
REXFORD MINERALS LTD.
PERMIAN RESOURCES LTD.

SCALE 1:100,000

0 1 2 3 4 5 Km

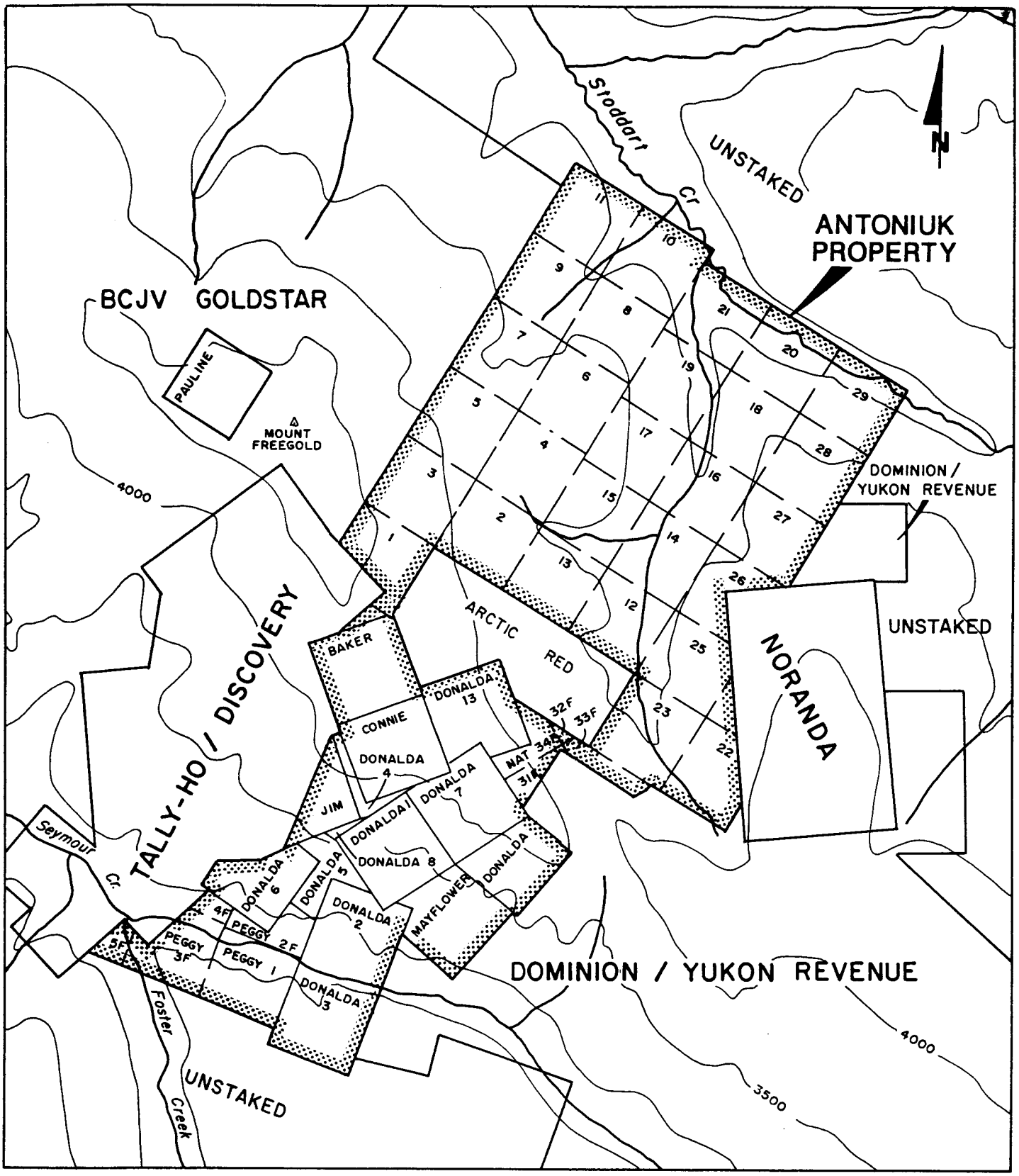


Figure A-3

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

CLAIM LOCATION MAP

ANTONIUK PROPERTY

BIG CREEK RESOURCES LTD.
REXFORD MINERALS LTD.
PERMIAN RESOURCES LTD.

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

0 500 1000 1500 m

0 1500 3000 4500 ft

PHYSIOGRAPHY AND GEOMORPHOLOGY

The Antoniuk Property lies on the east side of Mt. Freegold covering a broad, gentle sloping ridge separating Seymour Creek to the south and Stoddart Creek to the north. Seymour Creek has cut a deep V-shaped valley and the south-facing part of the property adjacent to Seymour Creek is much steeper than the rest of the property, with slopes locally exceeding 30°. Local elevations range from 850 m on the floor of Seymour Creek Valley to 1300 m on the crest of the ridge. Vegetation ranges from spruce and poplar trees on south- and east-facing hills to stunted black spruce and thick moss on north- and west-facing slopes.

Soil profiles typically include 1 to 10 cm of "A" horizon organics, 0 to 100 cm of volcanic ash, 5 to 50 cm of "B" horizon soil and 100 to 300 cm of "C" horizon soil over deeply weathered bedrock. Glacial and fluvial deposits are rare except on the floor of Seymour Valley. Permafrost is extensive, particularly on north- and west-facing slopes. Outcrop and talus is restricted to ridge crests and steep southeast-facing slopes.

HISTORY AND PREVIOUS WORK

The Antoniuk Property has a long work history extending back to the discovery of the Rambler Vein in 1931 by Afe Brown and George Fairclough during the staking rush triggered by the discovery of gold on BCJV's nearby Goldstar property. After the initial claims lapsed, they were restaked as the Mayflower and Donalda claims in November, 1939. The G-3 Vein, located 800 m west of the Antoniuk deposit, was also located in 1931 and was developed as the LaForma Mine, which was in production briefly in 1939 and 1965.

Exploration on the Rambler Vein between 1931 and 1974 was directed toward gold-bearing quartz veins resembling the G-3 Vein. It included four adits between 7 to 45 m long, ten diamond drill holes (+300 m) and approximately fifty hand trenches and forty bulldozer trenches to bedrock. This early exploration gave disappointing results.

Commencing in 1965, Discovery Mines performed mapping of the property in conjunction with development of the LaForma Mine. A geochemical sampling program was conducted over the claims in 1974 under the direction of Terry Antoniuk which outlined a strong soil gold-arsenic anomaly extending about 500 by 300 m. This survey outlined poorly exposed porphyritic and brecciated intrusive rocks containing low grade gold mineralization, now called the Antoniuk deposit. Subsequent exploration included further detailed mapping and soil sampling, geophysical surveys, ten diamond drill holes (1393 m) in 1975, and ten diamond drill holes (1193 m) in 1981. The 1981 work, which was managed by Archer, Cathro, was financed by Arctic Red Resources Corp. under a brief option and performed in conjunction with a larger exploration program that covered the remainder of Discovery Mines' claims on Mt. Freegold, including the LaForma Mine. Arctic Red's interest in the Antoniuk portion of the option was terminated in 1982.

In 1985, the Antoniuk Property was optioned by Archer, Cathro and assigned to Big Creek Resources and Permian Resources. Later that year, a program of systematic bulldozer trenching provided bedrock exposure for rock sampling and mapping. About 4950 m of trench length was excavated within an area approximately 1300 m long by 400 m wide. Some 1050 channel samples were collected at 4.6 m intervals and eight rotary percussion holes (607 m) were drilled within four of the better grade zones outlined by trenching.

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 were drilled under contract by E. Caron Diamond Drilling Ltd. of Whitehorse to systematically test the mineralized area outlined by previous bulldozer trenching. Two of the initial holes were drilled with a vertical orientation but fracturing in the core was found to be steeply dipping and the remaining 20 NQ holes were drilled at -50° toward grid west, at azimuth 320°, usually to depths of 91 m. Two vertical HQ holes were drilled to 61 m to provide material for metallurgical testing of oxidized, partially oxidized, and unoxidized mineralization by Witteck. The 1986 field work is described in a report by R.J. Cathro and C.A. Main dated November, 1986. Reports on all metallurgical work by Witteck were received in late spring, 1986 (Phase One) and March, 1987 (Phase Two).

1987 PROGRAM

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. The results of this work are discussed below.

GEOLOGY AND MINERALIZATION

The geology of the Antoniuk deposit is shown in plan on Figure A-4 (in pocket). Trenching and drilling has outlined a crudely elliptical, plug-like diatreme of heterolithic breccia, the outline of which is shown on Figures A-4 and A-5 (in pocket). This breccia body cuts an igneous complex near the centre of a geochemical gold-arsenic soil anomaly.

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 axis of the breccia body trends west-northwest roughly parallel to major regional faults such as the Pal Fault and Camp Fault, which lie 700 and 900 m to the south. On the northeast side of the breccia body, quartz porphyry

and quartz-feldspar porphyry dykes are abundant. They commonly trend northeast, subparallel to a series of weak faults as well as the Rambler Vein, which lies just west of the breccia body, and the G-3 (LaForma) Vein, which lies a farther 800 m west. On the southwest side of the breccia body, porphyritic rocks are uncommon and the small dykes that are present tend to strike north. This suggests that the diatreme may have intruded along a zone of structural dislocation that also exerted some control on the emplacement of the porphyry bodies.

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 unoxidized hypogene zone, the principal sulphide mineral 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 A-7 following page 20) 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 m.

1987 Trench Geology

The geology and assays from trenches A87-1 and A87-2 are shown on Figures A-4 and A-5. The two 1987 bulldozer trenches were oriented northeasterly, which is perpendicular to the previous bulldozer trenches and diamond drill sections. This orientation was useful as it revealed structures and contacts which are subparallel or at an acute angle to the previous trenches and hence previously poorly exposed. In the 1987 trenches, a system of northeasterly- and northwesterly-trending faults can be seen to crosscut the Antoniuk deposit in a complex fashion. The northeasterly faults appear to be related to the gold mineralization in that they are more common in well mineralized areas or they bound areas with better gold values. The northwesterly faults seem to be later than, and appear to offset, the earlier northeasterly faults. This system of faults still requires further clarification, especially along the south and east side of the breccia body.

Two zones of ore grade mineralization were intercepted by the trenches. In Trench A87-1, an interval of 55 m graded 1.75 g/t Au. This occurs in an area which had already been identified as having ore of this grade. The rest of the trench had lower grades, of about the same value as predicted by earlier trenching. In Trench A87-2, an interval of 50 m graded 1.92 g/t Au. This

occurs in an area which had been previously poorly sampled and where waste had been predicted. The result adds 162,000 tonnes to the ore reserves (and reduces the waste tonnage accordingly with a slight improvement of waste/ore ratio). The balance of the trench had low values.

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. 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.

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	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

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 showed that most of the gold occurs as coarse free particles associated with iron oxides located along fractures.

The first part of Phase Two 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	TR21	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. 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 reduced 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 are 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 considers 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.

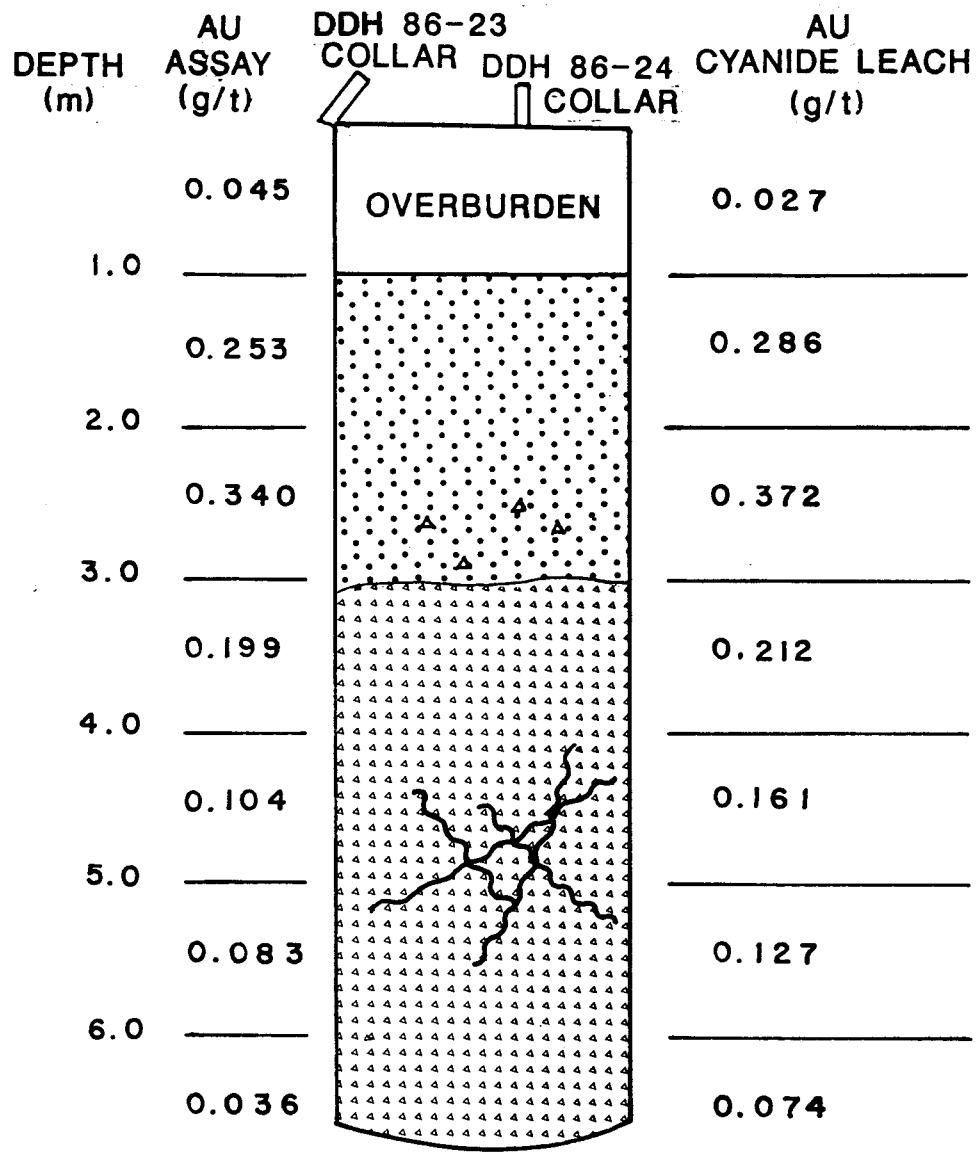
1987 Metallurgical Test Pits

The two test pits were cut adjacent to the two vertical HQ diamond drill holes on Trenches 24N (Pit #1) and 18N (Pit #2) 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 one metre 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 Ltd., 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 gm 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 gm 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 are shown below.

X-SECTION

FACING

LINE 24N, 1+15E



LEGEND

HETEROLITHIC BRECCIA

BRECCIA TEXTURES

SERICITIC QUARTZ-FELDSPAR PORPHYRY

FAULT

FIGURE A-6a

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

METALLURGICAL TEST PIT #1

ANTONIUK DEPOSIT

BIG CREEK RESOURCES LTD.
REXFORD MINERALS LTD.
PERMIAN RESOURCES LTD.

SCALE 1:50

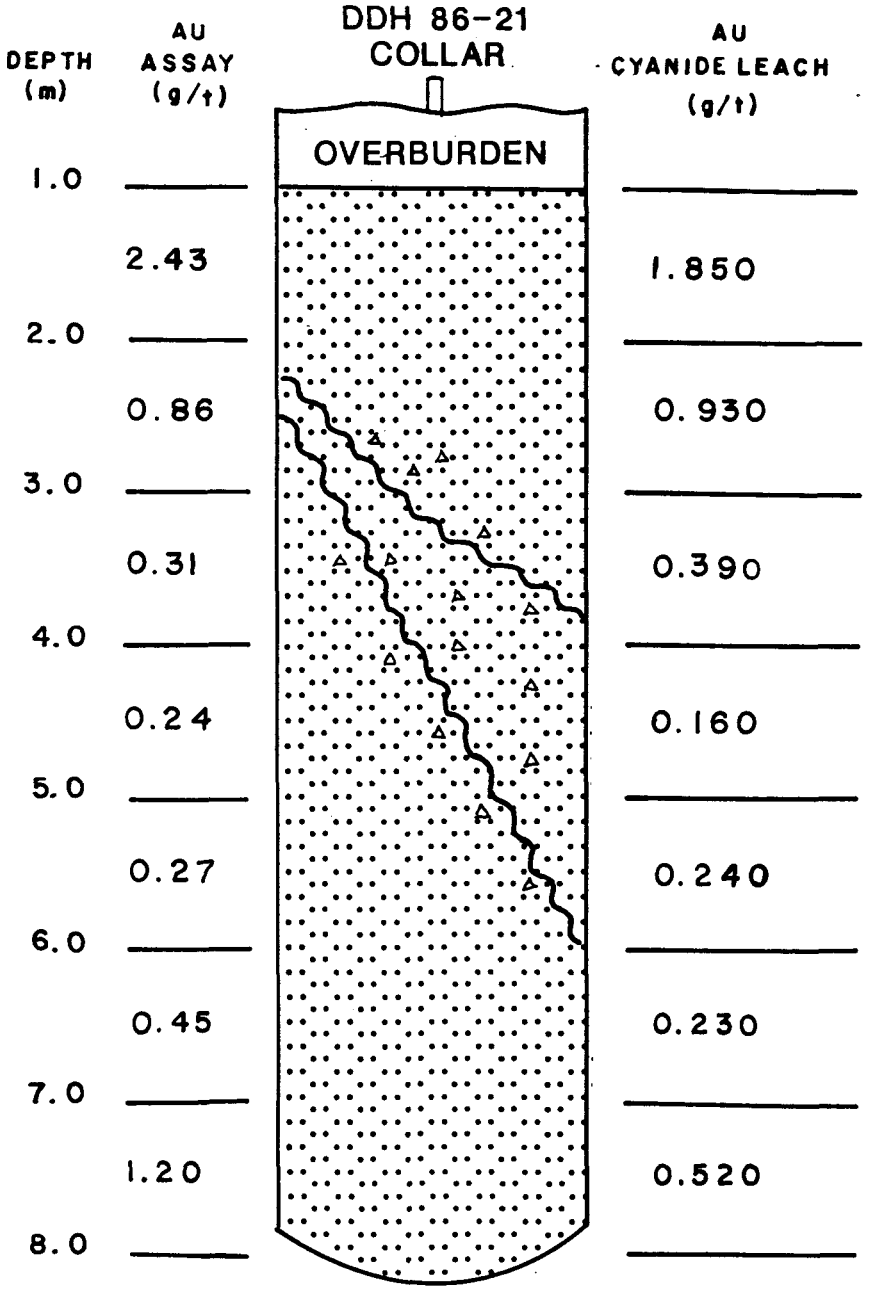
0m 2m

To accompany report dated January, 1988

X - SECTION

FACING

LINE 18N, 0+25E



LEGEND

BRECCIA TEXTURES

SERICITIC QUARTZ-FELDSPAR PORPHYRY

FAULT

FIGURE A-6b

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

METALLURGICAL TEST PIT #2

ANTONIUK DEPOSIT

BIG CREEK RESOURCES LTD.
REXFORD MINERALS LTD.
PERMIAN RESOURCES LTD.

SCALE 1: 50

0m 2m

To accompany report dated January, 1988

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 8.84m	-	14.98
8.84 to 10.36m	-	20.26

These samples 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 2.13 m	- 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 indicates 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 these pits.

MINERAL INVENTORY

All available data was reviewed by E.S. Holt, P.Eng., who calculated the following drill indicated, open pittable reserves. Holt's September, 1986 report is attached as Appendix One which includes vertical sections showing Holt's calculations (Figures A-8a to A-8f).

TABLE 1
DRILL INDICATED, OPEN PITTABLE RESERVES
ANTONIUK DEPOSIT - SEPTEMBER, 1986
E.S. Holt, P.Eng.
(Reserves Modified to Include 1987 Data)

(A) Using a 0.5 g/t (0.015 opt) gold cutoff

Category	Tonnes (Tons)	Grade g/t (opt)	Cumulative Gold - gm (ozs)	Waste Tonnes (Tons)	Waste/Ore Ratio
OXIDE	2,783,900 (3,039,000)	1.040 (.030)	2,895,000 (92,000)	579,000 (632,000)	0.21
SULPHIDE	1,093,600 (1,194,000)	1.500 (.044)	1,640,000 (52,000)	2,405,000 (2,626,000)	2.20
COMBINED	3,877,900 (4,234,000)	1.160 (0.34)	4,498,000 (143,000)	2,985,000 (3,258,000)	0.77

(B) Using a 0.7 g/t (0.020 opt) gold cutoff

OXIDE	2,054,400 (2,243,000)	1.200 (.035)	2,465,000 (79,000)	1,309,000 (1,429,000)	0.64
SULPHIDE	1,068,000 (1,166,000)	1.520 (.044)	1,623,000 (52,000)	2,430,000 (2,653,000)	2.28
COMBINED	3,123,000 (3,409,000)	1.310 (.038)	4,091,000 (130,000)	3,739,000 (4,082,000)	1.20

In his report, Holt classified these reserves as probable (i.e. drill indicated) with a significant portion bordering on the proven category. Additional sampling and more geological data is required to upgrade the probable reserves into a proven (measured) category. This could best be achieved through continued bulldozer trenching and selected diamond drilling.

Holt's calculations were based on assumed specific gravities of 2.4 for oxidized material and 2.6 for unoxidized. To calculate ore/waste ratios, he used a preliminary pit design prepared by Archer, Cathro within the following criteria:

a) pit walls were designed to have 6.9 m wide berms, 7.6 m high benches and a minimum inclined exit of 6° (a geotechnical study will be required to determine if these are optimum parameters);

b) an optimization of pit shape with respect to topography was not attempted although the designated outlines are considered to be good preliminary estimates;

c) two separate pits were outlined (called the "West Pit" and "East Pit") and each was designed for cutoff grades of both 0.5 and 0.7 g/t. Pit slopes were optimized for both configurations; and,

d) Archer, Cathro categorized pit material as oxide, transition, or unoxidized, based on a visual estimate of the oxidized condition of sulphides in drill core. It was assumed that oxide and transition material would leach equally well and they were included together in an "Oxide Pit". Sulphide material, on the other hand, was included in a "Sulphide Pit" and calculated separately.

FURTHER EXPLORATION

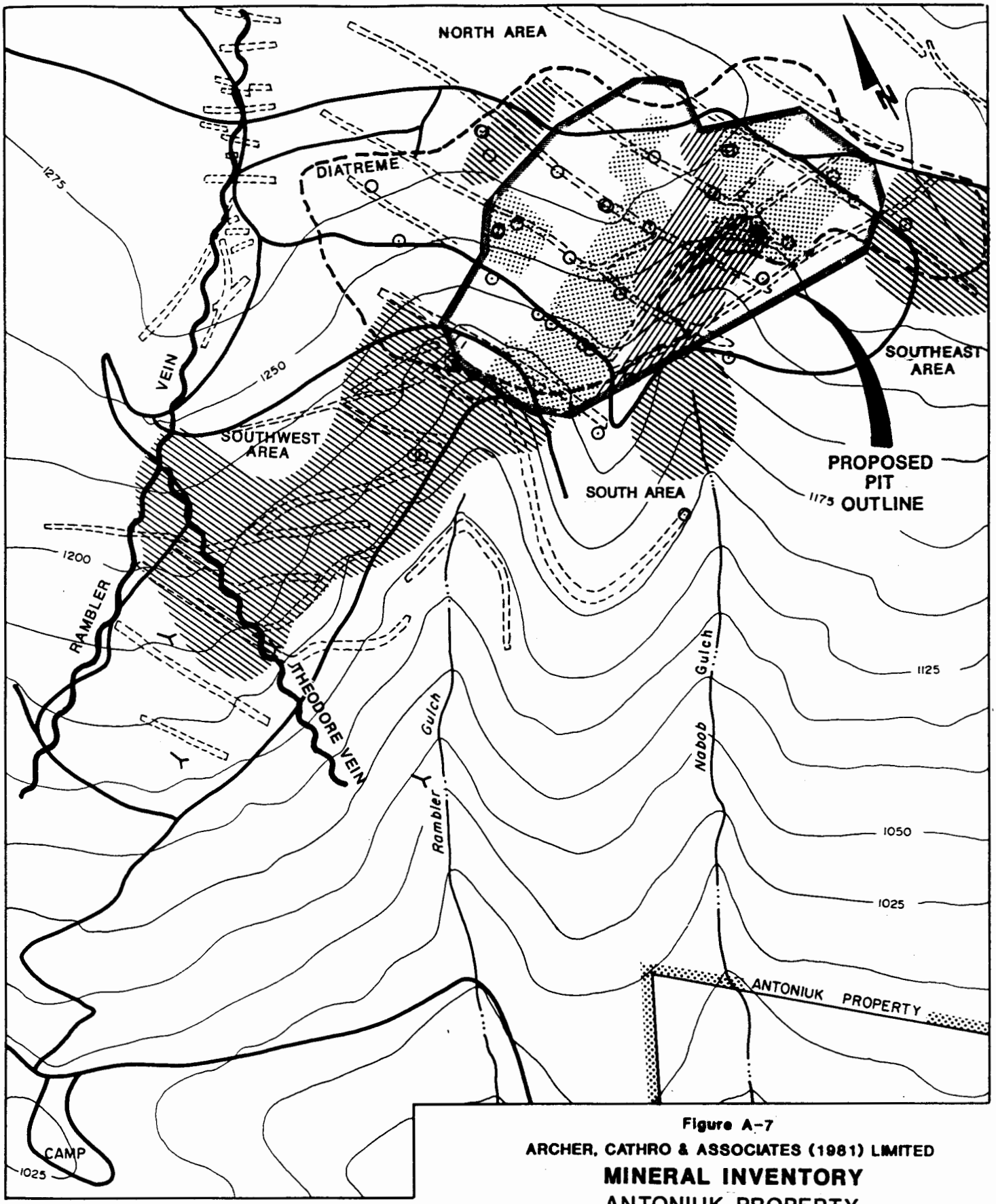
Adjacent to the drill indicated reserve blocks calculated by Holt, there is potential for additional reserves within mineralized zones that have not received sufficient drilling to merit inclusion in the drill indicated category. The portion within the preliminary pit is considered to be an "inferred reserve". The balance outside the pit walls is too poorly sampled at this time to be designated as more than a zone of prime exploration potential. A simplified compilation showing the location of these inferred reserves and potential reserve blocks is presented on Figure A-7 on the following page.

The following programs of exploration are recommended as follows (in order of priority).

1. Proving of Inferred Reserves

a) Diamond Drilling

The highest priority should be assigned to exploration of the inferred reserve block which has good potential because it lies so close to mineralized zones outlined by drilling. This zone, within the provisional pit, has not been drilled and it was therefore designated as waste in Holt's calculations. The limited trench sampling over this block generally carried low values but the terrain is steep and sample quality was poor. A breakdown of the inferred reserves by section, with the amount of drilling needed for exploration, is tabulated below:



DRILL INDICATED RESERVES

DRILL INFERRED RESERVES

EXPLORATION POTENTIAL

DRILL COLLAR

ROAD

ADIT

TRENCH

DIATREME BOUNDARY

Figure A-7

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

MINERAL INVENTORY

ANTONIUK PROPERTY

Mt. Freegold, Y.T.

BIG CREEK RESOURCES LTD.
REXFORD MINERALS LTD.
PERMIAN RESOURCES LTD.

SCALE 1:5000

0 50 100 200 300 m

0 500 1000 ft

TABLE 2
INFERRED RESERVES WITHIN PRELIMINARY PIT
ANTONIUK DEPOSIT

(Altered to delete ore found in 1987)

Section	Tonnes	Diamond Drill Holes (Number/Length)
16N	50,000	1/40 m (130 feet)
18N	110,000	2/160 m (525 feet)
20N	220,000	4/290 m (955 feet)
22N	640,000	4/400 m (1310 feet)
24N	50,000	3/290 m (955 feet)
26N	65,000	3/240 m (800 feet)
TOTAL	1,135,000	

A list of the diamond drill holes required to sample these inferred reserves is given in Appendix Two. This program is budgeted to cost $200,000.

b) Trenching

The success of the 1987 trenching indicated that additional trenching would be useful in locating new ore reserves and defining existing reserves particularly along the south margin of the deposit. The proposed trenching should be oriented northeasterly parallel to the 1987 trenches. About 2000 m of trenching would require 150 bulldozer hours with a Caterpillar D8 bulldozer with ripper, at a budgeted cost of $50,000.

2) Bulk Sampling

The results of 1987 test pitting were inconsistent in showing whether drill core samples predict comparable assays to large bulk samples. The bulk samples from Pit 87-1, located in high grade material, were significantly higher than a corresponding drill hole, while the assays from Pit 87-2, located in low grade material, seemed to have an opposite result. The situation in the

higher grade material affects the overall ore grades much more than the lower grade material. Similar testing should be continued in other areas where there is a discrepancy between trench assays and drill hole results.

Location		Average	Coincident	Average
Section	Coordinate	value (g/t)	Drill Hole	grade (g/t)
18N	2E	1.0	86-22	0.10
20N	2W	1.10	86-03	0.87
22N	2E	0.85	86-20	0.42
26N	5E	1.95	75-23	0.83
	Less commonly, drill holes gave better gold values than the overlying trench samples:			
22N	0E	0.21	86-08	0.95
26N	2E	0.24,0.58	86-13, 75-05	0.93, 1.65

Holt described the matter of grade variations in his report (page 9) and noted that the best quality samples consistently gave the highest values. For example, trench assays are generally higher than the HQ drill core assays, which tend to be higher, in turn, than NQ drill core assays and assays from rotary percussion cuttings. He stated that his calculated reserves are possibly biased on the low side because of this grade discrepancy and that detailed investigation is warranted. Since Holt's reserve calculations include the lower values as well as the higher values, bulk sampling of the lower valued material could improve the calculated grade of some of this near surface material.

The testing of this surface material would require 200 hours of excavator work at a budgeted cost of $50,000, which includes some $10,000 of metallurgical test work on samples obtained.

3) Bulldozer Trenching - Peripheral Exploration

As shown on Figure A-7, good exploration potential exists in three areas peripheral to the preliminary pit. These areas contain scattered trench and drill hole assays exceeding 0.7 g/t but have not been explored systematically and no reserve figures have been calculated. Exploration would consist of 2000 m of trenching in the three most promising areas, along the Theodore Vein and south and southeast of the Antoniuk deposit. This would require 150 hours with a Caterpillar D8 at a budgeted cost of $50,000.

a) Southwest Area (Theodore Vein)

The largest of these areas, which lies downhill from the provisional open pit, contains over 25 sampled intervals with values exceeding 0.7 g/t Au. These range up to 1.92 g/t Au over 13.7 m in a trench over the Theodore Vein and 2.33 g/t Au over 22.56 m in drill hole 75-22. Oxidation is not expected to penetrate as deeply as it does higher on the mountainside.

b) South Area

Trenching in 1987 indicated mineralized northeasterly-trending structures which are unexplored to the southwest. This strike direction extends from the breccia diatreme into more competent granodiorite and the structures may change character to resemble the G-3 Vein to the west.

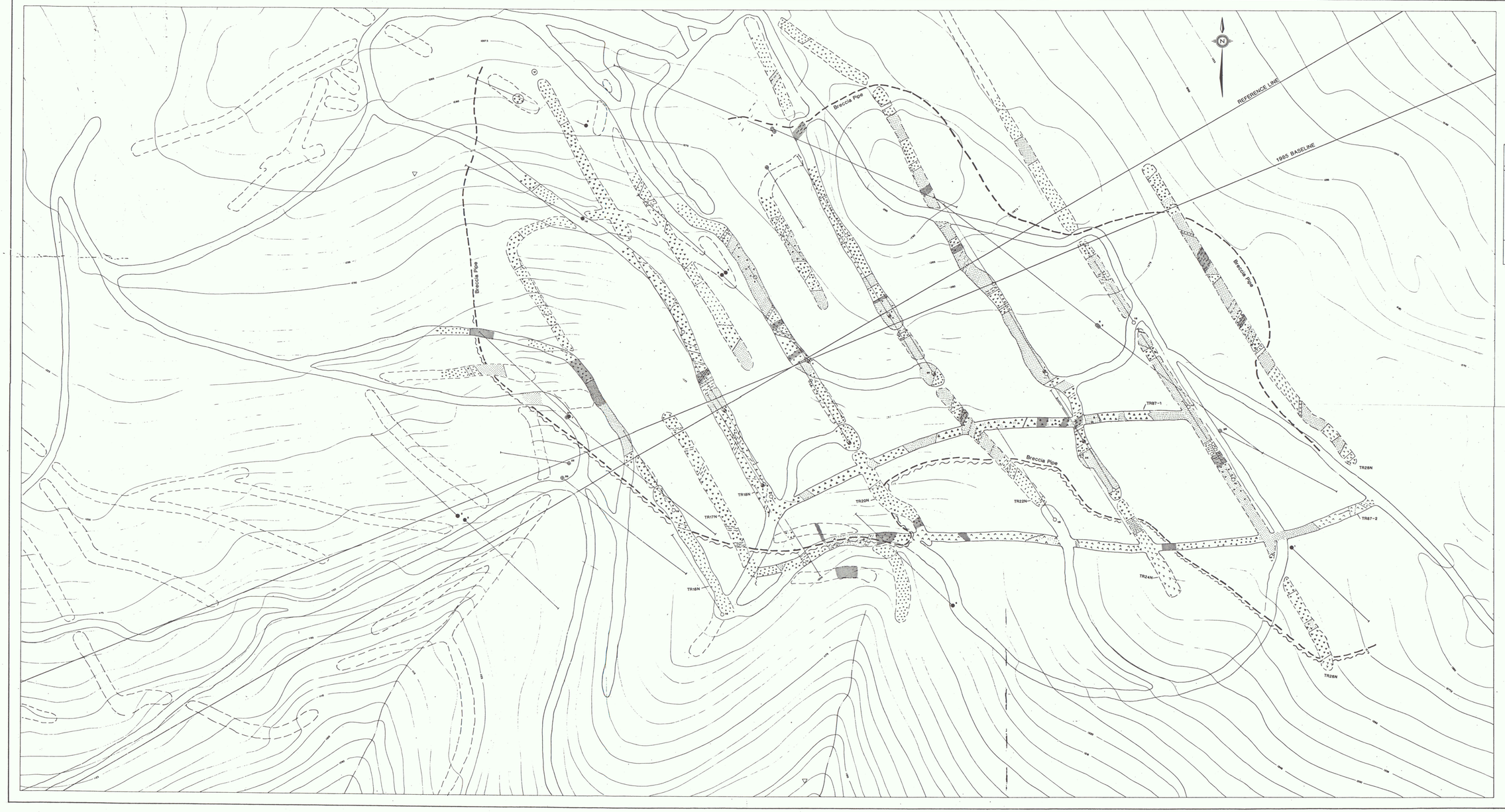
c) Southeast Area

The most easterly drill holes (81-01 and 75-23) intersected a highly favourable host rock consisting of intensely altered heterolithic breccia. Unfortunately, these holes did not produce any significant gold assays although, as mentioned on page 12, they do carry significant silver

assays. Since the best gold values on the property occur in similarly altered rocks along the breccia contact 200 m to the west, it is felt that the remainder of the breccia contact in this area should be tested. Oxidation appears to extend to depths of at least 75 m.

d) North Area

Several drill and trench intersections occurring in this area appear to be related to minor structures that may be associated with the breccia contact. These warrant further investigation because the North Area is situated close to the top of the ridge where oxidation could extend over 60 m deep.



LEGEND

- FAULT
- INTENSE ALTERATION (FAULT GOUGE) ZONE
- DIATREME BOUNDARY
- ROAD OR BULLDOZER TRENCH
- BULLDOZER TRENCH

CRETACEOUS

MT. NANSEN GROUP

- HETEROLITHIC BRECCIA — Intrusive Breccia containing fragments of all rock types noted below
- RHYODACITE — Quartz - feldspar porphyry
- ANDESITE — Feldspar porphyry
- RHYODACITE — Hornblende - quartz - feldspar porphyry

CASINO GRANODIORITE

- HORNBLENDE-BIOTITE GRANODIORITE

JURASSIC

BIG CREEK SYENITE

- HORNBLENDE SYENITE

805

092161

DRILL HOLES

- 1975
- 1981
- 1985 (percussion)
- 1986

Figure A-4

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

GEOLOGY

ANTONIUK PROPERTY

BIG CREEK RESOURCES LTD.
REXFORD MINERALS LTD.
PERMAN RESOURCES LTD.

SCALE - 1:4000

0 5 10 15 20 25 50 m

to accompany report dated February, 1987

Figure A-5

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

TRENCH ASSAYS

ANTONIUK PROPERTY

BIG CREEK RESOURCES LTD.
REXFORD MINERALS LTD.
PERMIAN RESOURCES LTD.

SCALE - 1:1000

To accompany report dated January, 1988

APPENDICES

APPENDIX ONE

RESERVE ESTIMATE BY E.S. HOLT, P.Eng.

September 25, 1986

RESERVE ESTIMATE

ANTONIUK GOLD DEPOSIT

MOUNT FREEGOLD, YUKON TERRITORY

September 25, 1986

Prepared by

HOLT ENGINEERING LTD.

TABLE OF CONTENTS

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

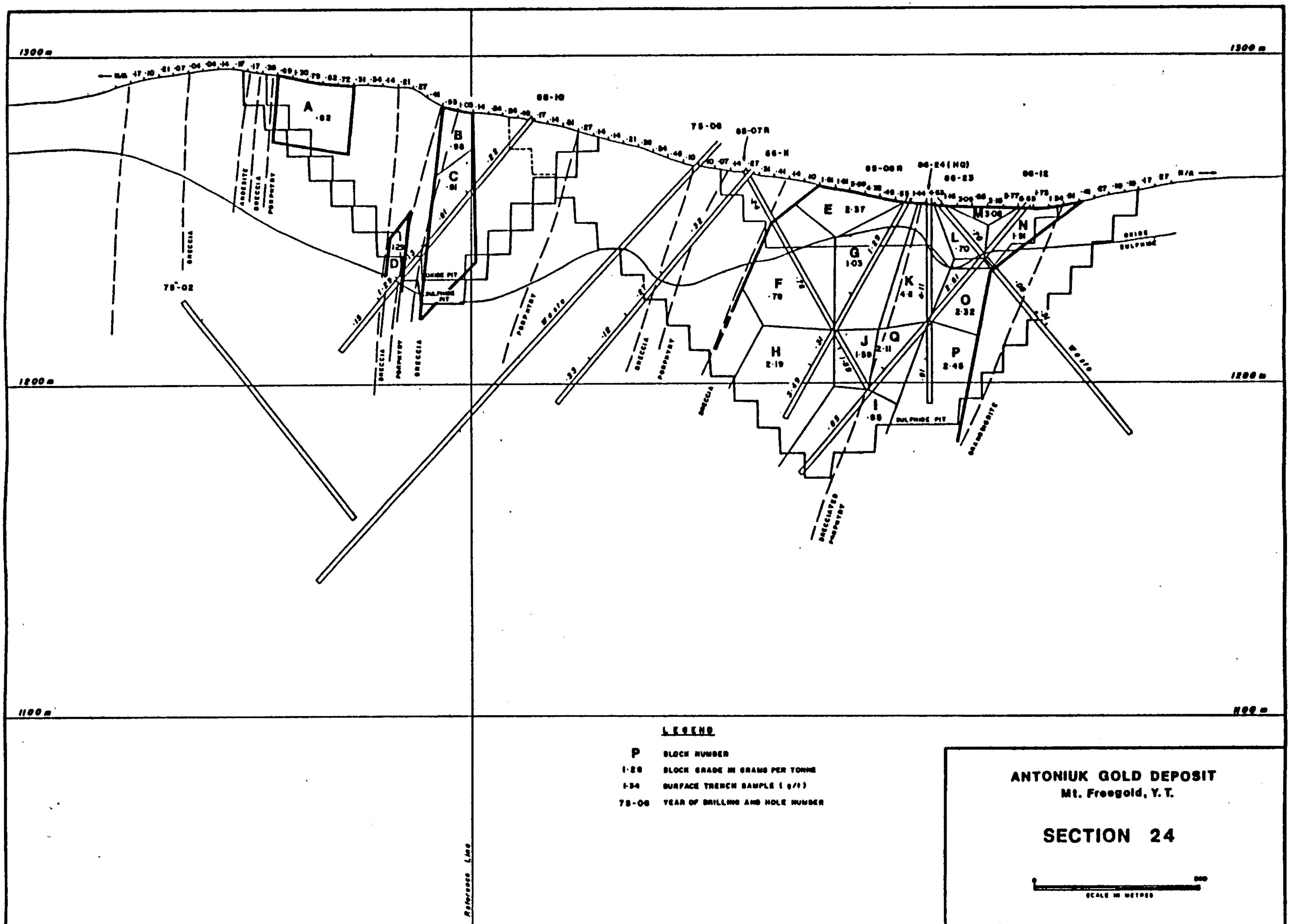
500 x 300m anomaly

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


1300 m
1200 m
1100 m
A
B
C
D
E 2·37
F ·78
G 1·03
H 2·19
I
J 1·59
K 4·8
L ·70
M 3·06
N 1·91
O 2·32
P 2·45
Q 2·11
75-02
75-06
88-10
OXIDE
SULPHIDE
OXIDE PIT
SULPHIDE PIT
BRECCIA
PORPHYRY
ANDESITE
GRANODIORITE
BRECCIATED PORPHYRY
Waste
Reference Line
LEGEND
P BLOCK NUMBER
1·28 BLOCK GRADE IN GRAMS PER TONNE
1·34 SURFACE TRENCH SAMPLE (g/t)
75-06 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	

APPENDIX TWO

RECOMMENDED EXPLORATION DIAMOND DRILL HOLES

ANTONIUK DEPOSIT

APPENDIX TWO
RECOMMENDED EXPLORATION DIAMOND DRILL HOLES - ANTONIUK DEPOSIT

DDH	SECTION	METRES FROM REFERENCE LINE	AZIMUTH (°)	DIP (°)	LENGTH (m)	LENGTH (FEET)
1)	16N	40E	150	-50	40	130
2)	18N	30E	150	-50	108	360
3)	18N	100E	150	-50	50	165
4)	20N	30W	330	-50	76	250
5)	20N	40E	330	-50	40	130
6)	20N	45E	150	-50	125	410
7)	20N	110E	150	-50	50	165
8)	22N	35W	150	-50	108	360
9)	22N	115E	150	-50	108	360
10)	22N	60E	150	-50	108	360
11)	22N	115E	150	-50	76	250
12)	24N	60W	150	-50	107	350
13)	24N	50E	150	-60	125	410
14)	24N	200E	240	-50	61	200
15)	26N	50E	240	-50	61	200
16)	26N	150E	150	-50	91	300
17)	26N	250E	240	-50	91	300
					1,433	4,700

APPENDIX THREE

STATEMENT OF QUALIFICATIONS

STATEMENT OF QUALIFICATIONS

I, Charles A. Main, geologist, with business addresses in Whitehorse, Yukon Territory and Vancouver, British Columbia and residential address in Vancouver, British Columbia, hereby certify that:

1. I graduated from the University of British Columbia in 1971 with a B.Sc. majoring in Geological Sciences and Chemistry.
2. I have been actively engaged as a geologist in mineral exploration since 1971 and as a partner of Archer, Cathro & Associates (1981) Limited since June 1, 1981.
3. I have personally participated in or supervised the field work reported herein.

Charles A. Main

Charles A. Main, B.Sc.

APPENDIX FOUR

LIST OF EMPLOYEES

NAME	POSITION
C.A. Main	Geologist
J.L. Duke	Geologist
L. Robinson	Fieldman
G. MacIntosh	Fieldman
F. Andersen	Fieldman