EUREKA RESOURCES, INC.

FRASERGOLD PROPERTY

CARIBOO MINING DIVISION, B.C.

RESULTS OF 1987 EXPLORATION PROGRAM

NOVEMBER 1987

CAMPBELL & ASSOCIATES
GEOLOGICAL CONSULTANTS

VOLUME 1

REPORT ON THE GEOLOGY AND RESULTS
OF THE 1987 EXPLORATION
ON THE FRASERGOLD PROPERTY

MacKay River Area
Cariboo Mining Division, British Columbia
N.T.S. Map Area 93A/7E
Latitude 52° 19'N   Longitude 120° 37'W

for

EUREKA RESOURCES, INC.
837 East Cordova Street
Vancouver, B.C.
V6A 3R2

by

K.V. Campbell, Ph.D.
B.E. MacKean, M.Sc.
D.A. Leishman, B.Sc.

November, 1987

VOLUME 1

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K.V. Campbell
November 30, 1987

SUMMARY

The 1987 exploration program of Eureka Resources, Inc. on their Frasergold property in the Cariboo Mining Division was successful in futher defining mineralization and potential ore zones.

This years work was the fifth consecutive season of exploration on the claims and focused along a 1 km length of mineralization. It consisted of reverse circulation drilling (21 holes totalling 1710 m), 660 m of trenching and preparation of an exploration adit portal site on the Jay Zone. Studies were also done on sludge assays vs drill cuttings assays and on total metallic assays vs conventional fire assays.

Additional structural studies elucidated the geological model of mineralization. Quartz veins, which host the gold, occur in the basal section of a lustrous, porphyroblastic phyllite. These rocks, which lie on the upright, southwesterly dipping limb of the Eureka Syncline, have been locally deformed into asymmetric drag folds. Quartz, which arose through metamorphic processes, migrated into the hinges of these folds. Subsequent rotation of the folds by an axial plane crenulation cleavage produced minor folds which plunge slightly northwest of the earlier drag folds. It is in quartz-filled fold hinges of the youngest structures that gold is thought to be concentrated.

The mineralization in the central work area has a drill indicated average total width (down to about 50 m vertical depth) of 17.6 m with an average grade of 0.071 oz/ton Au. This zone has been drilled at 25 and 50 m intervals over a strike length of 750 m. Contained within the zone is an enriched horizon, at least 225 m long, averaging 11.3 m width and 0.155 oz/ton Au. Analysis of approximately 140 samples by total metallic assay methods suggest a possible upgrading factor averaging 15%.

Similar mineralization continues at least another 750 m to the southeast, for a total mineralized strike length of 1½ km. There is a strong inferred geological potential for this area to yield in neighbourhood of 20 million tons with an average grade between 0.05 and 0.08 oz Au/ton. As determined in earlier programs, there is a very good possibility that mineralization extends more than 10 km on the Frasergold property. Four holes sited northwest of the Jay Zone in 1987 (between that zone and the northwest extension drilled in 1986) intersected narrow widths of low and marginal grade.

A two phase program of continued drilling (diamond and reverse circulation) and underground exploration is recommended, at a total cost estimated to be about $2,750,000.

Volume 1

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1 INTRODUCTION

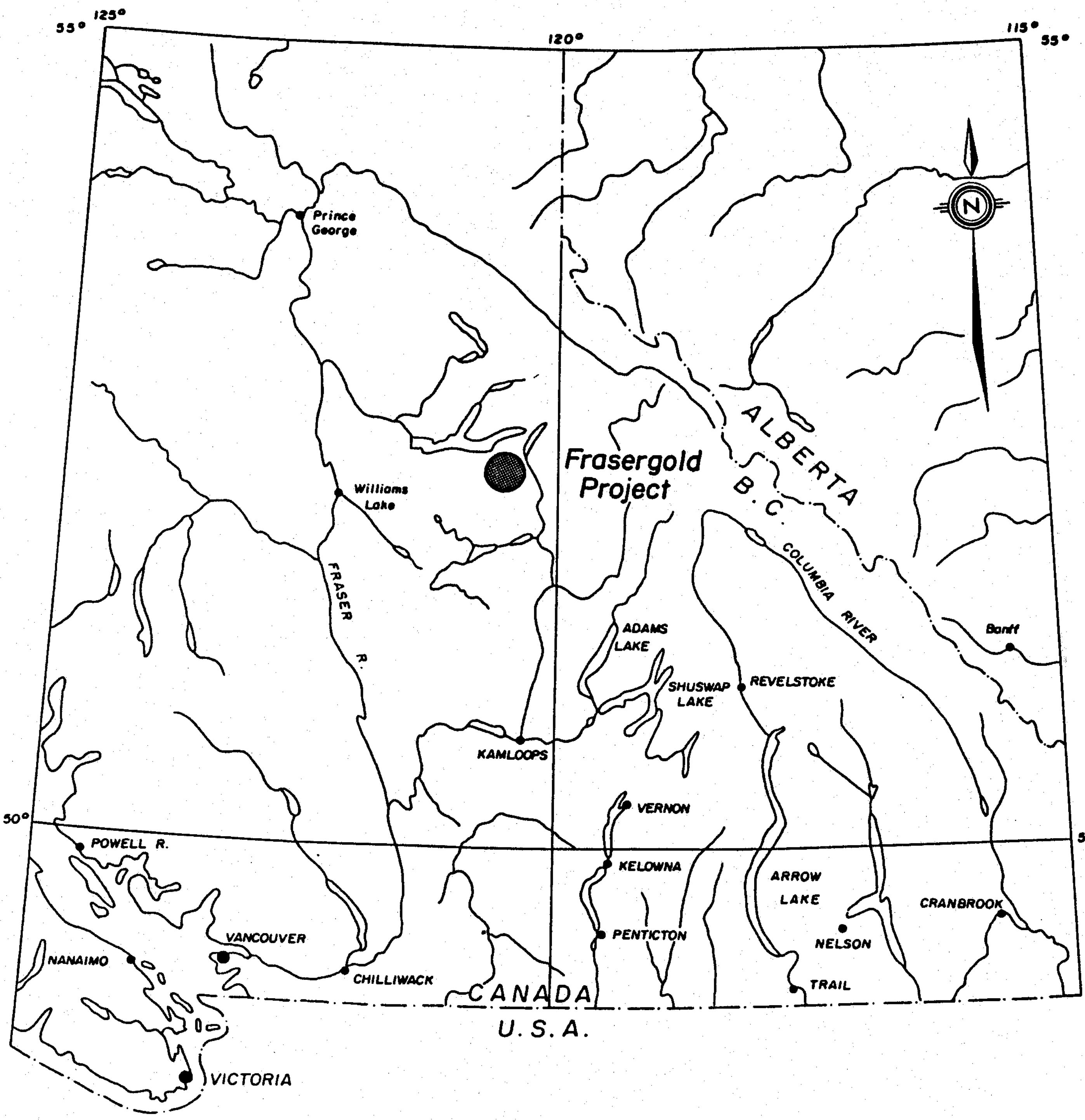
1.1 Location, Access and Terrain

The Frasergold Property lies in the central Cariboo region of British Columbia, approximately 100 kilometers east of Williams Lake in the Cariboo Mining Division. The claims straddle the MacKay River Valley and are centered approximately at 52° 19'N and 120° 37'W within National Topographic System area 93A/7E, Figures 1 and 2.

Road access to the property is east for 55 kilometers on the paved Highway 97 from 150 Mile House to Horsefly, then northeasterly along an all-weather logging road following the Horsefly River for 55 km, past the Crooked Lake road junction near Post 145, to a branch road to the southeast which enters the MacKay River valley. At the Crooked Lake junction, the diamond drill core from the property is stored at a logging camp there.

The MacKay River road bears east upon crossing Carlson bridge over the Horsefly River, then extends 7 km to the Hawkley Creek and MacKay River junction near the northwest corner of the northern group of the Eureka claims. The logging road which branches southwest across the MacKay River continues southeasterly within the central portion of the property for 7 km to the start of the 4x4 mining road which continues in the same general direction another 3 km over steep terrain.

The Frasergold Property occurs on the west flank of the Cariboo Mountain Range. Topography is moderately steep in the northwest portion of the claim group and steeper in the southeast portion. Relief on parts of the property exceeds 1,000 meters. Most recent exploration work has been on the



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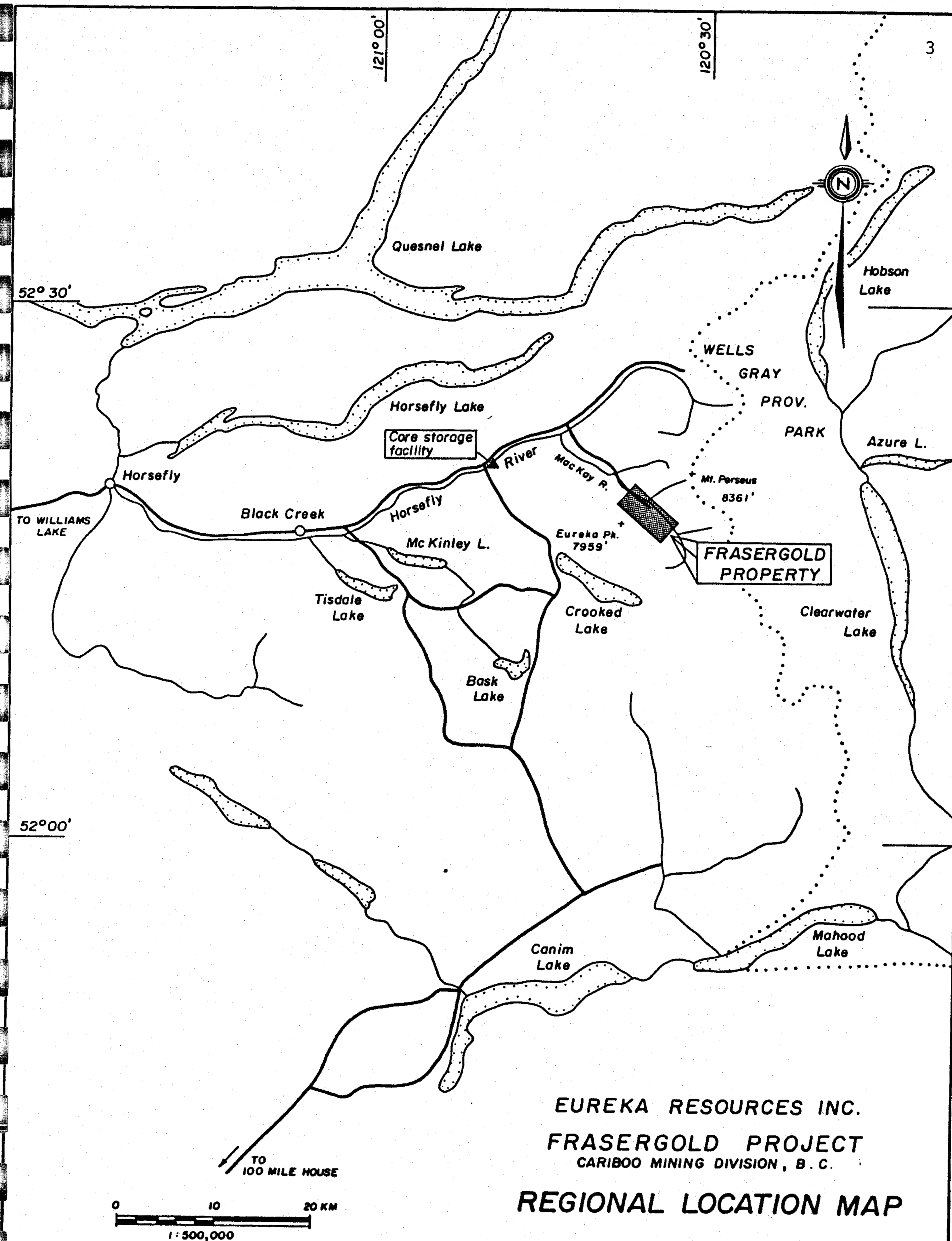
LOCATION MAP

FRASERGOLD PROJECT

CARIBOO MINING DIVISION, B.C.

Technical Work By:	Scale: 1:2,500,000 (1cm=25km)	
Date: Nov., 1985	Drawn By: W.G.	Fig. No. 1



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FRASERGOLD PROJECT

CARIBOO MINING DIVISION, B.C.

REGIONAL LOCATION MAP

FIGURE 2

northeasterly facing slope of the MacKay River valley between elevations of 1,200 and 1,550 meters (Figures 2 and 3).

The vegetation along the MacKay River valley consists of good stands of commercial spruce and balsam with thick underbrush. Forest cover is lighter above 1,600 meters and alpine vegetation is encounterd at approximately 1,800 meters elevation. Large areas of the claim group have undergone logging which has left a good network of access trails.

1.2 Claims Status

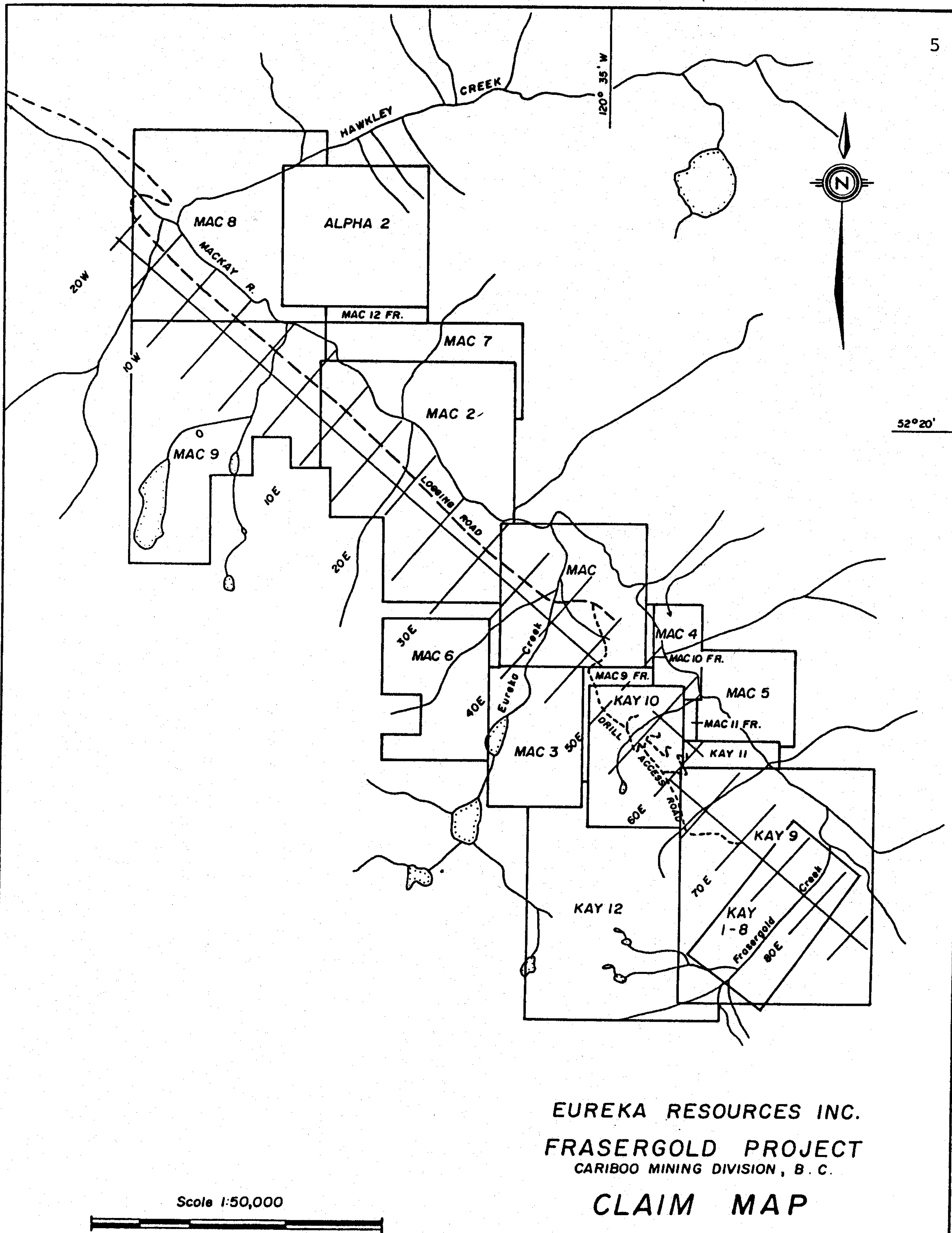
The Frasergold Property consists of 27 mineral claims (183 units) all located and recorded in accordance with the mining laws of the Province of British Columbia.

Eight of the original claims are two-post claims, four are fractional claims while the remaining 15 are modified grid claims. All claims are in good standing until 1992 - 1996. Table 1 lists particulars of the claims.

In 1984 the legal corner posts of all the claims were surveyed to legal survey standards by McElhaney Associates Ltd. of Vancouver.

All claims are recorded in the name of Eureka Resources, Inc. The original claims staked by the vendor, Clifford E. Gunn, have been transferred to Eureka and upon termination of the agreement with Amoco in 1985 all claims located by Amoco were transferred to Eureka.

In July, 1986 a 20 unit claim, Mac 10, was staked along strike of the geochemical anomaly to cover its projected extension to the northwest. Work performed on this claim is



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CLAIM MAP

FIGURE 3

Table 1. Claim Data

Claim Name	Units	Record No.	Expiry Date
Mac	9	1286	Oct. 19, 1993
Mac 2	20	2078	Oct. 22, 1992
Mac 7	8	6249	July 27, 1992
Mac 8	16	6250	July 27, 1992
Mac 9	20	6251	July 27, 1992
Mac 9Fr.	1	6204	July 16, 1993
Mac 12Fr.	1	6253	July 27, 1992
Kay 10	6	1961	Sept. 25, 1993
Alpha 2	9	5159	Sept. 23, 1992
Kay 1-8	8	1182-89	Sept. 04, 1992
Kay 9	20	1810	Aug. 11, 1993
Kay 11	2	1962	Sept. 25, 1996
Kay 12	20	4631	Jan. 26, 1992
Mac 3	6	3074	Dec. 23, 1993
Mac 4	2	3075	Dec. 23, 1993
Mac 5	4	6248	July 27, 1993
Mac 6	9	3077	Dec. 23, 1993
Mac 10Fr.	1	6231	July 19, 1993
Mac 11Fr.	1	6252	July 27, 1994
Mac 10	20	7838	July 31, 1989

described in a separate assessment report by D. Leishman (1987, in preparation).

1.3 History

The original claims of the Frasergold property were staked by Clifford E. Gunn to cover an occurrence of placer gold in Frasergold Creek. In the late 1970's Mr. Gunn was attracted to the MacKay River valley on the basis of references in B.C. Ministry of Mines reports from the turn of the century which referred to the testing of the valley's placer potential.

Prior to the 1970's the only documented reports of exploration in the area were on the Eureka Peak property (immediately to the southwest of the Frasergold property) which was explored for porphyry copper deposits by both Amax and Rio Tinto. The mineralization there is within a different geological setting than that encountered on the Frasergold property, although some gold values have been reported.

Subsequent work to that of Mr. Gunn by a private company (the predecessor of Eureka Resources, Inc.) revealed the existence of a large soil geochemistry anomaly with apparent stratigraphic control. This feature, together with the history of gold mining in the Cariboo, led to the formation of Eureka Resources, Inc. whose objective was to systematically explore and develop the potential of the Frasergold property.

Below is a summary of documented work on the property, subsequent to the acquisition of the original claims by Mr. Gunn.

1978-1979: Prospecting and staking of the original ground (Alpha, Mac and Kay 1-6 mineral claims) by Mr. Gunn.

1980-1982: The ground was optioned by Keron Holdings Ltd. and NCL Resources Ltd. who expanded the claims to include the Kay 9-12 and Mac 2-9 claims. A preliminary geochemical survey was made over the entire claim block with a total of 3,000 soils and 150 rock chip samples collected. Soil profiles were also taken to study the nature of gold in soil (250 samples). At the same time the property was geologically mapped on a scale of 1:10,000.

1983: Eureka Resources, Inc. acquired the property in 1983 and optioned it to Amoco Canada Petroleum Co. Ltd.

Amoco completed seven kilometers of drill access road and 1.2 km of hand trenches, collecting 1,070 samples. An additional 820 soil samples were collected over the anomalous section of the original survey. Limited electromagnetic and magnetic surveys were also completed. A five hole diamond drill program totalling 1,644 m was completed over a 0.8 km portion of the geochemical survey.

A total of 20 intervals of anomalous gold intersections were encountered with a range of values from 0.028 oz/t Au over 3.0 m to 0.180 oz/t Au over 4.5 m. Coarse visible gold was noted in the first three drill holes.

1984: Amoco continued their exploration, performing geochemistry, geophysics and diamond drilling. Geochemical sampling included 1,959 soil samples and 190 rock chip samples. Radem-Electromagnetic and magnetometer surveys were performed over the main part of the gold anomaly. A legal survey was made of the claim posts.

Nine holes (NQ size), totalling 2,875 m, were drilled along the trace of the soil geochemical anomaly. These holes confirmed the existence of subeconomic to economic grade mineralization for 1.5 km along the strike of the soil anomaly.

As in previous drilling, gold values were encountered in every hole with values ranging from 0.098 oz/t up to 0.342 oz/t Au over 1.5 m and 0.144 oz/t Au over 4.5 m. In addition, numerous intersections were made where values ranged from 0.023 oz/t Au over 7.5 m to the values quoted above. Visible gold was noted in all nine drill holes.

1985: After termination of the option agreement with Amoco, Eureka Resources, Inc. continued with further exploration.

A total of 1,020 soil samples were collected over the northwest part of the claims. This confirmed the continuity of the anomalous soil geochemistry for approximately six kilometers northwest of the area drilled.

A test I.P. survey of six line km was completed on very widely spaced lines over the mineralized horizon and its projected extension. A sharp change in resistivity was noted along the contact of the knotted phyllites and the underlying black banded phyllites. It was assumed that this resistivity difference was due to the variation in graphite content. This contact also forms the footwall of the gold-enriched horizons on the property.

Bulk sampling was undertaken and one of the samples was subjected to milling and cyanidation by Coastech Research Inc. This particular sample was split with a total of 56 individual assays obtained from three different laboratories. The mean values obtained varied from 0.06 to 0.128 oz/t Au. Upon being subjected to a milling and cyanidation process a value of 0.137 oz/t recoverable Au was obtained. This implies, therefore, that conventional fire assay techniques are probably not adequate for determining true gold content in samples taken from the property.

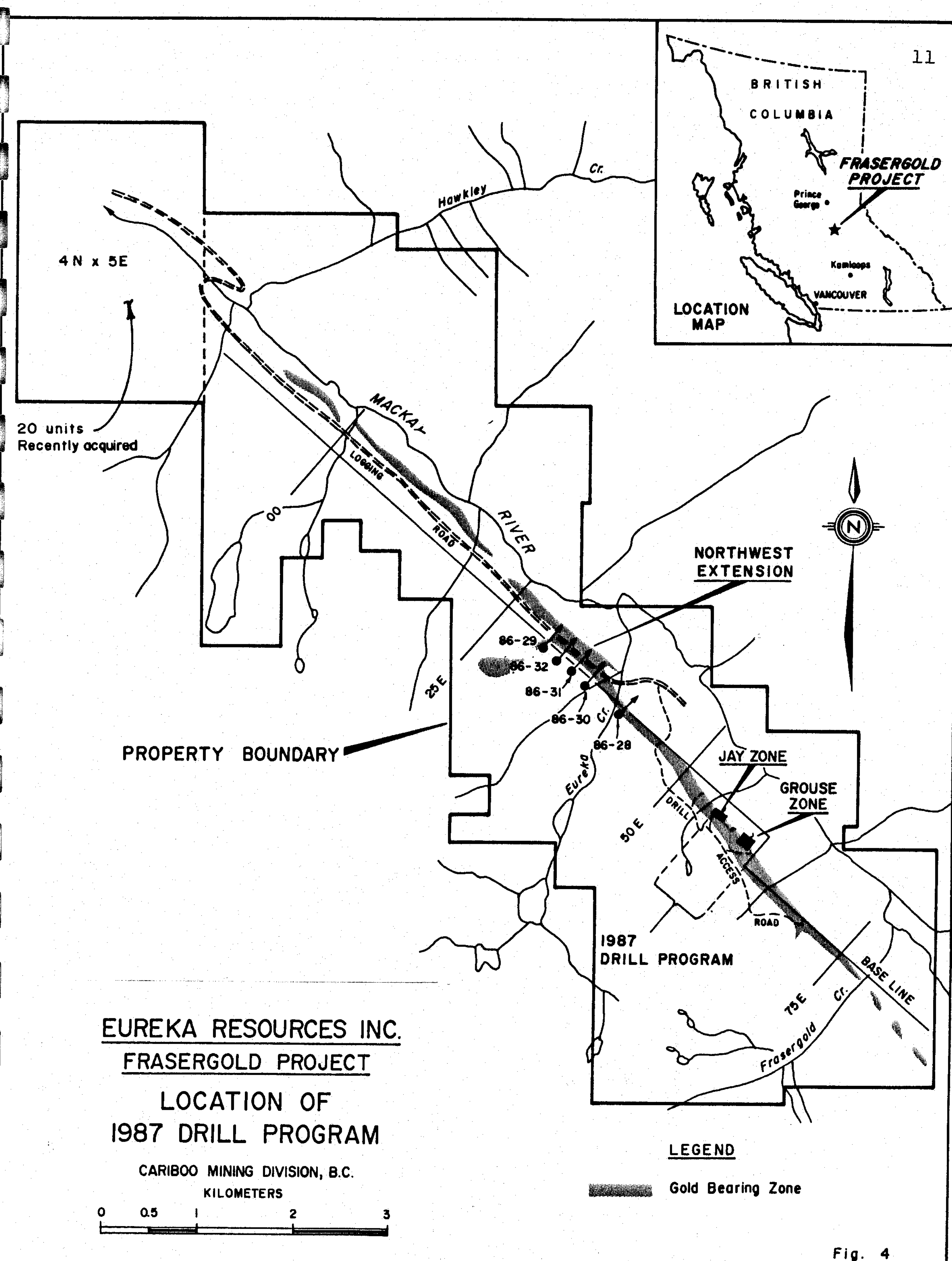
1986: Trenching, bulk sampling, reverse circulation drilling and diamond drilling were done by Eureka Resources in the 1986 field program. This work was concentrated in two areas, the Jay and Grouse zones, although some drilling was done at the northwest extension of the geochemically anomalous zone. These areas are shown in Figure 4.

The trenching program included the collection of 230 rock chip samples in 14 trenches. Bulk samples, 150 to 500 kg in size, were then taken from eight sites. In addition, three samples were subjected to metallurgical tests.

Four reverse circulation holes, with a diameter of 4½", were attempted but the overall performance was unsatisfactory with only one hole reaching the desired depth (148.5 m). However, the limited results obtained do indicate that large volume drilling upgrades the assay grades, as would be expected from the coarse particulate nature of the gold.

Eighteen HQ holes, totalling 2,021 m, were diamond drilled. Findings are summarized as follows:

Grouse zone - 2 holes; 10 to 20 m wide zone of sub-economic values (about 0.020 oz/ton) down-dip of trenches reporting



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LOCATION OF
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Fig. 4

0.037 oz/ton over 20 meters.

Jay zone - 11 holes; 10 to 30 m wide zone of significant values (0.05 to 0.10 oz/ton).

Northwest extension - 5 holes; The gold-enriched knotted phyllites do occur beneath the geochemical anomaly here. Values reported in these holes are less than those reported to the southeast.

The results of the 1986 program (Leishman and Campbell, 1986) indicated the following:

1) Total metallic assays nearly double those of original conventional assays.
2) Large diameter reverse circulation drilling returns higher values than with NQ diamond drilling over the same mineralized interval.
3) Bulk sampling demonstrates that a wide variation in assay results can be expected from a single sample.
4) Assay results from HQ core are more impressive than those of NQ core.
5) Gold-enriched horizons do extend along the projected strike of the knotted phyllite unit to the northwest for at least 4 kilometers.

1.4 Summary of 1987 Exploration Program

The 1987 work program consited of bulldozing 2 kilometers of new drill access roads, 660 meter of trenching and preparation of 16 drill sites. The proposed adit portal site and work/storage area was also prepared along with access to the adit site.

Reverse circulation drilling commenced on July 5 and was halted August 14, 1987 after drilling 21 holes totalling 1,710 meters. Two holes were not completed. Drilling was along a strike distance of about 1,000 m bounded by Sections 50+00E and 60+00E and centered on the Jay zone. All holes were inclined 50° to 60° from the horizontal in the direction of 045° azimuth. Diamond drill holes of previous programs provided geological control. Sludge samples were collected during the drilling of six reverse circulation holes for assay comparison. Drill cuttings were sampled at 1½ m intervals over the entire length of each hole. Total metallic assays were made on 139 selected samples.

Ninety-five trench samples were channel chipped over 1.0 to 1½ or 2 meter intervals along eight trenches (T8701 to T8708). Chip samples of quartz veins only (C87-) were sampled while geological mapping.

2 GEOLOGY

2.1 Regional

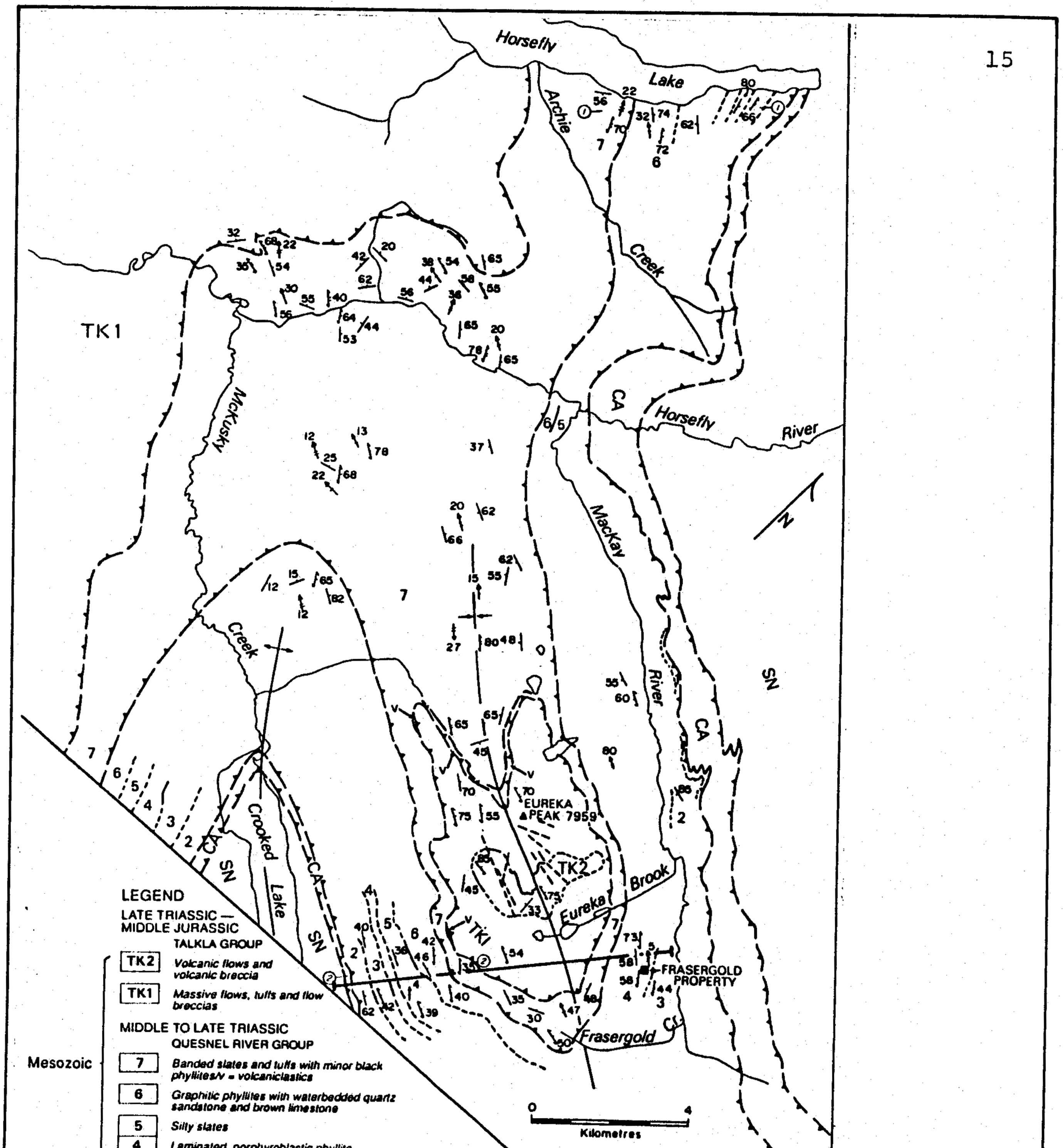
Figure 5 illustrates the regional geology of the MacKay River area (Bloodgood, 1987). The property lies across the boundary between two major tectonic belts of the Canadian Cordillera; the Omineca Tectonic Belt on the east and the Quesnel Trough of the Intermontane Belt on the west. Three regional tectonostratigraphic sequences are shown in Figure 5.

1) On the eastern side of the area shown are Haydrynian to early Paleozoic quartz-mica schists and gneisses of the Snowshoe Group.

2) Along the boundary between the two tectonic belts lies the Eureka thrust (Struik, 1986). Pennsylvanian and Permian amphibolite, chlorite schist and chlorite-epidote schist, which make up the Crooked Amphibolite, overlie the Snowshoe Group metasedimentary rocks above the thrust. These rocks are probably equivalents of the oceanic crustal rocks of the Antler Formation (Slide Mountain Group) in the Wells area.

3) Above the Paleozoic metavolcanic rocks lies the (predominantly) sedimentary, Middle to Late Triassic Quesnel River Group and the volcanic, Late Triassic to Middle Triassic Takla Group.

Bloodgood (1987) has subdivided the Quesnel River Group in the Eureka Peak area into 7 units as shown in Figure 5. Black phyllites predominate. Basic volcanic rocks of the Takla Group occupy the core of the Eureka (Peak) syncline.



LEGEND

LATE TRIASSIC — MIDDLE JURASSIC

TALKLA GROUP

Mesozoic	TK2	*Volcanic flows and volcanic breccia*
	TK1	*Massive flows, tuffs and flow breccias*
		MIDDLE TO LATE TRIASSIC QUESNEL RIVER GROUP
	7	*Banded slates and tuffs with minor black phyllites/v = volcaniclastics*
	6	*Graphitic phyllites with waterbedded quartz sandstone and brown limestone*
	5	*Silty slates*
	4	*Laminated, porphyroblastic phyllite*
	3	*Phyllitic siltstone*
	2	*Micaceous black phyllite and minor tuffs*
	1	*Micaceous quartzite*
		MISSISSIPPIAN TO EARLY PERMIAN
Paleozoic	CA	*Crooked Amphibolite: amphibole – chlorite schist, chlorite – epidote schist*
		HADRYNIAN AND YOUNGER
Proterozoic Early Paleozoic	SN	*SNOWSHOE GROUP: quartz-mica schist*

(from Bloodgood, 1987)

EUREKA RESOURCES, INC.			
FRASERGOLD PROJECT			
REGIONAL GEOLOGY			
PROJECT 84-72C	DRAWN	DATE Nov. 1987	FIGURE 5
Revised		N.T.S. 93 A	
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Metabasalt, augite porphyry flows, tuffs and volcanic breccias have been metamorphosed to a low grade. The contact with the underlying black phyllites is a fault (Bloodgood, 1987).

The dominant structures in the region are the northwest trending Eureka Syncline and Perseus Anticline (Campbell, 1971). The intervening limb of these structures is overturned to the southwest and contains the contact between the Quesnel Trough and Omineca Tectonic Belt; i.e. the contact between the Crooked Amphibolite and rocks of the Upper Triassic unit. These large folds display a change in attitude along their trend. Southeast of the project area the folds are overturned to the southwest (axial planes dip steeply northeast) whereas to the northwest the folds are upright.

Regional dynamothermal metamorphism affected all the pre-Tertiary rocks in the area. The lowest grades are seen along the Horsefly River road where clastic textures are well preserved. In the Eureka Syncline the metamorphic grade of all units increases towards the Perseus and Boss Mountain Anticlines (the latter is south of the area shown in Figure 5). Large areas reached medium grades of metamorphism (amphibolite facies) and some rocks in the core of the anticlines reached the kyanite-staurolite-fibrolite zone. The metamorphism is considered to be Jurassic to early Cretaceous.

The MacKay River valley marks a major zone of vertical or near vertical fracturing. The Upper Triassic black phyllite unit is sandwiched here between two more competent units; younger intrusives and volcaniclastics above and to the southwest and older amphibolites to the northeast. In order to accommodate the transition of fold form; i.e. the change from upright to overturned limb, structural adjustments, shearing etc., have been concentrated in the incompetent phyllitic unit.

2.2 Property

2.2.1 Lithology

The property is underlain by a thick sequence of dark grey to black, lustrous phyllites with minor intercalations of limestone, calcareous siltite, light grey siltite and greenish grey carbonate-quartz-sericite schist. Within the phyllite sequence is a 200-300 meter wide zone of porphyroblastic phyllite, locally referred to as the "knotted phyllite". Figures 6a, b and c show the bedrock geology of the property. A brief description of lithologies follows:

1) Black banded, graphitic phyllites

Foliation and original bedding planes are very distinct within this rock unit. Black, graphitic smears are common along the foliation planes. Original grain size of the sediments are of mud-silt grain size, however thin sand-sized horizons are common.

2) Dark grey knotted phyllite

The characteristic knots of this phyllite are porphyroblasts, a product of regional metamorphism, which have been identified as an iron-rich carbonate (siderite/ankerite). Surface weathering gives this rock unit a distinctive brown mottling texture. The knots are elongated within the foliation planes and vary in diameter from 2 mm to 2 cm. The original bedding features are not as discernible as in the black banded phyllite. The unit is approximately 200 meters thick and is located in the central portion of the sedimentary unit.

3) Calcareous banded phyllite

In hand specimen, this unit is not easily distinguished from the black banded phyllite. However, it is generally brownish weathering, of lighter grey colour and reacts to acid. The calcareous sediments occur as irregular horizons over thicknesses of 30-50 meters.

4) Light grey siliceous metasediment

These rocks are distinguished from other units by their light grey colour and coarse sandy texture. Petrographic examination shows the rocks to have originally been a quartz-rich sandstone or quartzite, and has eliminated the possibility of a volcanogenic origin. The original bedding features are distinct but the foliation is not as well developed as in the phyllites. The main horizons occur as erratic lenses with thicknesses ranging from 1 to 25 meters.

5) Light green carbonate-quartz-sericite-chlorite schist

The colour and coarse granular aspect of this rock caused it to be identified in the field as a volcanic tuff. This is not substantiated by the petrography. The white, medium to coarse grained clasts are carbonate, probably dolomite, which occur as porphyroblasts in some places. The matrix of the rock is a mixture of varying amounts of sericite and clinochlore. Fine grained quartz occurs as inclusions in the carbonate grains and in fine laminations. The overall texture of the rock is finely laminated to streaked. This texture is unlike that of a volcaniclastic rock and one author (Campbell) considers these rocks to be the product of greenschist facies metamorphism of impure calcareous sediments with excess SiO_2 and K_2O.

2.2.2 Structure

The phyllites of the Frasergold Property lie on the northeastern limb of the northwest trending Eureka Syncline. Bedding (S_0) can be clearly identified in several outcrops of phyllite and is defined by laminations to thin layers of light colored quartzite. However, in the majority of exposures of black phyllite S_0 has been obliterated by the penetrative schistosity (S_1).

On an outcrop scale the phyllites locally display asymmetric folding characterized by vergence (the direction of movement and rotation during deformation) to the northeast. This is compatible to the direction of the major anticlinal axis. In all cases observed, the upper fold limb dips 30° to 45° southwest. The lower and overturned limb dips 60° to 82° southwest. The axial plane of these folds is the S_1 foliation. The steep overturned limb, which locally is faulted, parallels the later crenulation cleavage (S_2). For the purpose of projecting bedding a general dip of 45° is recommended. For the time being, these folds are referred to as the "main phase" or "F_1" folds.

The penetrative foliation of the phyllites (S_1) is an axial plane schistosity. It is defined by a crystallization schistosity with micas oriented along foliation planes parallel to the composition laminations. In most outcrops of black phyllite S_0 has been transposed (rotated) into parallelism with S_1. Mineral segregation and redistribution (metamorphic differentiation) accompanied the transposition. In at least one case, the metamorphic differentiation is so well developed as to be easily mistaken for bedding laminations. The dip of S_1 ranges from 32° to 78° southwest but in general is about 55° to 60°.

Crenulation cleavage (S_2) is less commonly seen than S_1, and is particularly well developed in the vicinity of megascopic (main phase) folds. This coarse cleavage (spaced 5 to 10 cm) dips 68° to 85° southwest and appears to have arisen in S_1. With subsequent rotation S_2 has taken on the axial plane orientation of (now rotated) F_1 folds.

In the vicinity of the proposed adit portal a second crenulation cleavage (S_3) was noted. It is more coarsely spaced (1 to 2 cm) with a strike of 160° to 170°, dipping 60° to 70° southwest.

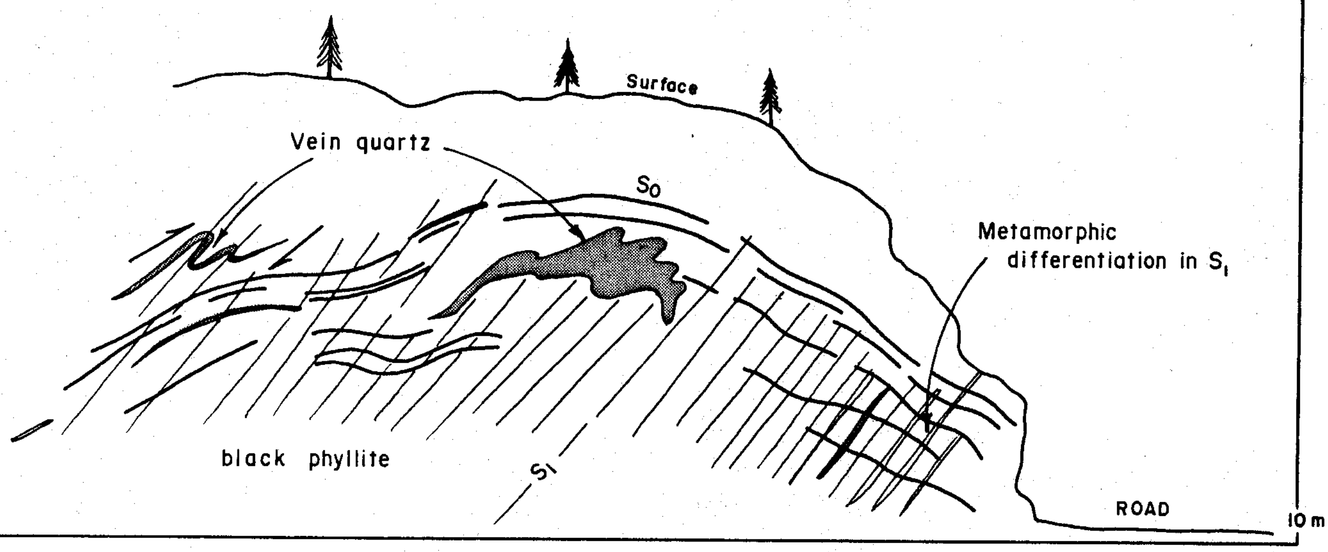
Figures 7, 8, and 9 illustrate characteristics of the foliations and folds. Figure 7 is a section along the main road, south of DDH-84-14. Here a broad, open fold displays northeastwards vergence of parasitic folds, pronounced metamorphic differentiation in S_1, and typical segregation and concentration of quartz in the hinge area,and in S_1.

This outcrop demonstrates that:

1) quartz veins develop or have been emplaced parallel to S_0, and

2) quartz veins also develop in S_1.

Figure 8 is a section along trench 8703 and illustrates the typical development of a gently dipping upper limb and steep, overturned lower limb. Note the lower limb has been brought into parallelism with the steep crenulation cleavage.



View to Northwest
Along Roadcut

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FOLDS AND QUARTZ VEINING MAIN ROAD NEAR GROUSE CREEK			
PROJECT 84-72C	DRAWN K.V.C.	DATE Nov. 1987	FIGURE 7
Revised		N.T.S. 93 A/7	
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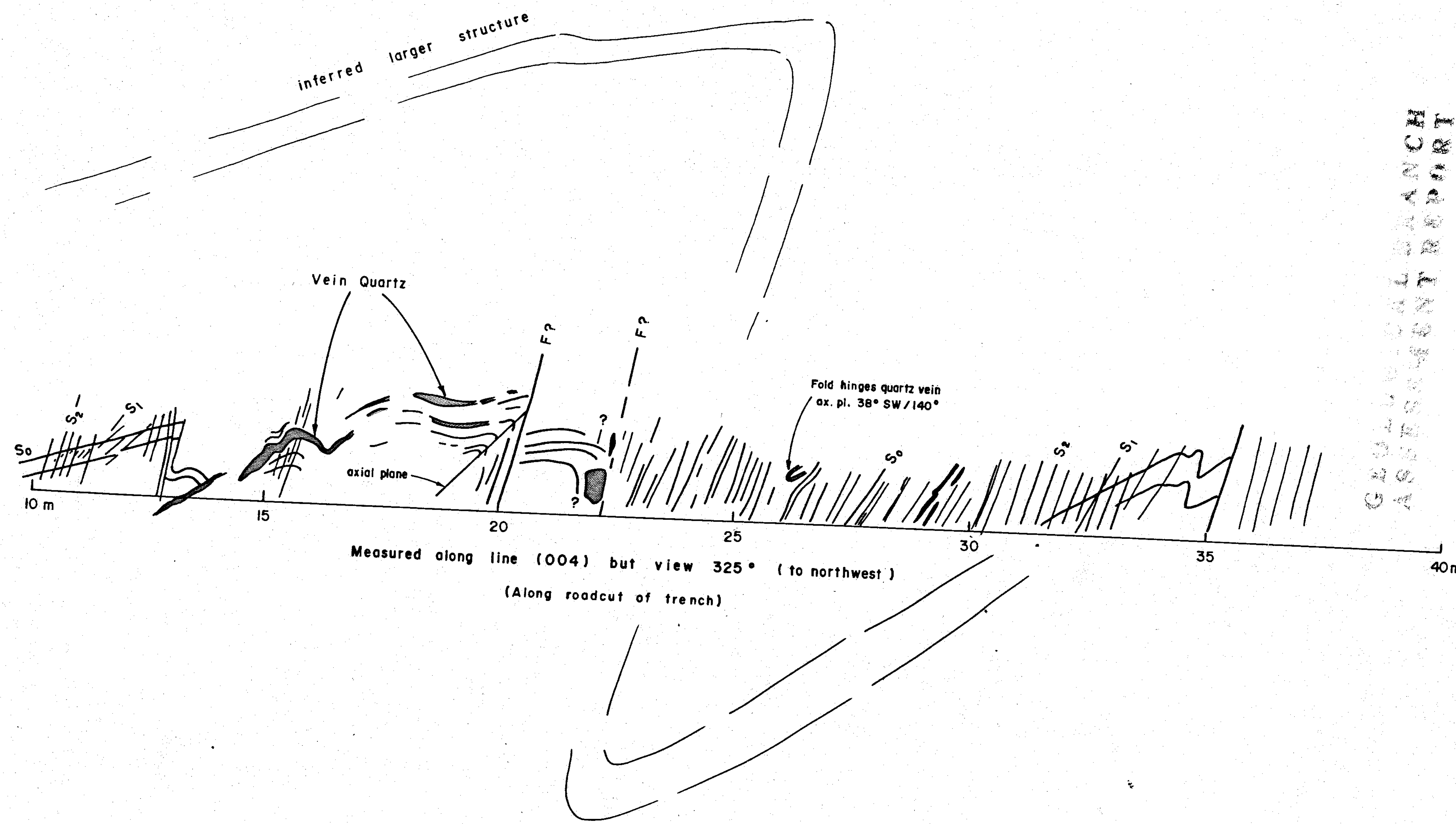

S_0 bedding

S_1 axial plane schistosity

S_2 crenulation cleavage

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FOLIATION AND QUARTZ VEINING VICINITY TRENCH 87-03			
PROJECT 84-72C	DRAWN K.V.C.	DATE Nov. 1987	FIGURE 8
Revised		N.T.S. 93 A/7	
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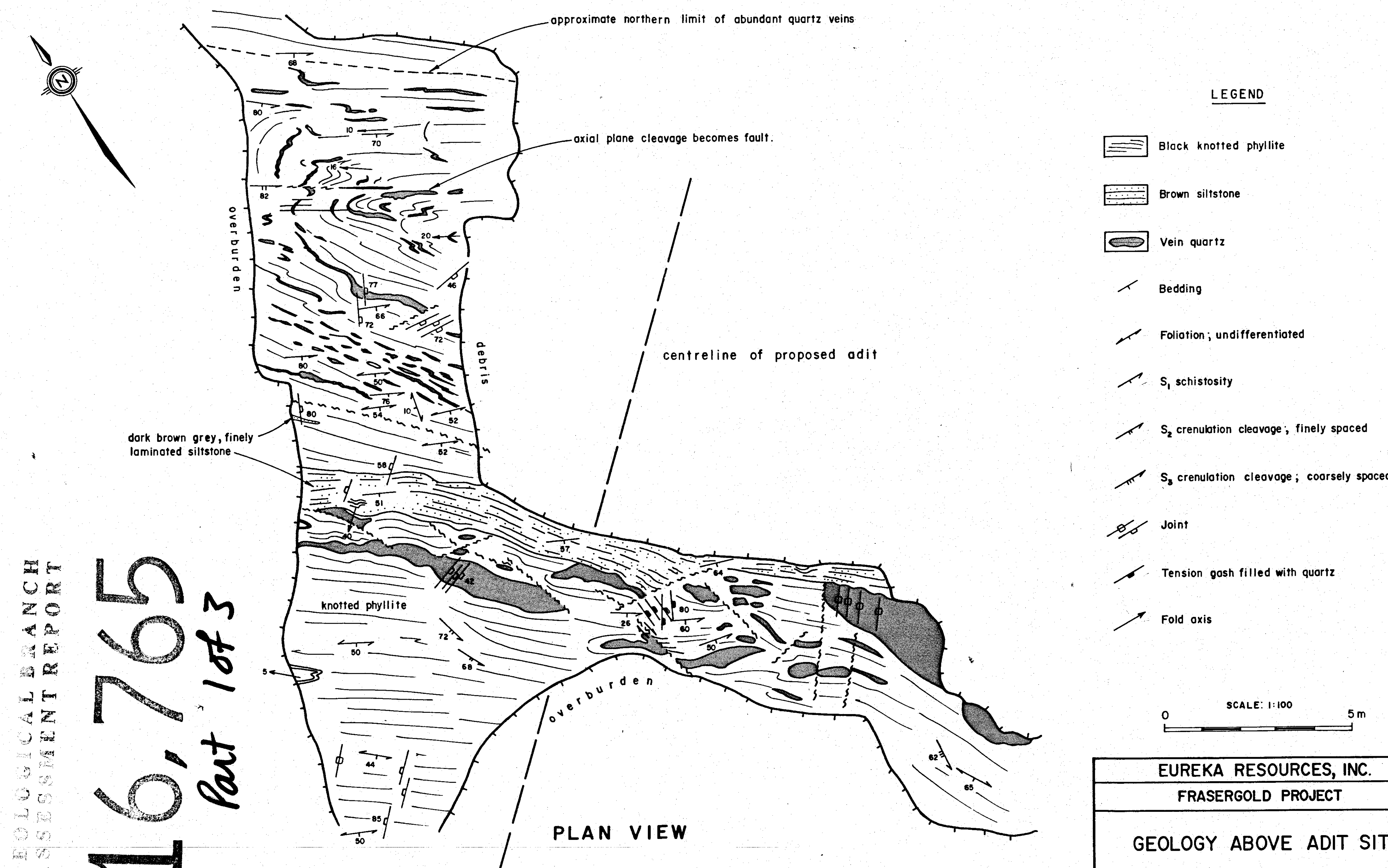
Between the 22 and 32 meter horizontal reference points in the figure the steep S_0 (which is parallel to S_1) represents the overturned limb of a larger fold whose inferred form is indicated on the sketch. Another feature of the folding seen in this locality is the prevalence of quartz swells, veins, and boudins in the hinge area of the open folds. It can be realized from this diagram that projection of veins arbitrarily along either the bedding, axial plane schistosity, or crenulation cleavage would not result in a correct interpretation of their extension. The structural importance of this outcrop is that it demonstrates quartz veins (or zones) on the upper limb of large folds (in the context of the property scale) can be projected for relatively short distances along a dip of about 35°, whereas on the steep, overturned limb quartz veins should be projected along the orientation of the crenulation cleavage at dips of about 80°.

It was first realized in 1984 that there was a spatial relationship between abundance of quartz veins and folds. It was also speculated then that there are discrete zones of folding. The present work supports these initial conclusions. The structural picture now emerging is that there are zones of folding which trend slightly west of the regional northwest strike through the Jay and Grouse Zones. These structural zones are where quartz veining and gold mineralization is concentrated.

Figure 9 is a outcrop plan above the proposed adit site. The area demonstrates a number of significant relations:

1) The zone of quartz veining along the southern edge of the cleared area has continuity, although individual veins do not. Pinching out in the S_1 foliation and truncation are clearly evident.



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FRASERGOLD PROJECT

GEOLOGY ABOVE ADIT SITE

PROJECT	DRAWN	DATE	FIGURE
84-74 C	K.V.C.	Nov. 1987	9
Revised		N.T.S. 93 A / 7	

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2) A large fold hinge occurs north of the brown siltstone unit. It is about the same scale as that inferred in Figure 8 and also clearly shows the transition from axial plane crenulation cleavage to fault in the hinge. This main phase fold plunges at a low angle to the northwest.

3) The true thickness of this quartz vein-rich zone is about 10 m, similar to that encountered in many of the drill holes.

A structural analysis of the field measurements assists in clarifing and summarizing the relations of the tectonites as follows:

1) The black phyllites are situated on the upright, southwesterly dipping limb of a syncline.

2) There are numerous smaller scale or parasitic folds on this limb, with amplitudes up to 5 m and wavelengths up to 10 m. These F_1 or main phase folds display vergence to the northeast. The folds are characterized by a gently dipping (30° SW/142°) upper limb and a steeply dipping, overturned (60° SW/130°) lower limb. The fold axes plunge 10° to 20° towards 310° +/- 10°. Away from area of folding the bedding dips 45° southwest.

3) The penetrative foliation of the phyllites is an axial plane schistosity (S_1) to these folds. It has a dip ranging from about 30° to 80°, but most commonly dips southwest about 55°. In most cases the bedding has been transposed to the attitude of the axial plane schistosity.

4) The crenulation cleavage (S_2) dips about 70° to 85° southwest. Crenulations of S_1 plunge 10-20° towards

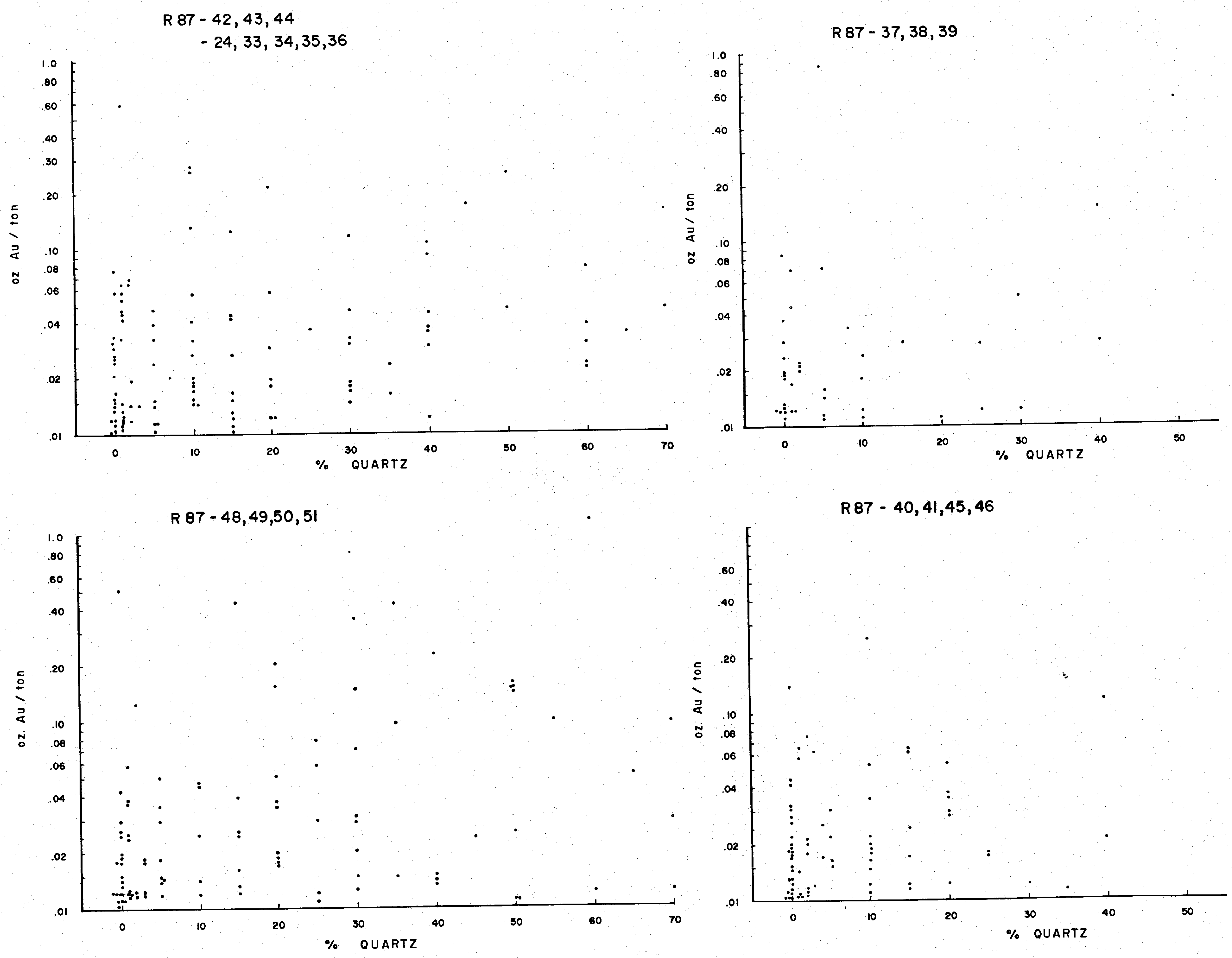
290° to 300°, about 10° to 15° west of the earlier (F_1) folds. Many quartz rolls, hinges and mullions have a similar orientation, leading to the conclusion that they developed contemporaneously to the crenulations. These features are then secondary or F_2 folds and tectonites. It appears that the crenulation cleavage arose in the S_1 schistosity. During further rotation of the F_1 folds, S_2 became the axial plane cleavage.

5) At a few places a second, coarser crenulation cleavage (S_3) was noted. It strikes 160° to 170° and dips 60° to 70° to the southwest. Also noted a few localities are isolated kink folds (F_3) which plunge 40° to the southwest.

2.2.3 Mineralization

Particulate gold mineralization occurs in quartz segregations; namely stringers, veins, boudins, and limbless rolls or mullions, within the base of the porphyroblastic ("knotted") phyllite unit. The origin of the quartz has been through secretions (mobilized "sweats") and metamorphic differentiation. The quartz is commonly nearly white, compact and is intimately associated with a white carbonate (dolomite - siderite). Pyrite and pyrrhotite are commonly found in the quartz veins in minor amounts.

In the 1986 drill program it was evident from the drill (core) logs that as the abundance of quartz increased so did the assay values. Contrary to this overall impression, many cases are known where high grade assays are returned from samples with very little quartz. The 1987 drill program indicated there is no obvious relationship between the amount of vein quartz in cuttings and the amount of gold. Figure 10 shows plots of the estimated gold content in the 1987 drill



GEOLOGICAL BRANCH
ASSESSMENT REPORT

16,765

Part 1 of 3

Legend

· Drill Sample Assay (oz Au / ton)

R87 - Drill Holes

EUREKA RESOURCES, INC.		
FRASERGOLD PROJECT		
COMPARISON OF GOLD VALUES TO QUARTZ CONTENT		
DATE:	SCALE: as shown N.T.S. 93 A/7	Figure 10
CAMPBELL & ASSOCIATES		

cuttings vs their assays. Many high gold values are found in cuttings with less than 5% quartz. This supports the observation made earlier, that specific quartz content does not relate directly to gold assays. It may also be that estimation of relative abundance of quartz in drill cuttings is not a reliable method.

Quartz veins and stringers are a common occurrence in the knotted phyllite unit. The veins are up to 1 m wide but in most cases are 2 to 20 cm in width and extend along strike 5 to 10 m. They are often disrupted, truncated or attenuated by the axial plane schistosity. The great majority of veins parallel S_1 where it is the only discernable foliation. In places where bedding can be identified (eg. Figures 7,8,9) quartz veins parallel S_0 or lie in the S_1 foliation.

The dynamo-thermal metamorphism producing the host phyllites was not a simple, single stage event but rather a series of crystallizations, cleavage developments, recrystallizations and metamorphic differentiations attendent to folding on a regional scale. As a result, quartz emplacement parallels various foliations (bedding, axial plane foliation and crenulation cleavage). Quartz also moved into hinge zones of small folds developed at the intersection of S_0 by S_1 and S_1 by S_2. It is quartz rolls and fold hinges of the latter intersection, producing F_2, that carried the spectacular gold seen in surface specimens. These features plunge at an angle of 10°, 10° to 20° slightly northwest of the strike of the knotted phyllite zone. It is considered likely that the youngest quartz emplacements contain the most gold, having crystallized last. It is noteworthy that similar transverse trends can be distinguished on the contoured geochemical map of gold in soils.

Because of the spatial relation between quartz veining and areas of folding it follows that mineralization should be sought where there has been the most deformation with its attendant migration of quartz solutions.

3 1987 WORK PROGRAM

3.1 Trenching

A total of 660 meters of new trenching was completed by bulldozer at several places across the projected mineralized zone. Ninety-five channel samples (T87-) were collected over 1.0, 1.5, or 2.0 meter intervals along 8 trenches, T8701 to T8708. A further 48 rock chip samples (C87-) of quartz veins were sampled while doing geologic mapping.

The location of trenched samples and their results are given in Figures 11a to c. The analytical procedure is described in Appendix I. Only some of the chip samples taken in 1986 are displayed, these are the assays plotted on the maps accompanying the 1986 report.

The first trench, T8701, was sited above the proposed adit portal across the lithologies shown in Figure 8. This same area of the Jay zone was trench sampled in 1986, but the results do not compare; T8701 reporting 0.021 oz/ton over 1½ m vs T8661202 carrying 0.072 oz/ton over the same quartz-rich interval. Visible gold has also been reported at this locality.

The second trench, T8702, along a drill access road, is also disappointing although it does indicate slight mineralization. Sample T870202 reports 0.031 oz Au/ton over 1½ meters. In the 1986 sampling T862702 carried 2.683 oz Au/ton over 1.0 m in the same area. Visible gold has been reported at this locality.

Chip samples of vein quartz over particular channel sample intervals were collected separately. Table 2 compares results of their assays vs those of the channel samples and demonstrates the lack of sample consistency.

Table 2. Results of Quartz Chip Sampling vs Channel Sampling

Quartz Chip Sample No.	Width* (m)	Assay Au oz/ton	Channel Sample No.	Width (m)	Assay Au oz/ton
C872108	1½	0.001	T870501	1½	0.277
C872201	1½	0.023	T870401	1½	0.066
C872202	3	0.053	T870402	1½	0.001
			T870403	1½	0.005

* Sample represents total vein quartz present over indicated width.

3.2 Reverse Circulation Drilling

3.2.1 Introduction

Locations of drill holes are shown in Figures 12a to c. Table 3 summarizes particulars of the drill holes, whose logs are given in Appendix II. Drill sections are given in Appendix III, which includes profiles of holes drilled in previous work programs. The location of the section grid is shown in Figure 3. Appendix IV cross references all drill holes, sections and figure numbers.

Reverse circulation drilling was undertaken in three areas along the mineralized zone outlined by previous programs. These areas are as follows:

1) Jay Zone, between Sections 53+00E and 55+00E, with a

Table 3. Summary of Reverse Circulation Drilling Program, 1987.

Hole	Section	UTM Co-ordinates North	UTM Co-ordinates East	Length (m)	Azimuth	Dip	Elevation (m)
R87-24	54+25E	97477.6	65291.5	75.0	045°	-50°	1530.7
R87-33	54+12E	97488.9	65284.0	73.0	045°	-50°	1530.6
R87-34	53+00E	97568.2	65195.5	70.5	046°	-50°	1536.0
R87-35	54+37E	97468.7	65300.4	72.0	048°	-50°	1531.2
R87-36	54+75E	97440.5	65321.8	69.0	045°	-50°	1532.3
R87-37	56+50E	97322.4	65449.7	82.5	047°	-50°	1517.4
R87-38	57+50E	97239.5	65504.2	91.5	045°	-60°	1515.1
R87-39	58+50E	97151.8	65561.7	79.5	045°	-50°	1519.5
R87-40	59+50E	97087.2	65631.5	81.0	045°	-50°	1515.0
R87-41	60+50E	97010.5	65697.6	90.0	045°	-55°	1509.8
R87-42	52+00E	97633.6	65122.9	98.0	045°	-50°	1539.2
R87-43	51+00E	97744.4	65080.3	93.0	045°	-50°	1522.5
R87-44	50+00E	97785.5	65019.1	70.5	048°	-50°	1520.6
R87-45	60+00E	97049.7	65669.2	81.0	047°	-50°	1511.4
R87-46	59+00E	97112.9	65605.6	90.0	043°	-50°	1517.0
R87-47	58+00E	97173.0	65492.5	15.0	047°	-60°	1537.7
R87-47A	58+00E	97169.3	65497.8	27.0	046°	-50°	1537.7
R87-48	57+00E	97248.8	65445.0	109.5	048°	-55°	1536.0
R87-49	56+00E	97342.9	65385.6	111.0	046°	-55°	1536.0
R87-50	55+25E	97397.0	65346.2	111.0	045°	-54°	1536.0
R87-51	57+50E	97198.4	65475.0	120.0	045°	-60°	1536.0
				1710.0m			

Drilling started July 5, 1987
Drilling stopped August 14, 1987

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strike length of about 250 m. Drilling achieved 25 m spacing between drill holes in this zone. Holes drilled in 1987 were R87-24,33,34,35 and 36.

2) Southeast extension of Jay Zone, between Sections 55+00E and 60+00E, with a strike length of 500 m. Holes drilled in 1987 were spaced about 100 m apart, but most were 25 to 50 m from previous holes. Holes drilled in 1987 were R87-37,38,39,40,41,45,46,47,47A,48,49,50 and 51.

3) Holes R87-42,43 and 44 were sited northwest of the Jay Zone to test for the latter's continuation there.

3.2.2 Method

SDS Drilling of Calgary, the drilling contractor, used a Bombardier track mounted unit, capable of drilling from vertical to 45° inclination. The diesel power plant, mounted on a separate tracked vehicle, supplied compressed air at 300 psi or pressurized water to flush the percussion cuttings from the hole to the surface. The center sample recovery drill stem consists of an inner pipe returning cuttings and an outer pipe down which air (or water) is pumped. The cuttings were separated from their carrying medium in the cyclone situated at the drill then dropped through the cyclone into 5 gallon buckets or, if desired, directly into a series of splitters. The 4½" diameter bit is capable of producing 80 to 100 lbs of rock chips for each 1.5 meter sample interval. Two assistants collected the samples taken at the drill.

Under dry conditions the cuttings were split 1 in 8 using a series of Jones riffle splitters. Under wet conditions, a similar split was made with a Humbel splitter. The dry split was divided into a storage sample and an assay sample. The wet split was sent for assaying while the storage sample was taken from the 7 in 8 portion to be discarded. Generally the assay sample weighed 5 to 10 lbs. There was concern that fines were lost with the water discharge. Sludge samples were taken for the first six holes and analysed for gold.

Drilling was rapid with 70 to 80 meters drilled in one 12 hour shift, and under dry conditions, recovery was good. Wet conditions produced by ground water created problems such as loss of fines and, what appeared to be, a greater amount of "blowby". Blowby is caused when compressed air escapes between the drill stem and the rock wall generally breaking out at the bottom of the casing. An added and unexpected problem associated with blowby resulted when dry cuttings encountered water seepage causing a buildup of mud near the surface. This mud collection in the drill hole was enough to prevent the drill bit being pulled out of the hole, at the end of the drilling.

Two holes, R87-47 and R87-47A, were not completed because of the depth of overburden. In both cases, the drill machine was not able to secure drill casing into bedrock. It is thought that several other holes, R87-24, R87-33 and R87-34, also shared the same problem in that the drill casing was not securely seated.

Geological notes were made as the cuttings came out of the cyclone, but eventually this practice was abandoned and the geologist examined the cuttings, storing a representative sample in a plastic vial. The cuttings were examined with a binocular microscope; noting alteration, amount and type of sulfides, rock type, amount and type of quartz, and reaction to HCl acid. No visible gold was observed in the cuttings.

Each sample of cuttings, which represented a 1½ meter interval, was fire assayed for gold. Selected samples were analysed for total metallic gold. Details of the two analytical procedures are given in Appendix I.

3.2.3 Results

Table 4 summarizes the anomalous gold values encountered in the reverse circulation drilling. All of the holes intersected gold mineralization above 0.020 oz/ton over widths up to nearly 30 m. These widths often include 7.5 to 12.0 m of 0.045 to 0.098 oz/ton (eg. R87-33,35,40,45,49, 50 and 51).

High grade assays (>0.2 oz/ton) over 1½ m intervals are reported in holes:

R87-33 (0.249 oz/ton), R87-37 (0.569 oz/ton)
R87-38 (0.858 oz/ton), R87-40 (0.365 oz/ton)
R87-44 (0.204 oz/ton), R87-45 (0.268 oz/ton)
R87-49 (0.220 oz/ton), R87-50 (0.356, 0.429, 1.085 and 0.510 oz/ton)
R87-51 (0.438 oz/ton).

These high values are probably due to sample inclusion of coarse gold and should be devalued by some factor. This is further discussed in Section 4.1.1

All of the holes were collared approximately at the same structural and stratigraphic level. In addition to the mid-depth mineralized zones (30-70 m) holes, there appears to be an upper level zone at depths of 7-30 m with significant gold. Examples are:

R87-37; 6.0 m of 0.185 oz/ton commencing at 7.5 m
R87-38; 7.5 m of 0.193 oz/ton commencing at 27 m

R87-40; 7.5 m of 0.096 oz/ton commencing at 25.5 m
R87-45; 10.5 m of 0.055 oz/ton commencing at 12 m
R87-49; 4.5 m of 0.061 oz/ton commencing at 22.5 m

Table 5 lists the total widths of significant mineralization in the drill holes along with the weighted average over that width.

In the Jay zone, along a strike length of 250 m, holes R87-33 to 36 intersect an average total width of 14.6 m with an average grade of 0.059 oz/ton Au. The 1986 results, mostly based on HQ drilling, indicated a 22 m width with an average grade of 0.060 oz/ton. When the total significantly mineralized widths and average grades of the 1986 and 1987 drilling in the Jay Zone are combined, as in Table 6, an average grade of 0.057 oz/ton Au over 17.9 m is indicated.

No factor, however, should be applied to the reverse circulation drilling results to bring them in line with those of the HQ drilling. Difficulties were experienced in sampling of the first five 1987 reverse circulation holes; i.e. in the Jay zone, and it is felt that the lower indicated gold values has much to do with these problems. A good example of this is provided by R87-24, which did not return any significant values, in comparison with DDH 86-24 which reported an 18.4 m interval with an average grade of 0.068 oz/ton. Such a marked discrepancy is suspicious as these two holes are only about 1 m apart and have a similar plunge and bearing.

In the southeast extension of the Jay Zone, along a strike length of 500 m, holes R87-37 to 41 and R87-45 to 51 intersect an average total width of 19.1 m with an average grade of 0.077 oz/ton Au. Six of the best holes in this

Table 4. Summary of Reverse Circulation Drill Results.

Hole No.	Interval (m)	Width (m)	Au oz/ton*	High Grade (assay/meters)
R87-24	21.0 - 28.5	7.5	0.017	
	39.0 - 51.0	12.0	0.020	
	54.0 - 67.5	13.5	0.024	
	39.0 - 67.5	28.5	0.020	
R87-33	13.5 - 18.0	4.5	0.026	
	24.0 - 27.0	3.0	0.076	
	37.5 - 48.0	10.5	0.073	0.249/1.5 @ 39 m
	51.0 - 55.5	4.5	0.032	
	13.5 - 27.0	13.5	0.029	
	37.5 - 55.5	18.0	0.052	
R87-34	15.0 - 33.0	18.0	0.037	0.105/4.5 @ 15 m
	57.0 - 58.5	1.5	0.041	
	49.5 - 60.0	10.5	0.017	
R87-35	22.5 - 28.5	6.0	0.044	
	39.0 - 49.5	10.5	0.045	
	55.5 - 60.0	4.5	0.032	
	63.0 - 66.0	3.0	0.051	
	39.0 - 66.0	27.0	0.030	
R87-36	28.5 - 31.5	3.0	0.142	
	43.5 - 45.0	1.5	0.036	
	49.5 - 51.0	1.5	0.064	
	57.0 - 67.5	10.5	0.033	
	39.0 - 67.5	28.5	0.022	
	49.5 - 63.0	13.5	0.033	
R87-37	7.5 - 13.5	6.0	0.185	0.569/1.5 @ 7.5 m
	21.0 - 22.5	1.5	0.070	
	34.5 - 39.0	4.5	0.023	
	7.5 - 22.5	15.0	0.083	
R87-38	27.0 - 34.5	7.5	0.193	0.858/1.5 @ 30 m
	37.5 - 42.0	4.5	0.041	
	63.0 - 67.5	4.5	0.039	
	70.5 - 72.0	1.5	0.025	
	27.0 - 42.0	15.0	0.110	

Table 4. (continued)

Hole No.	Interval (m)	Width (m)	Au oz/ton*	High Grade (assay/meters)
R87-39	6.0 - 9.0	3.0	0.030	
	39.0 - 42.0	3.0	0.033	
	67.5 - 69.0	1.5	0.020	
	39.0 - 48.0	9.0	0.021	
R87-40	12.0 - 15.0	3.0	0.047	
	25.5 - 33.0	7.5	0.096	0.365/1.5 @ 30 m
	49.5 - 54.0	4.5	0.063	0.137/1.5 @ 51 m
	63.0 - 72.0	9.0	0.031	
	12.0 - 31.5	19.5	0.049	
	49.5 - 72.0	22.5	0.028	
R87-41	63.0 - 70.5	7.5	0.035	0.106/1.5 @ 66 m
	43.5 - 70.5	27.0	0.017	
R87-42	85.5 - 87.0	1.5	0.027	
R87-43	19.5 - 21.0	1.5	0.059	
	55.5 - 58.5	3.0	0.026	
R87-44	31.5 - 37.5	6.0	0.022	
	61.5 - 67.5	6.0	0.057	0.204/1.5 @ 63 m
R87-45	12.0 - 22.5	10.5	0.055	0.268/1.5 @ 19.5 m
	46.5 - 49.5	3.0	0.025	
	57.0 - 64.5	7.5	0.027	
	76.5 - 81.0	4.5	0.027	
	46.5 - 64.5	18.0	0.019	
R87-46	46.5 - 57.0	10.5	0.045	
	58.5 - 63.0	4.5	0.021	
	46.5 - 69.0	22.5	0.029	
R87-47	Abandoned at 15 meters			
R87-47A	Abandoned at 27 meters			
R87-48	16.5 - 22.5	6.0	0.021	
	48.0 - 51.0	3.0	0.041	
	64.5 - 82.5	18.0	0.036	0.198/1.5 @ 76.5 m
	97.5 - 103.5	6.0	0.056	
	64.5 - 78.0	13.5	0.048	
	64.5 - 103.5	39.0	0.028	

Table 4. (continued)

Hole No.	Interval (m)	Width (m)	Au oz/ton*	High Grade (assay/meters)
R87-49	9.0 - 10.5	1.5	0.029	
	22.5 - 27.0	4.5	0.061	0.155/1.5 @ 22.5 m
	34.5 - 46.5	12.0	0.097	
	73.5 - 79.5	6.0	0.023	
	90.0 - 93.0	3.0	0.025	
	109.9 - 111.0	1.5	0.027	
	22.5 - 46.5	24.0	0.062	
R87-50	33.0 - 39.0	6.0	0.136	0.253/3 @ 33 m
	40.5 - 55.5	15.0	0.194	0.559/4.5 @ 43.5 m
	66.0 - 69.0	3.0	0.264	0.510/1.5 @ 67.5 m
	70.5 - 81.0	10.5	0.036	
	85.5 - 88.5	3.0	0.027	
	94.5 - 99.0	4.5	0.021	
	102.0 - 103.5	1.5	0.095	
	33.0 - 55.5	22.5	0.165	
	66.0 - 88.5	22.5	0.058	
	94.5 - 103.5	9.0	0.029	
	33.0 - 103.5	70.5	0.076	
R87-51	13.5 - 15.0	1.5	0.026	
	75.0 - 82.5	7.5	0.098	0.438/1.5 @ 81 m
	102.0 - 103.5	1.5	0.028	

* Note: conventional fire assay

Table 5. Total Widths and Average Grades of Significant Mineralization.

Drill Hole	Total Width (m)	Average Grade (Au oz/ton)
R87-44	6.0	.057
R87-34	4.5	.105
R87-33	18.0	.063
R87-35	19.5	.046
R87-36	16.5	.056
R87-50	46.5	.111
R87-49	24.0	.062
R87-37	12.0	.110
R87-48	24.0	.047
R87-38	21.0	.089
R87-51	10.5	.078
R87-46	10.5	.045
R87-40	24.0	.059
R87-45	13.5	.056
R87-41	4.5	.050
Average	17.0	.072

Note: Drill holes listed from northwest to southeast.

Table 6. Summary of Drill Results from Jay Zone 1986 and 1987 Drill Programs

Drill Hole	Total Width* (m)	Average Grade (Au oz/ton)
R86-2	39.0	.057
R86-9B	3.0	.056
86-15	3.7	.039
86-18	37.5	.072
86-19	21.1	.028
86-23	36.5	.047
86-24	18.4	.068
86-26	3.5	.061
86-27	11.0	.068
R87-33	18.0	.063
R87-34	4.5	.105
R87-35	19.5	.046
R87-36	16.5	.056
Average	17.9	.057

* Refers to significant mineralization.

Note: Holes 86-16,17,20 report only anomalous values.

area, just southeast of the Jay Zone and including R87-50,49,37,48,38 and 51 along a strike length of 225 m, intersect an average total width of 23 m with an average grade of 0.083 oz/ton. The same holes display an enriched zone, within the wider lower grade zone, averaging 11.3 m width and 0.155 oz/ton.

There is a good possibility that there are other high grade zones within the mineralized band. The following holes reported two or more consecutive sample intervals with grades more than 0.1 oz/ton Au:

R87-36; 3 m averaging 0.142 oz/ton
R87-37; 3 m averaging 0.709 oz/ton
R87-49; 6 m averaging 0.157 oz/ton
R87-50; 3 m averaging 0.253 and 4½ m averaging 0.559 oz/ton.

Northwest of the Jay Zone, holes R87-42, 43 and 44 did not intersect any zones as mineralized or as wide as those to the southwest. However, they did intersect narrow widths of low and marginal grade and one sample interval of 0.204 oz/ton Au (R87-44). These results indicate the mineralized horizon does extend northwestwards, although significantly mineralized sections have not yet been intersected.

4 DISCUSSION

4.1 Sludge vs Cuttings Assays

Table 7 compares gold values of sludge samples with the average gold value of drill cuttings over the same sludge sample interval. Gold values of the sludge range from 6% lower to 78% higher than those of the cuttings. The average sludge gold value is about 30% higher. The explanation for this is thought to be that the sulfides, which carry some proportion of gold, are more likely to be pulverised into fines thereby increasing the likelihood of a representative sample; i.e. decreasing the nugget effect.

4.2 Conventional vs Total Metallic Assays

Total metallic assays involve grinding the entire sample. All of the oversize material (+120 mesh) is fire assayed. The undersize material (minus-120 mesh) are fired by a normal technique, using two samples of 500 gm sample of pulverized minus-100 mesh material. These two results are summed and a overall assay calculated. Metallic assays are considered to be the more accurate.

In 1986 metallic assays were run on eight samples containing visible gold. The average of these was nearly twice that of the average conventional assay. The 1987 program included many more metallic assays with the objective of determining whether or not a factor could be applied to adjust conventional assays. After fire assays were received, selected sections of mineralized core were submitted for metallic gold assays. In all, 139 samples were re-assayed.

Figure 13 is a plot of gold values reported by the two assay

Table 7. Comparison of Gold Content of Sludge Samples with that of Drill Cuttings.

Hole No.	Interval (m)	Nature	Au oz/ton Sludge	Au oz/ton Cuttings*
R87-24	6 - 13.5	dry	0.001	0.002
	13.5 - 21	dry	0.006	0.004
	21 - 28.5	dry	0.030	0.017
	28.5 - 36	dry	0.009	0.002
	36 - 43.5	dry	0.015	0.018
	43.5 - 51	dry	0.020	0.018
	51 - 58.5	dry	0.011	0.011
	58.5 - 66	dry	0.050	0.030
	66 - 72	wet	0.005	0.006
		Average	0.016	0.012
		Sludge	33% higher	
R87-33	12 - 18	dry	0.067	0.021
	18 - 24	mixed	0.014	0.007
	24 - 31.5	dry	0.027	0.032
	31.5 - 39	wet	0.017	0.008
	39 - 43.5	wet	0.106	0.115
	43.5 - 48	wet	0.040	0.051
	48 - 55.5	wet	0.020	0.022
	55.5 - 63	wet	0.011	0.005
	63 - 70.5	wet	0.001	0.003
		Average	0.034	0.029
		Sludge	17% higher	
R87-34	9 - 15	mixed	0.001	0.004
	15 - 21	wet	0.082	0.081
	21 - 28.5	wet	0.011	0.014
	28.5 - 36	wet	0.013	0.012
	36 - 43.5	wet	0.002	0.003
	43.5 - 51	wet	0.011	0.004
	51 - 58.5	mixed	0.016	0.017
	58.5 - 64.5	wet	0.001	0.007
	64.5 - 69	wet	0.001	0.001
		Average	0.015	0.016
		Sludge	6% less	

Table 7. (continued)

Hole No.	Interval (m)	Nature	Au oz/ton Sludge	Au oz/ton Cuttings*
R87-35	7.5 - 15	wet	0.001	0.003
	15 - 22.5	wet	0.001	0.002
	22.5 - 30	mixed	0.057	0.035
	30 - 37.5	wet	0.013	0.003
	37.5 - 45	wet	0.124	0.045
	45 - 52.5	wet	0.041	0.024
	52.5 - 60	wet	0.018	0.020
	60 - 66	wet	0.035	0.028
	66 - 72	wet	0.001	0.003
		Average	0.032	0.013
		Sludge	78% higher	
R87-36	7.5 - 15	dry	0.001	0.002
	15 - 22.5	dry	0.001	0.002
	22.5 - 30	mixed	0.088	0.023
	30 - 37.5	wet	0.031	0.036
	37.5 - 45	wet	0.009	0.016
	45 - 52.5	wet	0.005	0.015
	52.5 - 60	wet	0.023	0.025
	60 - 67.5	wet	0.035	0.027
		Average	0.024	0.018
		Sludge	33% higher	
R87-37	7.5 - 15	wet	0.116	0.148
	15 - 22.5	wet	0.006	0.0

techniques for the 139 samples taken in 1987 and the 8 samples run in 1986. The 45° diagonal line, "line of unity" or "LU", is the locus of equality.

Based on the distribution and pattern of points the plot is divided somewhat arbitrarily into four fields, A to D, described below. Summary statistics are given in Table 8. The discussion is based on the premise that the metallic assays approximate the "true" and representative gold content of low (Field A) and medium grade (Fields B and C) samples. It is possible that conventional assays give a satisfactory estimate of the gold content in high grade samples (Field D).

<u>Field A</u> (LU value <0.025) is characterized by a relatively tight grouping of plots on either side of LU, suggesting the variance is more an expression of analytical sensitivity than some basic difference bewteen the values. The statistics support this, showing little significant difference between the analytical variance. This is taken to indicate a "normal" distribution of assays for rocks with a very low gold content and no adjustment factor is warrented.

<u>Field B</u> (LU value >0.025,<0.0625) displays a greater spread of plots with a noticeable shift to the metallic assay side of LU; i.e. the metallic assay values are generally (but not always) higher than the normal assays. The statistics confirm this and indicate that on average the conventional assays can be increased by a factor of 22% to match the metallic assays.

<u>Field C</u> (LU >0.0625, <0.125) the shift to the metallic side of the diagram is more pronounced, with 12 of 14 metallic assays reporting higher values than the normal assays. On average the fire assays can be increased by a factor of 40%

to match the metallic assays.

Field D (LU >0.125) shows a marked contrast to the low and medium grade fields with a definite weighting in favor of the conventional assays. Deleting one sample with a fire assay of 0.268 oz/ton and a metallic assay with 0.032 oz/ton, the statistics indicate the conventional assays should be deflated by a factor of 34% to match the metallic assays.

The shift of the plots from the metallic to conventional assay side of the diagram is believed to be an expression of the varying grain size of particulate gold in the samples. In "low grade" samples (in general, <0.05 oz/ton) the gold is considered to be fine grained and/or distributed evenly (even if present only in sparse amounts).

In "medium grade" samples (with 0.05 to 0.125 oz/ton Au) the gold is thought to be coarser grained relative to the low grade samples. These particles are probably not uniformly distributed and the samples analysed by conventional techniques did not include (in general) a representative amount of the coarser particles, thereby giving a low estimate of the "true" grade. This is intuited by the fact that if medium grade samples were simply those with more fine particles, then their plots in Figure 13 would lie on LU, providing they were uniformly distributed.

Conventionally assayed samples with >0.125 oz/ton Au, probably do include one or more coarse gold grains. Grades calculated from their presence cannot be discounted but neither should they be over-emphasized. As there is probably only a limited number of coarse grains in a particular volume (unspecified) the balance of the core that was run for total metallic gold would not have included such large grains.

Table 8. Comparison of Metallic and Conventional Assays (Au oz/ton). To accompany Figure 13.

	Number	Range	Mean	Standard Deviation
Field A				
(LU <0.025)				
Conventional Assays	64	0.001 - 0.027	0.013	0.007
Metallic Assays	64	0.001 - 0.026	0.012	0.006
Field B				
(LU >0.025,<0.0625)				
Conventional Assays	52	0.010 - 0.078	0.036	0.014
Metallic Assays	52	0.021 - 0.072	0.044	0.014
Field D				
(LU >0.0625,<0.125)				
Conventional Assays	14	0.035 - 0.106	0.072	0.020
Metallic Assays	14	0.056 - 0.152	0.101	0.028
Field D				
(LU >0.125)				
Conventional Assays	8	0.137 - 1.085	0.360	0.313
Metallic Assays	8	0.105 - 0.543	0.239	0.151

Table 9 lists the high grade assays on which the 34% deflation factor is based, on the assumption that the metallic assays are more representative of the grade. Hole R87-50 has two pairs of consecutive samples with very similar metallic assays; 33 to 36 m reporting 0.178 and 0.177 oz/t and 43.5 to 46.5 m reporting 0.189 and 0.190 oz/t. These results are in marked contrast to the erratic fire assays over the same intervals. The conclusion drawn is that the metallic values approximate the correct grade over each of the 3 meter intervals. The average fire assay would be reduced 28% for the 33 to 36 m interval, and 36% for the 43.5 to 46.5 m interval. These factors compare well to the calculated overall deflation factor of 34%. It should also be noted that a much greater percentage of the high grade samples were re-analysed by the total metallic method than the low and medium grade samples, thereby distorting the statistics somehow.

In summary, the following preliminary conclusions are reached:

1) No adjustment factor is warranted for fire assayed, low grade samples.

2) Medium grade samples can be adjusted 20 to 40% upwards. A conservative factor of 15% is suggested.

3) High grade samples should be devalued 34%.

The flaw in applying any factor to conventional assays is the uncertainty as to which field the assay would plot in Figure 13. For example, a fire assay value of 0.225 could be in either Field C (warranting a +15% adjustment) or in Field D (requiring a -34% adjustment).

Table 9. Comparison of High Grade Conventional Fire Assays with Metallic Asssays

Hole No.	Interval (m)	Fire Assay (Au oz/ton)	Metallic Assay (Au oz/ton)	% Difference*
R87-40	30.0-31.5	0.0365	0.041	91
	51.0-51.5	0.137	0.132	104
R87-45	19.5-21.0	0.268	0.032	837
R87-48	16.5-78.0	0.198	0.105	189
R87-50	33.0-34.5	0.149	0.178	84
	34.5-36.0	0.356	0.177	201
	43.5-45.0	0.429	0.189	227
	45.0-46.5	0.162	0.190	85
	46.5-48.0	1.085	0.543	200
	Average**	0.360	0.239	148

* Fire assay expressed as % of metallic assay.

** Sample R87-45 deleted from calculation.

4.3 Potential Ore Grade and Width

The Jay Zone, including its southeast extension, has a total known strike length of about 750 m. Of this distance the northwesternmost 250 m has the potential to host an average total width of mineralization 17.9 m thick grading 0.057 oz/ton Au. While not being drilled as closely as the northwestern Jay Zone, the next 500 m strike length to the southeast has the potential of containing a total width of 19.1 m averaging 0.077 oz/ton Au. The drilling indicates that there is a good possibility this southeastern section includes 11.3 m with an average grade of 0.155 oz/ton.

When these estimates of average grade are weighted over their relative strike length, the total 750 m length has a drilled and geologically indicated average grade of 0.071 oz/ton Au over 17.6 m. The authors of this report consider that the average grade, based on conventional fire assays, can be adjusted upwards by about 15% to approximately 0.081 oz/ton to reflect a value as would be indicated if metallic assays had been employed. However, the upgrading factor has not been used in this report.

This 750 m length is half of the total 1.5 km strike length of mineralization indicated by the 1984 drilling. Given a total mineralized thickness of about 20 m and down-dip extension of 70 m for every 50 m vertical depth, then there is a strong potential of the 1.5 km long zone to yield in the neighbourhood of 6.1 million tons. If mineralization continues to 150 m depth, as indicated by earlier drill programs, then the same area could yield 18 to 20 million tons. The property is still in the exploration stage and much has yet to be done before such a preliminary estimate can be refined. The situation of other mineralized zones stratigraphically above and below the main band of mineralization addressed in this report, as indicated by earlier drill programs (such as the Grouse Zone) can only enhance the potential of the property.

It is thought that relatively high grade mineralization is indicated where two or more consecutive samples report more than 0.1 oz/ton Au. Five such occurrences were drilled in 1987. These include 3 m of 0.142, 0.253 and 0.709 oz/ton Au, 4½ m of 0.559 oz/ton Au and 6 m of 0.157 oz/ton Au. It has not yet been established if these enriched zones have continuity or if they are spatially related to fold hinge or other structural zones.

In summary, the findings of earlier programs and this study reveals extreme variations in grade estimates depending upon:

1) sample size; NQ (1 7/8" diameter vs HQ (2½" diameter) drill core,

2) drilling method; HQ drill core vs reverse circulation (4½" diameter) drill cuttings, and

3) conventional assays vs metallic assays.

The conclusion from examining the variation of the different methods is that it is not a routine matter to provide reliable grade estimates of the Frasergold type of deposit. Such an estimate can only be accomplished by a well coordinated bulk-sampling program, testing the full width of mineraliation, processing a large portion (say 50%) of the mined ore. Due to possible chemical leaching of surface ore, this program would best be accomplished by one or more underground test adits. Processing the ore while mining would serve as a further guide to the nature of the ore bodies and direction of the adit. For example, if bulk sampling did determine that mineralization is concentrated in the steeply dipping hinge zone of folds, as per the current model, then changes to the adit route could be made directly.

5 CONCLUSIONS

5.1 Lithology

Upper Triassic, lustrous black phyllites with abundant porphyroblasts of sideritic carbonate host gold-bearing quartz veins on the Frasergold property. Interbeds of gray to black calcareous and graphitic phyllites, siltite and green carbonate-chlorite-sericite schist occur in subordinate amounts. The black phyllite has been regionally metamorphosed to the greenschist facies, along with under- and overlying metasediments of similar age.

5.2 Structure

The phyllites dip about 45° to the southwest and are situated on the upright limb of the Eureka syncline. A penetrative schistosity (S_1) is the axial plane foliation in the region and dips 55° to 60° to the southwest. A steep crenulation cleavage (S_2) is locally well developed and parallels the lower limbs of F_1 folds of schistosity and bedding. It also approximates the axial plane of rotated F_1 folds.

An understanding of the folds is critical to the proposed model of mineralization. They are not developed everywhere along the geochemically anomalous band of phyllites but appear to be restricted to zones of deformation. They display a pronounced asymmetry, with gently dipping upper limbs and steeply overturned lower limbs. Their hinge zone is commonly a locus of shearing and faulting and their fold axes plunge at a shallow angle to the northwest. The trace of their axial plane strikes 10° to 20° west of the predominant schistosity and lithological trends. The axial plane crenulation cleavage originated as S_1 and has been rotated into its present orientation; i.e. the axial planes of F_2 folds. While overturned limbs of these megascopic folds have been observed there is no evidence to support large scale overturning; drill data must be interpreted as representing upright lithological sections, allowing for relatively short lengths of local, steep overturning.

5.3 Mineralization

Gold occurs in quartz veins, boudins, mullions and rolls and there have been numerous observations of fine to coarse grained (up to 5 mm diameter) native gold with vein quartz and associated white sideritic dolomite. Gold probably occurs in pyrite and pyrrhotite in quartz but no definitive examinations have been made. An appreciation of the distribution of quartz veins is critical to the interpretation of drill hole data and to exploration of the property.

Quartz veins typically occur in swarms and are up to 1½ m wide, but usually are less than ½ m. Veins are short and discontinuous, having been boudinaged, folded, sheared and faulted.

Quartz arose as a direct result of regional metamorphism and metamorphic differentiation. Sweats moved into first, the bedding planes and second, axial planes of folds. The great majority of quartz veins parallel bedding (which may be transposed), but quartz solutions were still mobile during folding of schistosity (F_1) and development of the axial plane crenulation cleavage (F_2). It is thought that gold would accompany quartz as long as the latter was fluid. This would explain the observation of gold in small fold hinges, rolls and mullions which plunge gently to the northwest.

Structural observations and studies indicate that in general, zones of quartz veining can be projected to surface along the bedding. It is also inferred, but not proved, that areas of relatively high grade mineralization are developed in fold hinges which can be projected along the steep crenulation cleavage forming the axial planes of these folds. These are the fold hinges where visible gold has been found. Furthermore, the trace of these hinge zones crosses the regional lithological trend at an acute angle, striking 10° to 20° west of the latter.

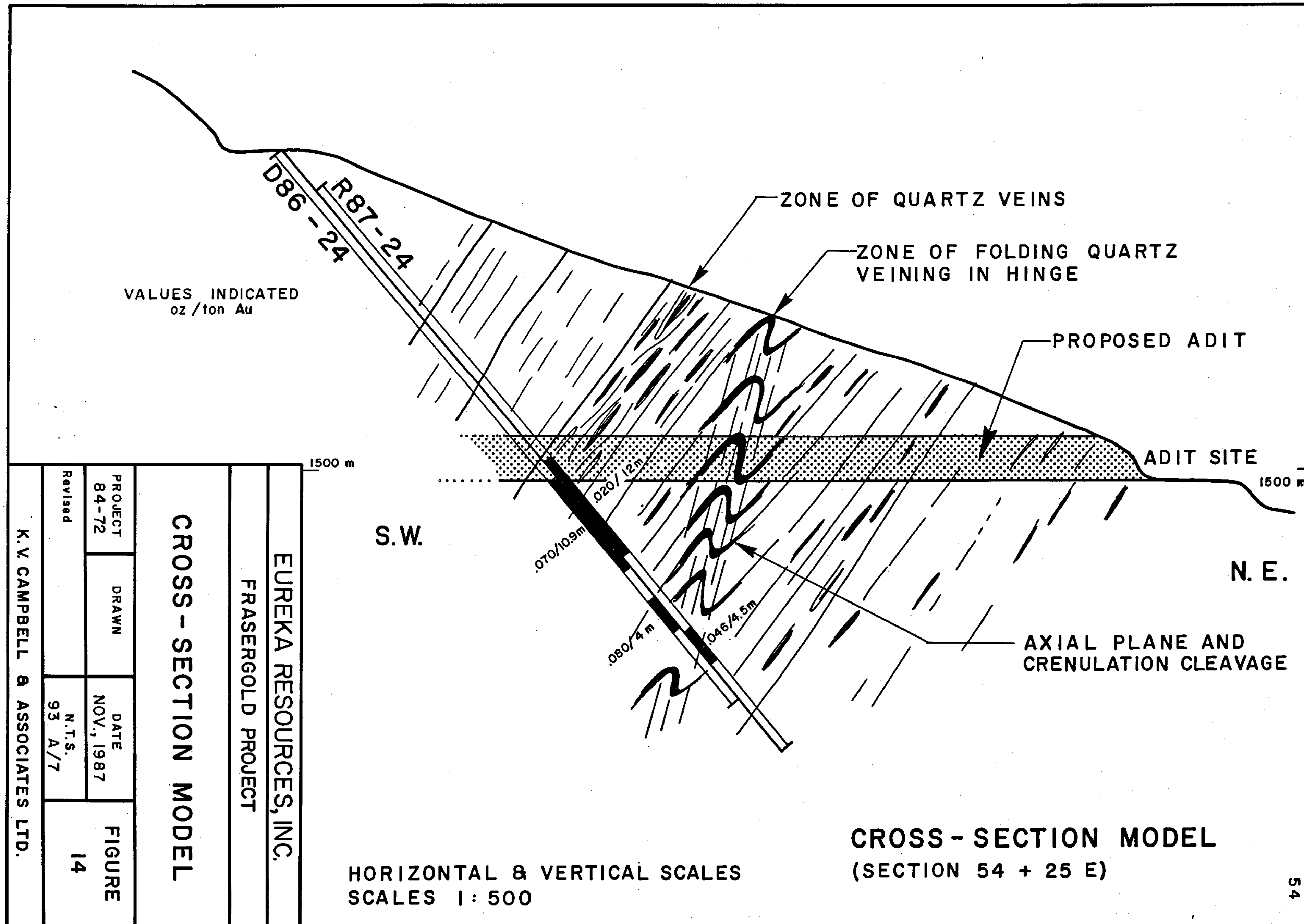
Figure 14 is a sketch model of the inferred structural style and associated mineralization, drawn through Section 54+25E, parallel to R87-24 in the vicinity of the proposed adit. It illustrates the scheme of projection used to correlate drill results and surface geology there.

Figure 15 is a 1:2500 compilation of the mineralized zones, showing drill locations and interpreted zones of mineralization projected to surface along the bedding orientation.

5.4 Estimate of Ore Potential

Results of the 1987 drill program are similar to and confirm the results of previous programs. All of the completed holes intersected gold mineralization above 0.020 oz/ton Au over 30 m. These widths often include 7½ to 12 m widths of 0.045 to 0.098 oz/ton Au. Numerous higher grade assays (>0.2 oz/ton) over 1½ m intervals are reported.

The geochemically anomalous band, which is believed to represent the mineralized, knotted black phyllites, extends along the property for about 10 km with a width of



VALUES INDICATED
oz /ton Au
D86 - 24
R87 - 24
ZONE OF QUARTZ VEINS
ZONE OF FOLDING QUARTZ
VEINING IN HINGE
PROPOSED ADIT
ADIT SITE
1500 m
1500 m
.020/ 12m
.070/10.9m
.080/4 m
.046/4.5m
S.W.
N. E.
AXIAL PLANE AND
CRENULATION CLEAVAGE
HORIZONTAL & VERTICAL SCALES
SCALES 1 : 500
CROSS - SECTION MODEL
(SECTION 54 + 25 E)
EUREKA RESOURCES, INC.
FRASERGOLD PROJECT
CROSS - SECTION MODEL
PROJECT
84-72
DRAWN
DATE
NOV., 1987
N.T.S.
93 A/7
Revised
FIGURE
14
K. V. CAMPBELL & ASSOCIATES LTD.

approximately 100 m. Of this distance, some 1½ km have been drilled to varying degrees of confidence. Spotty, erratic geochemical anomalies on the Mac 10 claim may indicate an additional 2 to 3 km distance to the potential length of the favorable horizon (Leishman, 1987).

The most drilled section, the 250 m long northwest section of the Jay Zone (drilled at 25 m intervals), has a drill indicated total mineralized width of 17.9 m with an average grade of 0.057 oz/ton Au, (combined 1986 and 1987 results). The adjacent 500 m southeast extension (drilled at 50 to 75 m intervals) has a drill indicated total mineralized width of 19.1 m with an average grade of 0.077 oz/ton Au.

The next 750 m section to the southeast was drilled at 100 to 250 m intervals in 1984, at which time mineralization was reported in amounts similar to those first reported in the Jay and Grouse Zones.

Two and a half kilometers to the northwest of the Jay Zone, preliminary drilling has shown that the geochemically anomalous band there is underlain by the same black phyllites with quartz veins. Mineralization is reported to be similar to that in the more drilled sections. This indicates that gold mineralization has a strong likelihood of extending at least 4 km along strike, and probably the entire 10 km along the property if similar relations exist between topography, geochemistry, lithology, structure and mineralization as they appear to.

Given the dimensions of the Jay and Grouse Zones there is a good likelihood of the Frasergold property hosting about 20 million tons of ore with an average grade of 0.05 to 0.08 oz/ton Au. This tonnage is that contained within a block approximately 1½ km long, 150 m deep and 150 m wide.

Enriched zones do occur along the mineralized band. High grade assays over consecutive drill sample intervals indicate the possibility of such zones to carry 0.142 to 0.709 oz Au/ton over 3 to 6 m widths. Southeast of the proposed adit site, along a strike length of 225 m, drilling has indicated an enriched zone averaging 11.3 m width and 0.155 oz/ton Au. If these widths (and grades) are continuous, then there could be ½ million tons of enriched material along this short distance to depths of 50 m. These zones should be explored by underground drifting.

Figure 16 is a 1:10,000 compilation of the Frasergold property showing the extent of the gold geochemical soil anomaly and the major work areas. It provides an overview of the geological potential of the property. Noteworthy are the subsiduary, slightly transverse geochemical trends. These have the same orientation as the trace of the younger folds and crenulation cleavage, suggesting a relation between them.

Uncertainties in the grade determination, brought about by the dispersed, particulate nature of the gold, make it necessary to drive one or more underground test adits across and along the mineralized zone. The inferred structurally controlled model of mineralization proposed makes it critical that bulk milling proceed concurrently with the drifting, so that necessary changes in the routing of the underground work can be made quickly and efficiently.

6 PROPOSAL FOR FURTHER DEVELOPMENT

6.1 Recommendations

It is recommended that the larger program proposed in 1986 be continued and that trenching, drilling (core and reverse circulation) and underground exploration proceed with the objective of thouroughly testing the 1½ km long stretch of known mineralization.

A two phase drilling and underground program is recommended. The focus of Phase I is on the Jay Zone. Phase II would explore the extension of that zone to the southeast. Some drilling in both phases would be along the geochemically anomalous band to the northwest of the Jay Zone.

The underground work would include the establishment of a small mill facility to process bulk samples along the adit. Drilling should be a combination of diamond drilling for geological control and reverse circulation drilling for reasons of speed and economy. Some trenching is anticipated.

6.2 Estimated Cost

Phase I May - July 31, 1988

Drilling:		
2,000 meters of HQ diamond drilling @ $150/meter	$	300,000
3,000 meters of reverse circulation drilling @ $80/meter	$	240,000
Mining: 500' Adit @ $400/foot	$	200,000
Milling: 3 months @ $60,000/month	$	180,000
Surveying, Engineering and Environmental Services	$	35,000
Trenching and Site Preparation	$	50,000
Total Estimated Cost	$	1,005,000
Contingency 10%	$	100,500
Total Phase I Budget	$	1,105,500

Phase II August 1 - December, 1988

Drilling:		
2,000 meters of HQ diamond driling @ $150/m	$	300,000
5,000 meters of reverse circulation drilling @ $80/m	$	400,000
Mining: 500' Adit @ $400/foot	$	200,000
Milling: 3 month rental @ $60,000/mo	$	180,000
Site preparation and Road uilding	$	100,000
Surveying, Engineering and Mine Design	$	150,000
Environmental Services	$	30,000
Metallurgical Studies and Laboratory Work	$	30,000
Total Estimated Cost ...	$	1,515,000
Contingency 10%	$	151,500
Total Phase II Budget ..	$	1,666,500
Total Budget, Phase I and II	$	2,772,000

7 BIBLIOGRAPHY

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8 ITEMIZED COST STATEMENT

Field Programme - June 1 - August 31, 1987

PROJECT MANAGEMENT

John R. Kerr, P.Eng.
70.75 days @ $325.00/day $22,993.75

LABOUR

Jeannette Kerr, Cook
77 days @ $125.00/day $9,625.00
Michael Montagne, Expediter & Camp Attendant
67.5 days @ $100.00/day $6,750.00
Kathy Thompson, Sampler
79 days @ $80.00/day $6,380.00
Paul Campbell, Field Assistant
55 days @ $70.00/day $3,850.00
Valerie Kerr, Field Assistant
35 days @ $60.00/day $2,100.00
Laurence Giesbrecht, Casual Labour
28 hours @ $12.00/hour $336.00

TOTAL $28,981.00

Employee Costs, plus 20% $5,796.20

$34,777.20

CONTRACT SERVICES

W.H. Thompson, Site Construction
24 days @ $175.00/day $4,200.00
Boyd McKean, B.Sc.
77 days @ $160.00/day $12,320.00

$16,520.00

CONSULTING SERVICES

K.V. Campbell & Associates Ltd. ...$5,413.63
Kilborn Engineering $7,207.65
D.A. Leishman, B.Sc. $5,583.96
Norelco Environmental Consultants Ltd. $5,256.85

$23,462.09

DRILLING COSTS

SDS Drilling Ltd.
1710 meters Reverse Circulation $71,334.40

BULLDOZING SERVICES

Gruhs Bulldozing Ltd.
Site Preparation $22,290.00
Trenching $11,136.00
$33,426.20

ASSAYS COSTS

Min-En Laboratories Ltd. &
B.C. Research $29,763.20

CAMP COSTS
(20 man trailer complex)

Camp Rental 3 months	$15,000.00	
Mobilization & Installation	$38,214.60	
Demobilization	$4,138.00	
Fuels	$5,804.91	
Food Costs	$7,387.91	
Misc. Rentals & Supplies	$8,496.00	
		$79,041.42

TRANSPORTATION

Vehicle Rentals	$7,056.68	
Helicopter Charges	$4,130.49	
		$11,187.17

MISC. SUPPLIES & SERVICES $13,750.42

MISC. TRAVEL $5,896.25

Compilation of Data & Final Report

September - November, 1987

K.V. Campbell and Associates Ltd.	$	6,000
Kilborn Ltd.	$	396
D.A. Leishman, B.Sc., 4 days @ $240/day	$	960
Boyd McKean, M.Sc., 32 days @ $160/day	$	5,120
John R. Kerr, P. Eng., 6 days @ $325.00/day	$	1,950
Geomin Computer Services	$	5,285
Photocopying & Reproduction	$	300
Drafting Charges	$	2,925
Secretarial	$	500
TOTAL	$	23,436

Total costs eligible for assessment
work & F.A.M.E. grant .... $ 365,588.10

Other indirect project costs not eligible
for assessment work or F.A.M.E. grant
Project administrative costs
6½ months @ $4,000.00/month $26,000.00

Advance to Mine Contractor
(R.F. Fry & Assoc. Ltd.) $10,500.00
less refund $5,500.00

$31,000.00

9 CERTIFICATES

I, KENNETH VINCENT CAMPBELL, resident of Wells, Province of British Columbia, hereby certify as follows:

1. I am a Consulting Geologist with an office at the corner of Blair and Dawson Avenues, Wells, B.C.

2. I graduated with a degree of Bachelor of Science, Honours Geology, from the University of British Columbia in 1966, a degree of Master of Science, Geology, from the University of Washington in 1969, and a degree of Doctor of Philosophy, Geology, from the University of Washington in 1971.

3. I have practiced my profession for 21 years. I am a Fellow of the Geological Association of Canada (F0078).

4. I have no direct, indirect, or contingent interest in the shares or business in the property of EUREKA RESOURCES, INC. nor do I intend to have any interest.

5. This report, dated November 30, 1987 is based on my geological field mapping, examination of available reports, drill hole results and analyses, between June 21 and September 16, 1987 and subsequent report preparation.

6. Written permission by the author is required to use this report dated November 30, 1987 in any Prospectus or Statement of Facts of EUREKA RESOURCES, INC.

DATED at Wells, Province of British Columbia this 30th day of November, 1987.

K.V. Campbell, Ph.D.
Geologist

I, BOYD EDMUND MACKEAN, resident of Vancouver, Province of British Columbia, hereby certify as follows:

1. I am a self-employed geologist with an office at 4337 West 12th Ave., Vancouver, British Columbia.

2. I graduated with a degree of Bachelor of Science, Geology, McGill University in 1957, and a degree of Master of Science, Geology, McGill University in 1960.

3. I have practiced my profession for 30 years. I am a Fellow of the Geological Association of Canada (F5053).

4. I am a member of good standing with the following professional societies : The Canadian Institute of Mining and Metallurgy (M23283), Mineralogical Association of Canada, and the Association of Exploration Geochemists.

5. This work is based on my geological field work on the Frasergold Property throughout the 1987 work program.

6. I have no interest in shares or business of EUREKA RESOURCES, INC. nor do I intend to have any such interest.

Dated at Vancouver, Province of British Columbia this 30th day of November, 1987.

BE MacKean.

B.E. MacKean, M.Sc.
Geologist

I, DOUGLAS A. LEISHMAN, resident of Kamloops, Province of British Columbia, hereby certify as follows:

1. I am a Consulting Geologist with an office at #74-1750 Summit Drive, Kamloops, B.C.

2. I am a graduate of the Northern Alberta Institute of Technology, Exploration Technology (Minerals Option), 1971, Edmonton, Alberta.

3. I graduated from the Imperial College of Science and Technology, Royal School of Mines, London, England, B.Sc. (Honors) Mining Geology, 1981. I have been actively involved in mineral exploration since 1971.

4. I have no direct, indirect, or contingent interest in the shares or business in the property of EUREKA RESOURCES, INC. nor do I intend to have any interest.

DATED at Vancouver, Province of British Columbia this 30th day of November, 1987.

D.A. Leishman

D.A. Leishman, B.Sc. (Hons).
Geologist