GEOLOGICAL, GEOPHYSICAL, GEOCHEMICAL AND GENERAL EXPLORATION INCLUDING TRENCHING WAS CARRIED OUT ON WEDGE #5, WEDGE #6, WEDGE #7, WEDGE #8, WEDGE #9, WEDGE #10, WEDGE #15, RAS1, RAS2, RAS3, RAS4, LGCS1, LGCS2, LGCS3 and MSL, Claim Sheet 115 I/3, 62° 05N, 137° 11'W, May 23rd, 1986 to August 16th, 1986.

A U R C H E M R E S O U R C E S L T D.

PROCHEM LTD.

Mark Langdon, Geologist,

August 10th, 1986

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ECONOMIC RESEARCH

TABLE OF CONTENTS

List of Claims and Tag Numbers

RAS 1	YA 93138)	
RAS 2	YA 93139)	Owned by R. Schneider,
RAS 3	YA 93140)	Acton, Ontario
RAS 4	YA 93141)	
LGCS 1	YA 95014)	Owned by L. Schneider, Calgary, Alberta
LGCS 2	YA 95015)	Owned by L. Schneider,
LGCS 3	YA 95016)	~~Acton, Ontario~~
Wedge 5	YA 82171)	
Wedge 6	YA 82172)	
Wedge 7	YA 82173)	Owned by G. Dickson,
Wedge 8	YA 82174)	Whitehorse, Yukon
Wedge 9	YA 82175)	
Wedge 10	YA 82176)	
Wedge 15	YA 82181)	
MSL	to be issued - owned by R. Schneider, Acton, Ontario.	

The location of these claims, along with our holdings of leases, are shown on Map #1 at the back of this report.

INTRODUCTION

The Claims and accompanying Leases were explored for Au/Ag mineralization during the summer of 1986 in the the months of May, June, and July.

A base line and grid was established and geological, local geochemical, and EM-16 geophysical surveys were carried out.

Based on results of the geophysics and limited geochemical surveys, a series of trenches and/or pits were dug with a D8-ripper to investigate the favourable responses.

An accurate record was kept on a daily basis to assess what work was done on claims and on the leases, and I believe that our figures contained in this report contain an accurate adjustment of our costs as so to pertain only to that work done on the claims.

REGIONAL GEOLOGY

The claims are located within the eastern half of the Coastal Crystalline Belt, which trends northwesterly across Southwest Yukon. This belt can generally be said to contain lithologies of acidic to intermediate intrusive bodies of post-Triassic age intruding into sedimentary, volcanic and minor intrusive lithologies of late Paleozoic age.

LOCATION AND ACCESS

The claims are located in the valley of Discovery Creek, a tributary of Nansen Creek. Access is from Carmacks west on the Mount Nansen Road. Our claims are approximately 10 km by road past the Mount Nansen Mine Site, or 70 km west of Carmacks.

GENERAL GEOLOGY

Outcrops on the property are rare (1%), and are usually exposed as weathered regolith of large frost-heaved blocks.

The bulk of the property is non-glaciated and overburden consists of relatively in-situ weathered material derived from the underlying bedrock. The depth varies from one to seventy-five feet before some competent bedrock is observed.

Overburden near Discovery Creek is of a glacial-fluvial origin of up to 80 feet thickness. The glacial-fluvial overburden extends up both valley walls to approximately 100-150 ft. above the present creek/water surface.

GENERAL GEOLOGY (continued)

Paleo-creek beds of possible glacial age or earlier are also found down the slopes of the valley walls heading into Discovery Creek.

Topographic relief is in the order of 1000 ft. with slopes of 15-40%. The high elevation combined with the latitude gives a scrub-bush vegetation, with rare growth-stunted trees.

Permafrost was found immediately under the thin moss and organic layers (approx. 6 inches) on both the north and south facing slopes. Past estimation is that the permafrost goes down to about 150 feet.

GEOLOGY - See Map #2

Lithologies

(a) Metavolcanics and Metasediments (Yukon Group) - No outcrops were observed of this unit on our property, but it is believed that a small amount of this lithology may be found on the extreme western edge of our property on the hill south of Discovery Creek. This "Yukon Group" lithology is the oldest rock in the area being Precambrian to Cambrian in age. The volcanics and sediments show a strong metamorphic grade in comparison to all the other lithologies and appear as quartz-hornblende/quartz-biotite gneisses, and amphibolitic phases, (also minor quartzites).

(b) Mount Nansen Group - This group of Jurassic age is composed of andesitic to basaltic flows, and derived volcanoclastics with minor sub-intrusive dioritic plugs.

Our claim group contains this group on the western 25% of our block with a diorite plug being found, to date, only on the south side of Discovery Creek.

(c) Intrusives - Large intrusive bodies of Cretaceous (95 million years) age were then intruded. Our property shows these as granodiorites with a few local variations near contacts and vein structures. The granodiorite in the northeast corner of our claim block has been silicified with a high influx of quartz.

(d) Rhyolitic sub-intrusives - A late stage of the above granodiorites produced rhyolite as sub-intrusive sills and dykes. No extrusive explosive rhyolites have yet been seen. The rhyolites vary in composition and character from fine grained siliceous varieties to qtz-feldspar porphyritic varieties. The qtz-tournaline plugs which somewhat surround our property may be related to the rhyolites (d) or the intrusives (c), as upper level roof pendants or as a separate intermediate or later phase of the two.

The rhyolites have been found in trenching to intrude the diorite of the Mount Nansen Group. It is my belief that the rhyolites occurred both before and during the epithermal event caused by the intrusives. Therefore, some rhyolites will be weakly mineralized and some show a strong hydrothermal alteration. Also, the epithermal vein structures appear to be related to faulted or weak zones (contacts) which would also structurally control rhyolitic intrusions and dykes. Therefore, a spatial relationship may show up between the mineralized veins and rhyolites because of genetically structural reasons.

Vein Lithologies and Characteristics

Our areas of mineralization we have termed our "vein zones" which comprise a great many variable types of veins within a certain zone or width. Individual veins vary slightly in width and composition along strike, but are highly variable with depth. A few of the vein types are :

(a) BLUE MUD - near surface this shows up as a white "mush" composed of clays, quartz, minor sulphides and other alteration products. With depth the vein turns royal blue to baby blue in colour depending on the sulphide content which ranges up to 50%. Also quartz/sulphide veinlets (<4 inches) become common with depth in the more sulphide rich dark blue mud. This unit to a depth of 50 ft. has gone from the white "mush" to a "plastersine-like" substance with quartz/sulphide clasts increasing in abundance. One panned sample of this showed 10 visible flakes of gold, but we have been unable to reproduce this with four other panned samples. A number of the "blue mud" veins occur in our vein zone which vary from inches to 20 ft. in width. Although nearly all the other vein types in our zone show carbonate, the blue mud does not except that some competent calcite veins (with sulphides) up to 10 inches wide have been found within the vein.

We believe at this time, that the blue mud vein(s) are a kaolinized (clay) vein zone representative of being higher up in an epithermal system (discussed later).

(b) Light Blue Quartz/Calcite Vein - this material varies from 1 inch to 15 feet in width and is highly variable along strike and with depth. Mineralization is erratic, but values are increasing with depth as silica content increases and carbonate decreases.

(c) Quartz/Sulphide Veins with variable amounts of Carbonate - these veins first appear as stringers within a zone of rusty weathering up to 60 feet wide. With depth the stringers increase in width. In all cases the greatest increase in width has been on the upper surface or edge of the dipping veins. In our trenches, the upper edge veins have increased up to 3.5 feet in width.

The material here also increases in silica content with depth and decreases in carbonate. Mineralization gives up to 30% sulphides. Rare galena/sphalerite veins are also found although galena is found also as blebs throughout.

(d) Mineralized Granodiorite - These units are highly variable in appearance from different degrees of alteration. Generally, an increase in silica content, a loss of ferromagnesiums and an increase in sulphides of differing degrees is found. Some shearing occurs at times.

Mineralization occurs up to 10% as disseminations, as sulphide stringers and as fracture cleavage coatings.

When alteration becomes very intense, the rock grades into (c) above, (total replacement?).

(e) Galena Veins - veins of galena/quartz/sulphides up to 6 inches wide are found randomly throughout. Some of these have the same strike and dip as the vein system and some are perpendicular to the main veins.

(f) Other - numerous other variations of the above vein types are found throughout. Brecciation is common with some zones showing many stages of re-brecciation. Some of the brecciation is caused by faulting but the majority appears to be of a hydrothermal origin.

Vein Characteristics and Faulting

Our major work to date has concentrated on one vein system on our claim group. Another vein system occurs on the west side of the property at the diorite/andesite contact. A trench on this system on its southern extention off our property shows a quartz vein of a couple of meters in width. Another vein system may lie to the east of our main area of interest also.

Major NNW trenching faults (Weber, Heustis, and Brown McDade) occurred pre/syn. vein emplacement. These faults appear to be the major controlling factors of vein emplacement. In our zone the veins appear to occur dominantly to the west of the fault zones. Many east-west trending faults post-date the major faults and the vein zones causing eastwest offsets. In our area the east-west faults are shown to be "dominantly" up-thrusts and down faults (dip-slip faults) rather than lateral displacement, (minor lateral faulting). Lateral displacement is then found on the surface due mainly to the dip of the veins when dip-slip faulted.

There are many small east-west faults on our property, but two of these play the major role on our vein structure. One of these occurs in Discovery Creek, where the south side has been down-faulted possibly 100 ft. or more. The other occurs just north of our 4H trench with the north side being downfaulted 100ft. or more. The result is in block between these two faults that has been dropped. Geologically in this zone (trenches 8S, 2S, W8, 3S, 11S) we have found the vein structures indicating that they are higher up in the epithermal model. Abundant carbonate, clay alteration (kaolinite), less silica content, very erratic mineralization, sphalerite, galena, the Ag/Au ratio, malachite, realgar, etc. all support this theory.

Our Courtland trench on the north side of Discovery Creek shows a very siliceous vein rock with very little carbonate possibly indicating the vein at a greater depth or within the siliceous cap.

Our trench 4H to the south of the downfaulted block also shows a much greater silica content.

The EM-16 data in interpretation shows the top of the main conductor to be at about a 150 feet of depth to the north of Discovery Creek and south of the other fault while the downfaulted block indicates a depth of 250-300 feet to the top of the conductor. Although this approach is highly interpretive, the data shows a definite strong support for the downfaulted block theory.

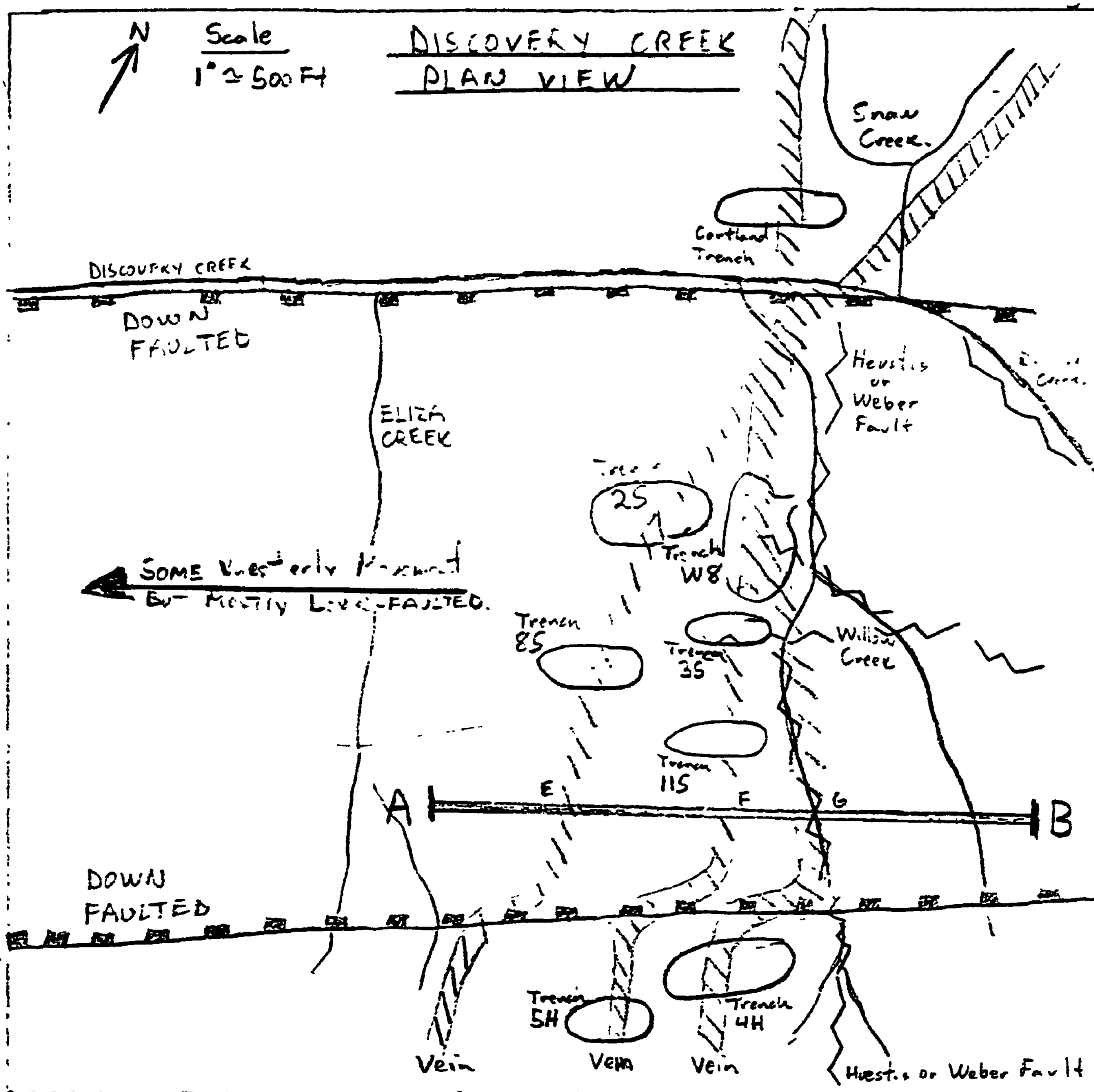
The veins themselves are very continuous along strike with little variation in composition or width. The greatest change takes place with depth as widths vary 75% and composition changes drastically in less than 10 ft. of depth.

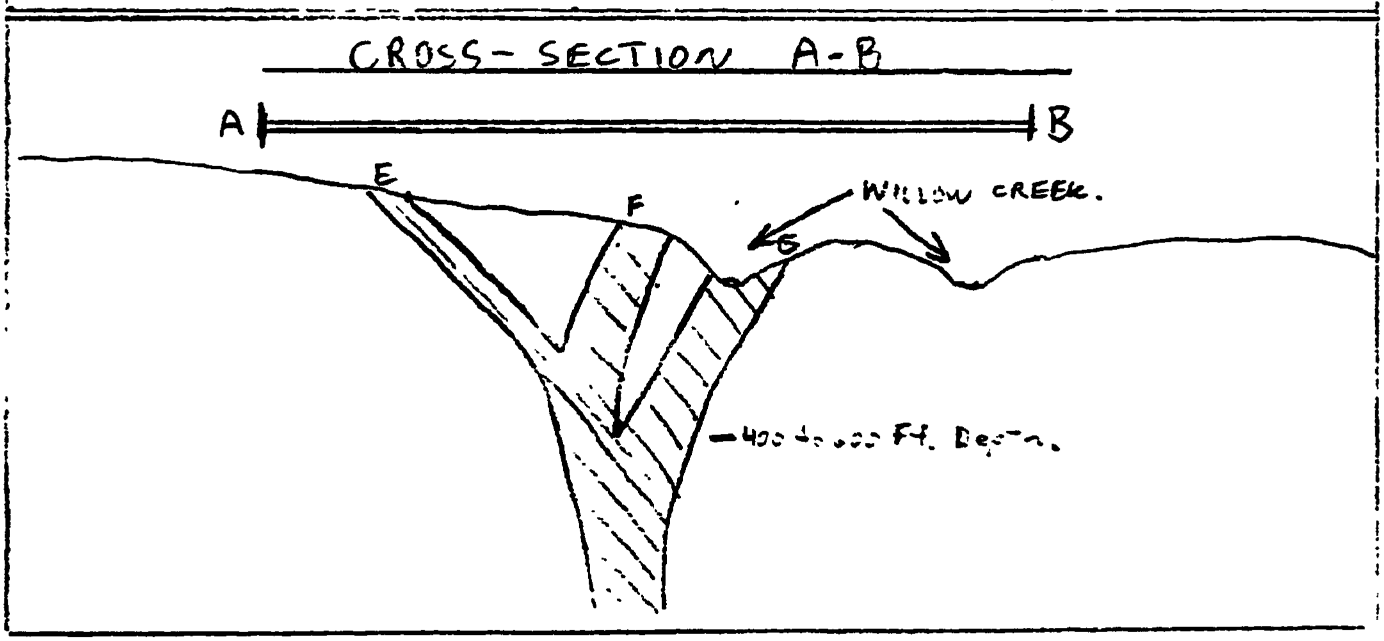
The schematic diagram on the next page shows our main vein system and its general characteristics. Strike and dips were obtained in the trenches to show the cross-sectional view.

The veins on surface appears to split and branch, but this is primarily due to branching in a vertical sense.

Mineralization

Assay values have been sub-economic with erratic higher values of silver and gold. We believe that we are quite high up in the epithermal system, and should expect as much. Silver values tend to increase with galena content. Gold values have shown (from the metallic assay procedure) that about 30% of our gold may be in the form of free gold. Metallurgical testing and panning have supported this. Further metallurgical testing is needed and will be carried out this winter. Arsenic and antimony values tend to show a direct relationship to the silver and gold values, but further work on this is required.


N
Scale
1" = 500 Ft
DISCOVERY CREEK
PLAN VIEW
Snow Creek
Cortland Trench
DISCOVERY CREEK
DOWN FAULTED
Heustis or Weber Fault
ELIZA CREEK
Trench 2S
Trench W8
SOME
Trench 8S
Trench 3S
Willow Creek
Trench 11S
A
E
F
G
B
DOWN FAULTED
Trench 5H
Trench 4H
Vein
Vein
Vein
Huestis or Weber Fault


CROSS-SECTION A-B
A
B
E
F
G
WILLOW CREEK.

Geology V.S. Mineralization

A very important factor determining the vein width has been the type of host rock. On our claims the granodiorite has shown to be by far the best host. In the granodiorite the vein materials replace and alter to increase their width. Pervasive alteration and partial to total replacement of granodiorite by solutions is very common. This results in wide mineralized zones of many variations and gradations. In many cases, vein material can be seen to have relic granodiorite textures. Much of the granodiorite between individual veins in a "vein zone" is very altered and nearly always has small anomalous gold values from secdonary alteration (seritization, kaolination, carbonitization, silicification, chloritization, minor phyllic alteration).

When the veins cut the diorite which is a much harder lithology, the veins become relatively thin (< 10 ft.). The solutions appear to be fault or fracture infillings, and the walls are relatively impermeable. Wall rock alteration is much less pronounced and is limited to less than 10 ft. where gold values are low. Alteration in the wall rock is more of a brecciation with small mineralized veinlets infilling some fractures.

Our western vein system appears to be at the diorite-andesite contact. The andesite is also very siliceous and hard so we expect similar type veins here as we found within the diorite. No trenching has been done on the vein system to date.

Grid

A picketed base line (chained) was set up at a bearing of 330°. Discovery Creek was used as a zero point for distances north and south from this station. Topofil lines were made to the east and west of the base line (60° and 240°) at 100 ft. separations with 50 ft. flagged stations. (See Map #3)

A small grid had been set up in 1985, but most of this was redone as these were carried out with pace and compass with no base line. The primary results were encouraging and indicated better control was needed for more advanced work.

Geophysics

An EM-16 V.L.F. survey was conducted over the grid using Seattle, Washington as the transmitter. The in-phase/out-of-phase data was plotted and can be seen on Map #4 with our interpretation.

The V.L.F. will respond to conductive bodies (i.e. sulphides) that have a N-S vector component for the Seattle, Washington signal. Our exploration is for precious metals which alone will not produce an anomaly. Therefore, although we have pursued the stronger anomalies, the weaker conductors may be significant, but lack the sulphides to show a strong conductor.

Cuttler Maine transmission was attempted for an E-W conductor vector, but the transmission was too weak and gave about a 50% error rate. Therefore, this was discontinued.

The V.L.F. anomalies have shown, from trenching, to produce extraordinary accuracy of vein location down to within 5 ft. The problem in this area is that we have "vein zones" of 100-200 Ft. width consisting of a great number of individual veins. Each vein sets up its own magnetic field creating multi-anomaly additions across the strike so interpretation of the anomaly's size is very difficult. In most cases, we found a vein of <15 ft. of width on the anomaly, but larger veins to the east that were sometimes 150 ft. away. This obscured using models to determine the dip of the veins.

Some highly interpretive data such as the depth to the top of the conductor was calculated from the V.L.F. data and can be seen on Map #5. Basically, we found that in the centre section (the dropped block area) the depth was about 250 ft. while to the north and south of this area the depth was 150 ft.

Geochemical Data

Over a number of more promising V.L.F. conductors, soil geochemical samples were taken of the B horizon. In most cases these soils were assayed for Au, Ag, Zn, Pb, As. Map #6 shows the results of these surveys.

A soil survey over the entire grid would be very beneficial to our programme but due to the permafrost nature of the ground, this is very slow and difficult work.

The soils were taken with a heavy pick to break the frozen soil and ice. The best methodology is to do a line by clearing the moss and organic layers off a 2 ft. diameter circle leaving these to thaw for a few days before taking the samples. This would be the best method for doing a more extensive soil programme to get consistent, good samples.

Assay Results & Procedures

The soils as stated above, were geochemically assayed usually for Au, Ag, Pb, Zn, As. (sometimes just for Au, Ag) Rock samples were assayed by a "metallic" procedure because of the free gold content that was found to be contained in the veins. In this procedure a 250 gram sample is pulverized and meshed at -150. The free gold when pulverized smears so will not go through the -150 mesh. Therefore, a gold/silver assay is run on all the +150 mesh material and a 30 gram sample of the -150 mesh material. The results are weight averaged to give the final assay result. The -150 mesh sample was then geochemically assayed for Zn, Pb, and As and in some cases Sb.

Rock samples were usually in the 10 lb. range taken over anumber of feet. This whole sample was crushed to 10 mesh and a 300 gram sample was riffled off for assay use. The rest of the sample was shipped to our office for use in metallurgical work to be done during the winter of 1986/1987. This work will be contained in a later assessment report.

Assay results are shown on individual trench maps and are listed in Appendix I.

REPORT ON TRENCHES

General Procedure

Once a V.L.F. anomaly and coincident soil geochem anomaly was established, the best sites for trenches were formed. Detailed soil geochems were then taken at 10 or 25 feet spacings over the trench area. Once the trench was completed soil geochem profiles were also conducted at this spacing at a 15 foot depth and usually on the floor of the trench (depending on the amount of competent rock exposed on the trench floor).

The trenches were then geologically mapped and sampled. Sampling usually consisted of chip samples over footages of bedrock with geochem samples taken where overburden or disintergrated rock still persisted.

The EM-16 (V.L.F.) was re-run at 25 ft. spacings over the trenches to tie in to the geology and assay results.

In some cases, some rock assays were taken at different depths as the trenching continued. Individual maps with assays and V.L.F. data were then formed for each trench.

Below is a brief description of each of the trenches. A trench location map is shown on page 7.

(1) Trench 8S - See Map #8

This vein structure is found within the diorite. It has a strike and dip of 340°/45E. The vein material here is of two kinds; (1) a siliceous and carbonate rich breccia (multi-stage brecciation and recrystallation) carrying 10% sulphides; (2) thin (<10") bands of massive pyritic sulphides.

The vein width varied with depth changing from 3 to 11 ft. With alteration halos the zone zone was about 25 ft. wide.

The diorite near the vein was mangesite rich and at times contained abundant magnetite.

To the east of the vein a fine grained rhyolite intrusion is found (at least 26 ft. wide) that contained minor sulphides and low anomalous gold values.

2) Trench 2S - See Map #9

This trench is on the same structure as trench 8S, but here, is at the granodiorite/diorite contact.

The strike and dip of the veins was 350°/60° E and of the contact 330°/80° E.

This trench was not dug to a great enough depth and most samples are of disintergrated bedrock (overburden).

Small veins and pods of mineralization occur at the diorite contact, but the bulk of the veins showing were within the granodiorite.

Two or three veins of "blue mud" (see earlier) were found and one vein of a rusty sulphide mixture was found. The latter vein was found to "pod" going from 1 ft. to about 15-20 ft. wide. Numerous pods of this type were found within the argillized granodiorite.

(3) Trench W8 - See Map #10

The vein zone here is quite large (150-200 ft. wide) and is within granodiorite. The vein(s) have a strike and dip of 330°/75-80°W.

This trench is our largest and deepest, being about 400ft. long, 100ft. wide and about 40ft. deep at its centre.

The main trench is shown on Map #10 but an extension was dug to the northwest for about 175ft. This extension did not reach bedrock in this area. An important feature seen here was a paleo-stream bed of semi-rounded fragments of diorite/granodiorite/quartz/minor clasts of other lithologies that had extensive coatings of mangesite/ilmenite. Cross-bedding was common giving a flow direction to the north. The paleo bed showed to be at least 4 ft. thick and open to depth. Visible gold flakes can be panned out of this unit (varies from none to 50 flakes).

The Wedge 8 trench has been dug at the intersection of two vein branches. The northwest half of the trench represents the veins also found in trenches 3S, 11S and 5H. The southeast half of the trench represents the veins which run up Willow Creek and are found at the 4H trench.

A mixture of many types of veins can be found in the W8 trench.

(4) Trench 3S - See Map #11

This trench is the southern extension of the western half of trench W8.

The strike and dip of the veins in 335°/75°W. The veins here basically are of two types; (1) a six foot wide blue mud vein; (2) a 30 foot wide vein of small sulphide/quartz veins.

(5) Trench 11S - See Map #12

This trench is an extension to the south of the same "vein zone" as in trench 3S. The strike and dips are the same but the veins have widened to 10ft. for the blue mud and 50-60 feet for the quartz/sulphide vein.

(6) Trench 5H - See Map #13

This trench was only dug to a 2 foot depth. Numerous diorite float was encountered so the trench ws abandoned due to the host rock type. Soil geochems were taken across the 125' long trench surface, and we are presently awaiting the results.

(7) Trench 4H - See Map #14

This trench is the southerly extension of the east half of trench W8.

On the western edge is the diorite/granodiorite contact. From the contact a number of veins occur for about 60 feet to the east, then a 100 foot section of granodiorite, and then another "vein zone" for about another 150 feet.

The strike of the veins here vary (see map) but the dip was fairly consistent at 80°W.

(8) Courtland Trench - See Map #15

This trench is the northern extension of the western half of trench W8. Time restraints did not permit this trench to be dug to the required depth. Two or three veins were encountered. The most significant being a light blue-grey quartz vein, with about 3% sulphides, and a 10 foot width. The material appeared only partly oxidized. Note: others have previously stated that material on south facing slopes is oxidized to a 150ft. depth.

CONCLUSIONS AND RECOMMENDATIONS

Although most assay results show a sub-economic grade, the large widths of "vein zones" of anomalous precious metal values is very signficant. It was found in all trenches that the deeper we dug the better the assay results. This along with geological studies of where we are located vertically in epithermal system models, all indicate a very promising economic potential at depth.

Therefore, it is highly recommended that further work should be done on the project.

Further trenching at this time would not be cost effective for the information it would bring. The western VLF anomaly on the diorite/andesite contact is worthy of investigation, but the steep topography here inhibits cost-effective trenching.

It is therefore recommended that further work should be in the form of diamond drill holes of a HQ or NQ size. Drill sites should be formed when all our information and data has returned and has been fully evaluated.

Mark Langdon

Mark Langdon
Geological Projects Manager,
AURCHEM RESOURCES LTD.

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Bostock (1936) Carmacks District, Yukon, Geological Survey, Canada. Mem.169.

FOOT-NOTE FOR EXPENDITURES

An accurate record was kept of the work done on claims or leases. The following gives the percentage of total work that was done on the claims versus the leases. From this the expenditures on the following pages have been adjusted to contain only that work that was done on the claim groups.

Assays —— 60% on claims

Personnel ——— 75% on claims

D8-Cat Ripper Rental —— 75%

Mark Langdon

Mark Langdon

LIST OF EXPENDITURES

(a) Personnel Expenditures

Mark Langdon,	Geological Project Manager, 511 Hayward Crescent, Milton, Ontario. Field Work; Geologic Mapping and Sampling, EM-16 survey, geochemical surveys, supervision. 56 Days AT $150/DAY - $8,400 x 75%	$ 6,300.00
	Compiling Data and Report and Map Forming, organization 45 DAYS AT $150/DAY - $6,750 x 75%	5,062.50
Mike Anderson,	Contract Geologist, St. Catherines, Ontario. Field & Office Work; $600/week for 12.4 weeks - $7,460.72 x 75%	5,595.54
Lee Schneider,	Geological Assistant, Calgary, Alberta. Field Work; EM-16, soil sampling, Claim Staking and line cutt. 9 Days at $100/day - $900 x 75%	675.00
Rob Schneider,	Geological Assistant, Acton, Ontario. Field Work; EM-16, geochemical sampling, drafting, grid formation, general. 32 Days at $450/week - $2,057.14 x 75%...	1,542.86
	Ofice Work; rock sorting, map drafting, compilation. 2 weeks at $321/week - $642.00 x 75%	481.50
John Schneider,	President, Aurchem Resources, Chem. Eng., Metallurgist. Field; trenching, compilation, supervision, sampling. 20 days at $250/day - $2,500 x 75%	1,875.00
	Office; Data Manipulation and Compilation. 15 days at $200/day - $3,000 x 75%	2,250.00
Secretarial Office Costs - $500 x 75%		375.00
(A) Total Personnel Expenditures		$24,157.40

List of Expenditures (continued)

(b) Rental Costs

1 Trailer at $400/month for 2-1/2 months Hidden Valley Outfitters Ltd., Carmacks, Yukon	$ 1,000.00
1 D8-Cat with Ripper $130/hour for 400.5 hours Hidden Valley Outfitters Ltd., Carmacks, Yukon $52,065.02 x 75%	39,048.77
1 Surburban at $1,650/month plus mileage for 2.5 months Norcan Lesing, Whitehorse, Yukon	4,384.18
1 Bronco for 5 days Tilden TruckRental, Whitehorse, Yukon	507.60
1 EM-16 VLF for 13 weeks at $145/week Geonics Ltd., Mississauga, Ontario	1,885.00
(B) Total Rental Expenditures	$46,825.55

(c) Miscellaneous Expenses

Groceries, Meals, Hotels, Propane, Gas for 2.5 months for staff	$ 6,260.50

(d) Material & Equipment Transport Costs

Field Equipment from Toronto to Whitehorse Air Canada Cargo - May 22, 1986	$ 104.34
Rock Samples for Metallurgical Testing from Whitehorse to Toronto; Yukon Freight Lines Ltd.	669.79
From Carmacks to Toronto; Yukon Freight Lines Ltd.	600.00
(D) Total of material and equipment transport costs	$ 1,374.13

(e) Cost of Flights to Yukon

Cost of all flights by staff to get to and from the Yukon	$ 9,497.02

(f) Assay Expenditures

All assays prepared in Whitehorse and sent to Ottawa lab for analysis.
Bondar-Clegg & Co. Ltd., Whitehosre, Yukon.

List of Expenditures (continued)

Geochemical Analysis of Soils :	
114 samples for Au, Ag at $9.40 each	$ 1,071.60
18 samples for Au, Ag, Pb, Zn, As, Sb at $19.90 each	358.20
47 samples for Au, Ag, Pb, Zn, As at $15.15 each	712.05
Assays of Rocks :	
3 samples for metallic Au, Ag, at $26.75 each	80.25
30 samples for metallic Au, Ag, Pb, Zn, Sb at $33.50 each	1,005.00
152 samples for metallic Au, Ag, As, Zn, Pb at $32.50 each	4,940.00
5 samples for Au, Ag, at $15.25 each	76.25
(F) Total - assay expenditures	$ 8,243.35
CLAIMS TOTAL x 60%	$ 4,946.01

(g) Equipment and Supplies Bought

Stationary, flagging, sample bags, hammers, tents, claim posts, pickets, back-packs, compasses, etc.	$ 2,090.74

SUMMARY OF EXPENDITURES

(a) Personnel Expenditures	$24,157.40
(b) Rental Costs.............................	46,825.55
(c) Miscellaneous Expenses	6,260.50
(d) Material and Equipment Transport Costs ...	1,374.13
(e) Costs of Flights to Yukon	9,497.02
(f) Assay Expenditures	4,946.01
(g) Equipment and Supplies Bought	2,090.74
GRAND TOTAL	$95,151.35

For all expenditures receipts are available on request.

Mark Langdon
Geological Projects Manager
AURCHEM RESOURCES LTD.

August 7th, 1986

APPENDIX I

LIST OF ROCK AND SOIL

ASSAYS FOR 1986

BONDAR-CLEGG

Certificate of Analysis

Report: 416-2689

Date: August 6, 1986

Client: Prochem

Submitted by: BCC Whitehorse

Samples:	Oz/ton Au +150	Oz/ton Ag +150	grams weight +150	Oz/ton Au -150	Oz/ton Ag -150	grams weight -150	Oz/ton Au weighted average	Oz/ton Ag weighted average
U4R 86-001	0.004	0.05	12.10	0.006	0.18	173.7	0.006	0.17
U4R 86-002	0.013	0.46	6.95	0.013	0.35	185.6	0.013	0.35
U4R 86-003	0.041	0.35	10.83	0.025	0.35	171.6	0.026	0.32
U4R 86-004	0.005	0.07	13.17	0.004	0.11	141.4	0.004	0.11
U4R 86-005	<0.001	<0.01	12.90	<0.001	0.01	136.9	<0.001	0.01
H486L-151	<0.001	<0.01	10.59	0.002	0.06	179.4	0.002	0.06
11S86-153	0.002	0.06	11.88	0.002	0.06	189.9	0.002	0.06
11S86-154	0.072	0.52	9.93	0.041	0.99	221.6	0.042	0.97
11S86-157	0.049	1.40	13.16	0.090	4.03	244.9	0.088	3.90
11S86-160	0.003	0.30	12.73	0.007	0.28	118.5	0.007	0.28
1[illegible]86-165	0.004	0.24	13.17	0.006	0.25	131.6	0.006	0.25
1[illegible]6-167	0.001	<0.01	12.69	<0.001	<0.01	92.3	<0.001	<0.01

P. Raulton

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Telex: 053-3233

BONDAR-CLEGG

Certificate of Analysis

Report: 416-2431

Date: August 5, 1986

Client: Prochem Ltd.

Submitted by: M. Langdon

SAMPLES:	Oz/ton Au +150	Oz/ton Ag +150	grams weight +150	Oz/ton Au -150	Oz/ton Ag -150	grams weight -150	Oz/ton Au weighted average	Oz/ton Ag weigh average
3586-L75	0.372	0.78	12.00	0.036	3.72	245	0.052	3.58
F3686-001	0.076	0.55	17.38	0.038	0.44	235	0.041	0.45
F3686-002	1.117	0.44	3.29	0.169	7.38	220	0.183	7.28
R2R86-018	0.004	0.03	10.24	0.004	0.09	212	0.004	0.09
R2R86-019	<0.001	0.06	9.79	<0.001	<0.01	252	<0.001	<0.01
R2R86-020	0.001	0.03	14.93	0.001	<0.01	187	0.001	<0.01
R2R86-021	0.002	0.02	12.03	<0.001	<0.01	216	<0.001	<0.01
R2R86-024	0.002	0.05	12.35	<0.001	<0.01	185	<0.01	<0.01
R2R86-056	0.002	0.11	13.63	0.002	0.09	106	0.002	0.09
W886-L60	0.019	0.20	13.40	0.016	<0.01	116	0.016	0.02
W886-L61	0.055	0.46	17.11	0.063	1.75	140	0.062	1.61
W886-L62	0.012	0.13	6.54	0.007	0.19	130	0.007	0.19
W886-L63	0.940	0.62	18.46	0.108	3.08	191	0.181	2.84
W886-L64	0.212	1.60	6.20	0.054	8.76	203	0.059	8.53
W886-L65	0.010	0.74	9.45	0.012	4.35	96	0.012	4.03
W886-L66	0.065	0.65	17.93	0.047	2.05	225	0.048	1.95
W886-L68	0.025	0.37	2.34	0.017	0.37	166	0.017	0.37
W886-L69	0.090	0.70	1.25	0.071	0.13	172	0.071	0.13
W886-L70	0.071	0.64	4.05	0.067	0.35	90	0.067	0.36
386-L71	0.005	0.13	13.55	0.002	0.20	160	0.002	0.20

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C. Rodan

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BONDAR-CLEGG

Certificate of Analysis

Report: 516-2124

Date: August 5, 1986

Client: Prochem

Submitted by: J. Schneider

SAMPLES:	ppm Au +150	ppm Ag +150	grams weight +150	ppm Au -150	ppm Ag -150	grams weight -150	ppm Au weighted average	ppm Ag weighted average
TW 8R-L1	4.78	48.6	6.17	1.55	330.1	257.4	1.63	323.5
W 8S86-1	«0.01	0.5	19.35	0.03	2.7	179.4	0.03	2.5

C. Doucet

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BONDAR-CLEGG

Certificate of Analysis

Report: 416-2431

Date: August 5, 1986

Client: Prochem Ltd.

Submitted by: M. Langdon

SAMPLES:	Oz/ton Au +150	Oz/ton Ag +150	grams weight +150	Oz/ton Au -150	Oz/ton Ag -150	grams weight -150	Oz/ton Au weighted average	Oz/ton Ag weighted average
W886-L72	0.002	0.12	12.27	0.004	0.05	129	0.004	0.05
W886-L73	1.040	0.43	13.46	0.049	0.27	155	0.128	0.28
~~W886-L74~~	~~0.092~~	~~2.04~~	~~18.19~~	~~0.058~~	~~0.95~~	~~170~~	~~0.061~~	~~1.06~~

C. Paulson

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BONDAR-CLEGG

Certificate of Analysis

rt: 416-2742

Date: August 6, 1986

nt: Prochem

Submitted by: BCC Whitehorse

les:	Oz/ton Au +150	Oz/ton Ag +150	grams weight +150	Oz/ton Au -150	Oz/ton Ag -150	grams weight -150	Oz/ton Au weighted average	Oz/ton Ag weighted average
-001	0.002	<0.01	9.00	0.003	0.06	139.3	0.003	0.06
-002	<0.001	<0.01	12.95	<0.001	<0.01	103.2	<0.001	<0.01
-015	0.002	0.03	11.42	0.002	0.10	122.8	0.002	0.10
-016	<0.001	<0.01	8.73	<0.001	<0.01	116.3	<0.001	<0.01
-017	<0.001	<0.01	8.99	<0.001	0.11	129.3	<0.001	0.11
-018	<0.001	<0.01	12.37	0.001	0.04	146.0	0.001	0.04
-019	<0.001	<0.01	11.30	<0.001	<0.01	147.0	<0.001	<0.01
-020	0.064	0.74	13.74	0.065	1.21	186.4	0.065	1.18
-021	<0.001	<0.01	9.50	<0.001	<0.01	78.4	<0.001	<0.01
6-004	0.004	<0.01	9.35	0.005	0.05	124.1	0.005	0.05
6-011	0.009	<0.01	3.40	0.003	0.07	186.3	0.003	0.07
6-(	<0.001	<0.01	6.21	0.009	0.43	176.4	0.009	0.42
-024	0.005	0.29	9.11	0.005	0.80	140.2	0.005	0.77
-025	<0.001	<0.01	7.63	<0.001	0.12	95.4	<0.001	0.12
-026	<0.001	<0.01	11.28	<0.001	0.09	117.0	<0.001	0.09
-028	0.001	<0.01	11.52	0.015	0.26	121.1	0.014	0.24
-036	0.014	0.03	11.33	0.001	0.07	122.5	0.002	0.07
-037	0.015	<0.01	8.65	0.005	0.09	162.7	0.006	0.09
-038	0.036	0.05	5.41	0.006	0.22	165.7	0.007	0.22
-039	0.002	<0.01	9.11	0.008	0.10	152.9	0.008	0.10

C. Naulen

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BONDAR-CLEGG

Geochemical Lab Report

REPORT: 016-2690

PROJECT: 9

SAMPLE NUMBER	ELEMENT UNITS	Zn PPM	Ag PPM	Pb PPM	As PPM	Sb PPM	Au PPB	TestWt gm
060S 050W		56	1.4	31	31	<1	10	10.00
060S 040W		78	1.7	27	28	<1	20	10.00
100S 055W		48	0.6	22	6	2	10	10.00
100S 040W		76	0.8	15	15	5	15	10.00
100S 035W		94	0.8	14	4	<1	26	10.00
000T 080W		66	0.4	23	31	<1	10	10.00
000T 075W		64	0.6	19	15	<1	15	10.00
000T 070W		168	0.6	106	19	<1	<5	10.00
000T 065W		460	3.8	1335	584	6	125	10.00
000T 060W		200	0.7	108	48	1	15	10.00
000T 055W		196	0.4	54	20	<1	20	10.00
000T 050W		98	0.1	42	2	2	5	10.00
000T 045W		110	0.7	66	61	<1	5	10.00
000T 040W		156	1.2	75	45	3	15	10.00
000T 035W		260	0.6	249	88	1	20	10.00
000T 030W		140	1.0	94	54	1	165	10.00
000T 025W		104	0.6	47	31	3	40	10.00
000T 020W		66	0.4	26	13	<1	15	10.00
U4S86001			1.8				140	10.00
U4S86002			0.4				10	10.00
U4S86003			0.3				<5	10.00
U4S86004			0.2				25	10.00
U4S86005			3.5				65	10.00
U4S86006			0.2				10	10.00
U4S86007			6.0				1320	10.00
11S86-152			0.7				10	10.00
11S86-155			8.3				275	10.00
11S86-156			10.2				215	7.50
11S86-158			9.1				565	10.00
11S86-159			4.5				580	10.00
11S86-161			1.0				20	10.00
11S86-162			6.8				335	8.00
11S86-163			2.1				50	10.00
11S86-164			1.8				65	10.00
11S86-166			3.1				130	10.00
11S86-168			15.2				800	10.00

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BONDAR-CLEGG

Geochemical Lab Report

REPORT: 016-2621

PROJECT: 8 PAGE 1

SAMPLE NUMBER	ELEMENT UNITS	Ag PPM	Au PPB	
~~E?S86018~~		~~0.2~~	~~[illegible]~~	
0605-035W		0.6	190	10
0605-045W	40	1.3	15	20
0605-055W	50	2.9	20	
0605-060W		0.5	10	
1005-045W	35 40	0.8	10	50 15
1005-050W		0.3	10	
1005-060W		0.4	15	

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BONDAR-CLEGG

Certificate of Analysis

REPORT: 416-2184

PROJECT: None Page 1

SAMPLE NUMBER	ELEMENT UNITS	Au ppm	Ag ppm	Weight grams	Au ppm	Ag ppm	Weight grams
		+150	+150	+150	-150	-150	-150
R2R-86-005		0.06	12.3	3.25	0.03	1.4	240
R2R-86-007		«0.01	1.1	3.94	0.03	«0.3	240
R2R-86-008		«0.01	1.5	6.69	0.03	3.7	255
R2R-86-014		«0.01	0.7	14.14	0.02	1.4	230
R2R-86-016		0.05	2.3	13.27	0.08	0.3	230
R2R-86-017		0.08	2.0	10.12	0.09	1.0	215
R2R-86-022		0.02	3.8	13.17	0.14	0.3	225
R2R-86-023		«0.01	«0.8	11.97	0.02	0.3	230
R2R-86-025		0.04	«1.2	8.31	0.28	7.5	235
R2R-86-035		«0.01	«0.8	12.11	0.05	«0.3	280
R2R-86-036		«0.01	1.4	14.73	0.01	2.1	250
R2R-86-037		«0.01	«1.0	10.47	0.03	0.7	250
R2R-86-038		0.04	2.7	14.55	0.08	4.1	265
R2R-86-039		0.38	0.6	16.93	0.08	2.1	215
R2R-86-041		«0.01	«0.9	10.72	0.02	0.3	230
R2R-86-042		«0.01	«0.6	15.65	0.03	«0.3	270
R2R-86-047		0.35	«0.9	11.00	0.44	1.4	290
R2R-86-049		«0.01	2.4	8.21	0.02	2.1	250
R2R-86-050		0.05	«1.7	5.83	0.18	5.8	235
R2R-86-051		0.01	1.1	9.18	«0.01	0.7	250
R2R-86-057		79.64	151.2	4.96	8.49	84.0	265
R2R-86-058		0.13	«2.5	3.97	0.04	«0.3	245

SAMPLE NUMBER	ELEMENT UNITS	Au ppm weighted average	Ag ppm weighted average	SAMPLE NUMBER	ELEMENT UNITS	Au ppm weighted average	Ag ppm weighted average
R2R-86-005		0.03	1.5	R2R-86-037		0.03	0.7
R2R-86-007		0.03	«0.3	R2R-86-038		0.08	4.0
R2R-86-008		0.03	3.7	R2R-86-039		0.10	2.0
R2R-86-014		0.02	1.4	R2R-86-041		0.02	0.3
R2R-86-016		0.08	0.4	R2R-86-042		0.03	«0.3
R2R-86-017		0.09	1.0	R2R-86-047		0.44	1.4
R2R-86-022		0.14	0.5	R2R-86-049		0.02	2.1
R2R-86-023		0.02	0.3	R2R-86-050		0.18	5.8
R2R-86-025		0.27	7.5	R2R-86-051		«0.01	0.7
R2R-86-035		0.05	«0.3	R2R-86-057		9.80	85.2
R2R-86-036		0.01	2.1	R2R-86-058		0.04	«0.3

NOTE: « means less than

?. Kantere

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BONDAR-CLEGG

Geochemical Lab Report

Wedge 8

REPORT: 016-2184

PROJECT: NONE

SAMPLE NUMBER	ELEMENT UNITS	Zn PPM	Pb PPM	As PPM
46-129-R2R86-005		560	31	72
46-129-R2R86-007		300	40	37
46-129-R2R86-008		123	13	29
46-129-R2R86-014		62	22	17
46-129-R2R86-016		76	24	34
46-129-R2R86-017		98	32	64
46-129-R2R86-022		178	104	112
46-129-R2R86-023		285	17	65
46-129-R2R86-025		230	275	$\boxed{1216}$ ✓
46-129-R2R86-035		40	13	9
46-129-R2R86-036		45	20	15
46-129-R2R86-037		90	32	68
46-129-R2R86-038		240	120	266
46-129-R2R86-039		315	110	85
46-129-R2R86-041		95	15	62
46-129-R2R86-042		116	88	40
46-129-R2R86-047		190	200	$\boxed{>2000}$ ✓
46-129-R2R86-049		198	95	90
46-129-R2R86-050		1750	1150	[illegible] ✓
46-129-R2R86-051		255	95	408
46-129-R2R86-057		3000	1200	$\boxed{>2000}$ ✓
46-129-R2R86-058		83	28	71

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BONDAR-CLEGG

Geochemical Lab Report

REPORT: 016-2349

PROJECT: 6 PAGE 1

SAMPLE NUMBER	ELEMENT UNITS	Zn PPM	Pb PPM	As PPM	SAMPLE NUMBER	ELEMENT UNITS	Zn PPM	Pb PPM	As PPM
R2R86001		1040	>20000	712	R2R86071		840	118	144
R2R86002		800	110	24					
R2R86003		420	1716	172					
R2R86004		3000	366	161					
R2R86006		180	54	28					
R2R86009		94	35	35					
R2R86010		78	23	22					
R2R86011		220	146	140					
R2R86012		140	29	25					
R2R86013		116	40	98					
R2R86015		144	80	151					
R2R86026		520	8	115					
R2R86027		1020	41	51					
R2R86028		570	17	28					
R2R86029		230	10	17					
R2R86030		340	969	672					
R2R86031		1700	439	151					
R2R86033		290	93	27					
R2R86034		78	28	15					
R2R86040		100	18	103					
R2R86043		76	23	65					
R2R86044		420	357	520					
R2R86045		126	30	45					
R2R86046		220	322	1104 ✓					
R2R86048		540	228	182					
R2R86052		480	424	760					
R2R86053		350	410	608					
R2R86054		240	185	193					
R2R86055		100	27	27					
R2R86060		230	18	37					
R2R86061		176	19	31					
R2R86062		240	24	38					
R2R86063		280	25	23					
R2R86064		360	72	33					
R2R86065		420	69	34					
R2R86066		560	87	41					
R2R86067		590	154	336					
R2R86068		820	206	264					
R2R86069		580	96	107					
R2R86070		1100	49	61					

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BONDAR-CLEGG

Certificate
of Analysis

Wedge 8

Report: 416-2349

Date: July 10,1986

Prochem

Samples:	weighted average value g Tot.Wt.	ppm Au	ppm Ag
32R86001	231.50	0.47	88.0
32R86002	258.15	0.02	<0.03
32R86003	205.51	0.21	27.8
32R86004	131.37	0.07	3.7
32R86006	231.45	<0.01	3.7
32R86009	209.14	<0.01	0.8
32R86010	223.16	0.02	14.4
32R86011	210.93	0.24	3.1
32R86012	237.55	0.06	1.6
2R86013	232.93	0.03	0.7
2R86015	252.15	0.03	1.3
2R86026	276.39	0.23	5.4
2R86027	243.51	0.07	3.7
2R86028	219.57	<0.01	1.3
2R86029	261.31	0.22	3.7
2R86030	188.95	0.21	23.1
2R86031	221.96	0.02	4.9
2R86033	218.49	0.02	2.0
2R86034	172.81	0.01	<0.3
2R86040	171.04	0.01	1.4
2R86043	212.89	0.09	3.1
2R86044	223.80	0.08	3.7
2R86046	164.88	0.24	5.0
2R86048	193.39	0.06	1.3
2R86045	222.83	0.01	0.9
2R86052	208.35	0.08	4.7
2R86053	191.76	0.03	3.6
2R86054	234.55	0.01	2.3
2R86055	246.73	0.01	<0.3
2R86060	218.41	0.02	1.1
R86061	210.56	0.03	1.1
R86062	232.38	0.02	0.9
R86063	225.20	0.01	0.9
R 64	246.89	<0.01	0.9
R86065	220.29	<0.01	3.0

[illegible signature]

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BONDAR-CLEGG

Certificate of Analysis

eport: 416-2349

Date: July 10,1986

rochem

amples:	g +150 weight	ppm +150 Au	ppm +150 Ag	g -150 wt	ppm -150 Au	ppm -150 Ag
2R86066	8.05	«0.01	1.2	244.13	2.4	«0.01
2R86067	16.83	0.08	«0.3	207.39	«0.3	0.08
2R86068	17.86	0.12	3.9	201.01	0.7	0.12
2R86069	8.15	0.04	«0.3	207.41	7.9	0.04
2R86070	16.72	0.01	«0.3	214.71	«0.3	«0.01
2R86071	11.70	0.01	2.6	195.05	«0.3	0.01

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BONDAR-CLEGG

Certificate of Analysis

ort: 416-2349

Wedge 8

Date: July 10,1986

chem

This should be confirmed

ples:	g +150 weight	ppm +150 Au	ppm +150 Ag	g -150 wt	ppm -150 Ag	ppm -150 ~~Ag~~ Au
86001	1.31	8.55	190.0	230.19	87.4	0.43
86002	8.79	<0.01	4.6	249.36	<0.3	0.02
86003	2.60	0.01	250.5	202.91	25.0	0.21
86004	10.10	0.03	13.9	121.27	3.8	0.07
86006	15.74	<0.01	8.3	215.71	3.4	<0.01
86009	12.33	<0.01	2.4	196.81	0.7	<0.01
86010	8.23	<0.01	6.1	214.93	14.7	0.02
86011	8.75	0.42	2.3	202.18	3.1	0.23
86012	7.96	<0.01	8.8	229.59	1.4	0.06
86013	5.38	<0.01	18.6	227.55	0.3	0.03
86015	11.98	0.30	<0.3	240.17	1.4	0.02
8[illegible]	11.40	<0.01	11.4	264.99	5.1	0.24
86027	12.90	0.94	8.5	230.61	3.4	0.02
86028	12.63	<0.01	0.8	206.94	1.4	0.01
86029	14.03	0.01	9.3	247.28	3.4	0.23
86030	7.13	0.06	88.4	181.82	20.6	0.22
86031	6.52	0.03	16.9	215.44	4.5	0.02
86033	4.14	<0.01	<0.3	214.35	2.1	0.02
86034	13.87	0.04	<0.3	158.94	<0.3	<0.01
86040	13.56	0.02	1.5	157.48	1.4	<0.01
86043	0.24	12.71	583.3	212.65	2.4	0.08
86044	12.34	0.09	8.9	211.46	3.4	0.02
86045	9.29	0.03	<0.3	213.54	1.0	<0.01
86046	15.43	0.14	<0.3	149.45	5.5	0.25
86048	8.73	0.10	<0.3	184.66	1.4	0.06
86052	3.86	0.06	2.6	204.49	4.8	0.08
86053	7.23	0.03	<0.3	184.53	3.8	0.03
86054	6.25	0.04	<0.3	228.30	2.4	0.01
86055	12.49	0.02	4.0	205.01	1.0	0.02
86060	13.40	0.03	2.2	205.01	1.0	0.02
86061	15.98	0.02	1.9	194.58	1.0	0.03
86062	13.02	0.02	4.6	219.36	0.7	0.02
8[illegible]3	12.56	0.03	6.4	212.64	<0.3	<0.01
8[illegible]4	0.09	<0.01	17.6	237.80	0.3	0.01
86065	17.66	<0.01	1.7	202.63	3.1	<0.01

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Certificate of Analysis

Wedge 8

port: 416-2349

Date: July 10,1986

ochem

	weighted average value		
	g	ppm	ppm
nples:	Tot.Wt.	Au	Ag
86066	252.18	«0.01	2.3
86067	224.22	0.08	«0.3
86068	218.87	0.12	1.0
86069	215.56	0.04	7.6
86070	231.43	0.01	«0.3
86071	206.75	0.01	«0.3

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Geochemical Lab Report

REPORT: 016-2430

PROJECT: 7 PAGE 1

SAMPLE NUMBER	ELEMENT UNITS	Ag PPM	Au PPB	Au oz/t. Eq.
F3S-86001		4.5	160	
F3S-86002		7.1	140	
F3S-86003		6.2	105	
F3S-86004		5.9	690	
F3S-86005		26.5	2790	.09
F3S-86006		13.8	1140	.036
F3S-86007		9.5	390	
F3S-86008		0.3	30	
F3S-86009		0.5	20	
F3S-86010		53.5	2990	.10
F3S-86011		47.5	3790	.126
F3S-86012		16.6	315	
F3S-86013		<0.1	5	
F3S-86014		6.1	370	
F3S-86015		>100.0	>20000	>.66
F3S-86016		18.5	740	
F3S-86017		1.4	60	
PSPC-1		0.6	2630	
PSPC-2		1.3	7575	.25
PSPC-3		0.1	2270	
PSPC-4		0.4	50	
W8PC-L67		>100.0	3890	.13
080N-025E		0.9	<5	
080N-030E		0.1	15	
080N-035E		1.3	50	
080N-040E		2.2	50	
080N-045E		1.8	165	
080N-050E		0.2	45	
120N-105E		1.1	40	
120N-110E		0.7	25	
120N-115E		0.6	30	
120N-120E		0.7	50	
140N-050E		0.2	40	
140N-055E		<0.1	30	
140N-060E		<0.1	25	
140N-065E		0.5	40	
140N-070E		<0.1	25	
140N-075E		<0.1	40	

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Certificate of Analysis

Wedge 8

Report: 416-2431

Date: July 17, 1986

Client: Prochem

Submitted by: M. Langdon

SAMPLES:	Oz/ton Au +150	Oz/ton Ag +150	grams weight +150	Oz/ton Au -150	Oz/ton Ag -150	grams weight -150	Oz/ton Au weighted average	Oz/ton Ag weighted average
3586-L75	0.372	0.78	12.00	0.036	3.72	245	0.052	3.58
F3686-001	0.076	0.55	17.38	0.038	0.44	235	0.041	0.45
F3686-002	1.117	0.44	3.29	0.169	7.38	220	0.183	7.28
R2R86-018	0.004	0.03	10.24	0.004	0.09	212	0.004	0.09
R2R86-019	<0.001	0.06	9.79	<0.001	<0.01	252	<0.001	<0.01
R2R86-021	0.002	0.02	12.03	<0.001	<0.01	216	<0.001	<0.01
R2R86-024	0.002	0.05	12.35	<0.001	<0.01	185	<0.01	<0.01
R2R86-056	0.002	0.11	13.63	0.002	0.09	106	0.002	0.09
W886-L60	0.019	0.20	13.40	0.016	<0.01	116	0.016	0.02
W886-L61	0.055	0.46	17.11	0.063	1.75	140	0.062	1.61
W886-L62	0.012	0.13	6.54	0.007	0.19	130	0.007	0.19
W886-L63	0.940	0.62	18.46	0.108	3.08	191	0.181	2.84
W886-L64	0.212	1.60	6.20	0.054	8.76	203	0.059	8.53
W886-L65	0.010	0.74	9.45	0.012	4.35	96	0.012	4.03
W886-L66	0.065	0.65	17.93	0.047	2.05	225	0.048	1.95
W886-L68	0.025	0.37	2.34	0.017	0.37	166	0.017	0.37
W886-L69	0.090	0.70	1.25	0.071	0.13	172	0.071	0.13
W886-L70	0.071	0.64	4.05	0.067	0.35	90	0.067	.36
W886-L71	0.005	0.13	13.55	0.002	0.20	160	0.002	0.20
W886-L72	0.002	0.12	12.27	0.004	0.05	129	0.004	0.05

Cont'd on next page

P. Moulton

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Certificate of Analysis

Wedge 8

Report: 416-2431

Date: July 17, 1986

Client: Prochem

Submitted by: M. Langdon

SAMPLES:	Oz/ton Au +150	Oz/ton Ag +150	grams weight +150	Oz/ton Au -150	Oz/ton Ag -150	grams weight -150	Oz/ton Au weighted average	Oz/ton Ag weighted average
W886-L73	1.040	0.43	13.46	0.049	0.27	155	0.128	0.28
W886-L74	0.092	2.04	18.19	0.058	0.95	170	0.061	1.06

C. Doulna

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BONDAR-CLEGG

Geochemical Lab Report

REPORT: 016-2431

PROJECT: 7 PAGE 1

SAMPLE NUMBER	ELEMENT UNITS	Zn PPM	Pb PPM	Sb PPM
[illegible]-L75		16000	8640	147
F3886-001		9600	4740	90
F3886-002		3700	>20000	[illegible]74
R2R86-018		330	637	12
R2R86-019		160	118	2
R2R86-020		200	144	2
R2R86-021		130	37	<1
R2R86-024		290	340	11
R2R86-056		300	779	8
W886-L60		2500	1680	31
W886-L61		10000	11960	133
W886-L62		140	254	94
W886-L63		560	7180	92
W886-L64		6000	9280	205
W886-L65		5600	>20000	145
W886-L66		7600	5990	95
W886-L68		560	1340	105
W886-L69		1450	1210	70
W886-L70		230	476	54
W886-L71		310	56	1
W886-L72		65	37	10
W886-L73		590	849	86
W886-L74		340	1200	64

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BONDAR-CLEGG

Geochemical Lab Report

REPORT: 116-2431

PROJECT: 7

SAMPLE NUMBER	ELEMENT UNITS	As PPM
3S86-L75		>2000
F3S86-001		>2000
F3S86-002		>2000
R2F86-018		856
R2R86-019		101
R2R86-020		134
R2R86-021		45
R2R86-024		96
R2R86-056		608
W886-L60		>2000
W886-L61		>2000
W886-L62		396
W886-L63		>2000
W886-L64		>2000
W886-L65		768
W886-L66		>2000
W886-L68		616
W886-L69		>2000
W886-L70		>2000
W886-L71		171
W886-L72		103
W886-L73		>2000
W886-L74		>2000

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BONDAR-CLEGG

Certificate of Analysis

Report: 416-2620

Date: July 31, 1986

Client: Prochem

Submitted by: BCC Whitehorse

Samples:	oz/ton Au +150	oz/ton Ag +150	grams weight +150	oz/ton Au -150	oz/ton Ag -150	grams weight -150	oz/ton Au wt.ave.	oz/ton Ag wt.ave.
R2R86-82	0.005	<0.01	7.56	0.002	0.01	142.5	0.002	0.01
R2R86-83	0.002	0.05	6.24	0.004	0.01	123.0	0.004	0.01
R2R86-84	<0.001	0.02	6.24	0.004	0.01	145.8	0.006	0.20
R2R86-85	0.033	0.21	9.69	0.035	0.71	216.6	0.035	0.69
R2R86-86	0.307	0.57	9.74	0.057	0.46	191.8	0.069	0.47
R2R86-87	0.002	0.21	9.85	<0.001	0.01	203.3	<0.001	<0.01
R2R86-88	0.003	<0.01	4.83	<0.001	0.36	202.4	<0.001	0.35
R2R86-89	0.893	0.41	6.37	0.056	<0.01	234.9	0.078	0.01
R2R86-90	0.026	0.56	9.36	0.006	0.29	204.7	0.007	0.30
R2R86-91	0.001	0.08	7.22	0.003	0.52	164.9	0.003	0.50
R2R86-100	0.014	<0.01	6.18	0.003	0.21	178.9	0.003	0.20
R2R86-101	0.003	0.05	7.46	<0.001	<0.01	163.2	<0.001	<0.01
R2[illegible]-102	0.009	0.41	6.36	0.005	0.02	205.9	0.005	0.03
R2R86-103	0.446	0.57	7.62	0.052	0.37	154.7	0.070	0.38
R2R86-104	0.030	0.81	7.89	0.032	0.13	239.0	0.032	[illegible].86 ?
R2R86-105	0.009	0.84	5.93	0.002	0.09	188.2	0.002	0.11
R2R86-106	0.015	0.23	10.25	0.009	0.08	218.4	0.009	0.09
R2R86-107	0.004	0.15	5.91	0.004	0.09	211.2	0.004	0.09
R2R86-108	0.046	0.81	5.40	0.053	0.38	83.7	0.053	0.41
R2R86-109	0.002	0.32	8.21	<0.001	<0.01	189.8	<0.001	0.01
R2R86-110	0.002	0.07	8.56	<0.001	<0.01	138.3	<0.001	<0.01
R2R86-112	0.014	0.31	6.58	0.005	<0.01	221.3	0.005	<0.01
R2R86-113	0.012	0.11	5.50	0.004	<0.01	211.3	0.004	<0.01
R2R86-114	0.005	0.07	4.41	0.004	<0.01	173.8	0.004	<0.01
R2R86-115	0.058	0.62	3.27	0.038	<0.01	213.2	0.038	<0.01
3S86-L80	0.080	2.26	1.29	0.036	0.32	223.8	0.036	0.33
3S86-L92	0.008	0.58	6.04	0.009	0.21	212.5	0.008	0.22
W886-L76	0.015	0.20	8.62	0.017	<0.01	198.1	0.017	<0.01
W886-L77	0.043	0.99	3.25	0.037	1.59	197.2	0.037	1.58
W886-L78	0.012	0.31	8.40	0.003	<0.01	238.2	0.003	0.01
W886-L79	0.005	<0.01	6.41	0.003	<0.01	246.0	0.003	<0.01
W886-L81	0.124	0.32	7.41	0.080	0.78	227.9	0.081	0.77
W886-A001	0.009	0.64	2.72	0.004	0.06	144.2	0.004	0.06
11[illegible]-L150	0.027	<0.01	5.49	0.009	0.28	219.8	0.009	0.28

Chief Chemist

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BONDAR-CLEGG

Geochemical Lab Report

REPORT: 016-2620

PROJECT: 8

SAMPLE NUMBER	ELEMENT UNITS	Zn PPM	Pb PPM	As PPM
R2R8682		800	356	73
R2R8683		340	184	352
R2R8684		196	235	1256
R2R8685		2500	2700	>2000
R2R8686		10000	1660	>2000
R2R8687		380	186	306
R2R8688		220	46	61
R2R8689		1250	617	>2000
R2R8690		184	134	312
R2R8691		350	779	884
R2R86100		390	132	177
R2R86101		128	37	181
R2R86102		94	105	1552
R2R86103		1600	371	>2000
R2R86104		11600	10700	>2000
R2R86105		520	586	768
R2R86106		520	974	>2000
R2R86107		270	287	556
R2R86108		98	162	>2000
R2R86109		68	36	115
R2R86110		60	45	40
R2R86112		124	76	145
R2R86113		96	171	102
R2R86114		60	28	75
R2R86115		60	137	>2000
3586-L80		1250	1760	>2000
3586-L92		580	302	>2000
W886-L76		52	54	264
W886-L77		5200	341	338
W886-L78		112	138	>2000
W886-L79		74	29	776
W886-L81		10400	1570	>2000
W886-A001		104	78	416
11586-L150		5600	1810	688

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Geochemical Lab Report

EPORT: 016-2124

PROJECT: NONE PAGE 1

AMPLE UMBER	ELEMENT UNITS	Zn PPM	Pb PPM	As PPM	Sb PPM	
46-122-MERV-1		625	370	588	29	
46-122-T11R86-1		80	20	8	3	
46-122-TWBR-L1		20000	20000	>2000	549	[illegible]
46-122-TW9R-19		325	400	1104	121	
46-122-T85R86-18		10150	5500	>2000	134	Rock Sample.
46-122-W8R86-4		150	1375	>2000	53	
46-122-W8S86-1		650	105	520	12	

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Certificate of Analysis

Report: 416-2124

June 26, 1986

Client: Prochem Ltd.

Submitted by: M. Langdon

	ppm Au	ppm Ag
1	0.16	3.01
2	0.01	0.66
3	1.51	333.16
4	1.14	18.90
5	0.58	21.08
6	0.59	17.41
7	0.03	<0.01

[illegible signature]

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Certificate of Analysis

Report: 416-2124

July 2, 1986

Client: Prochem Ltd.

Submitted by: M. Langdon

Rock Samples:	ppm Au +150	ppm Ag +150	grams weight +150	ppm Au -150	ppm Ag -150	grams weight -150	ppm Au weight average	ppm Ag weight average
46-122-MERV-1 LBM	0.15	8.2	3.66	0.16	3.1	220	0.16	3.1
46-122-T11R86-1 IIS	0.04	«1.6	6.28	0.01	0.7	210	0.01	0.7
46-122-TW8R-L1 bBM	4.17	49.4	3.44	1.47	337.0	255	1.51	333.2
~~46-122-TW8R-19~~ Pelso	~~2.88~~	~~12.4~~	~~8.09~~	~~1.08~~	~~19.2~~	~~225~~	~~1.14~~	~~18.9~~
46-122-T8SR86-18 Bs	1.80	9.0	7.77	0.54	21.6	245	0.58	21.1
46-122-W8R86-4 Zinc	0.44	5.5	9.06	0.60	17.8	220	0.59	17.4
46-122-W8S86-1 Granodiorit	0.04	«1.3	7.79	0.03	«0.3	210	0.03	«0.3

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Geochemical Lab Report

REPORT: 016-2689

PROJECT: 9 PAGE 1

SAMPLE NUMBER	ELEMENT UNITS	Zn PPM	Pb PPM	As PPM
35 ~~44~~R86001		1400	1760	1800
35 ~~44~~R86002		210	644	824
35 ~~44~~R86003		960	845	>2000
35 ~~44~~R86004		240	93	166
35 ~~44~~R86005		112	39	47
42 ~~44~~86L151		98	97	480
11S86-153		310	169	131
11S86-154		6000	2490	>2000
11S86-157		230	3185	>2000
11S86-160		2500	1885	560
11S86-165		740	806	1000
11S86-167		310	80	174

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Certificate of Analysis

eport: 516-2124

Wedge 8

July 3, 1986

lient: Prochem

Submitted by: J. Schneider

amples: W8	ppm Au +150	ppm Ag +150	grams weight +150	ppm Au -150	ppm Ag -150	grams weight -150	ppm Au weighted average	ppm Ag weighted average
W 8R-L1	4.78	48.6	6.17	1.55	330.1	257.4	1.63	323.5
8S86-1	<0.01	0.5	19.35	0.03	2.7	179.4	0.03	2.5

Repeats ? on Misc Samples.

C. Fautour

CERTIFICATE OF ANALYSIS

TO: PROCHEM LIMITED
ATTN: KEN FAIR
8032 TORBRAM ROAD
BRAMPTON, ONTARIO
L6T 3T2

RECEIVED
JUL -

CUSTOMER NO. 615

DATE SUBMITTED
27-MAY-86

REPORT 28148

REF. FILE 23608-J3

2 PULPS

WERE ANALYSED AS FOLLOWS:

	METHOD	DETECTION LIMIT
AU OZ/TON	FA	0.001
B PPM	DCP	10.000
F PPM	WET	20.000
S %	XRF	0.010
CL PPM	XRF	50.000
MN PPM	DCP	2.000
FE %	XRF	0.010
NI PPM	DCP	1.000
CU PPM	DCP	0.500
ZN PPM	DCP	0.500
AS %	XRF	0.010
SE PPM	GFAA	0.100
AG OZ/TON	FA	0.100
SN PPM	XRF	2.000
TE PPM	GFAA	0.100
BA PPM	XRF	10.000
W PPM	XRF	3.000
HG PPB	WET	5.000
PB PPM	DCP	2.000

X-RAY ASSAY LABORATORIES LIMITED

DATE 24-JUN-86

CERTIFIED BY ....

SAMPLE	AU OZ/TON	B PPM	F PPM	S %	CL PPM
YVM-100 HD-1	0.034	<10	210	3.69	<50
YVM-100 HD-3	0.039	<10	190	3.61	<50

SAMPLE	AS %	SE PPM	AG OZ/TON	SN PPM	TE PPM
YVM-100 HD-1	1.48	0.2	TRACE	<2	<0.1
YVM-100 HD-3	1.57	0.4	0.41	<2	<0.1

1

X-RAY ASSAY LABORATORIES LIMITED

1885 LESLIE STREET, DON MILLS, ONTARIO M3B 3J4

PHONE 416-445-5755 TELEX 06-986947

CERTIFICATE OF ANALYSIS

TO: PROCHEM LIMITED
ATTN: KEN FAIR
8032 TORBRAM ROAD
BRAMPTON, ONTARIO
L6T 3T2

CUSTOMER NO. 615

DATE SUBMITTED
29-MAY-86

REPORT 28120

REF. FILE 23638-A1

1 PULP

WAS ANALYSED AS FOLLOWS:

	METHOD	DETECTION LIMIT
AU PPB	NA	5.000
S %	WET	0.010
MN PPM	DCP	2.000
FE %	NA	0.020
CU PPM	DCP	0.500
ZN PPM	NA	50.000
AS PPM	NA	2.000
AG PPM	DCP	0.500
WEIGHT GM	NA	0.010

RECEIVED
JUN 2 1986
KF

DATE 20-JUN-86

X-RAY ASSAY LABORATORIES LIMITED
CERTIFIED BY ....

SAMPLE	AU PPB	S %	MN PPM	FE %	CU PPM
YVM-100-1-M1	260	1.13	NSS	59.4	NSS

NSS - NOT SUFFICIENT SAMPLE

SAMPLE	ZN PPM	AS PPM	AG PPM	WEIGHT GM
YVM-100-1-M1	710	3900	NSS	0.39

NSS - NOT SUFFICIENT SAMPLE

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BRAMPTON, ONTARIO
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CUSTOMER NO. 615

DATE SUBMITTED
2-JUN-86

REPORT 28003

REF. FILE 23662-N3

3 C.PULPS

WERE ANALYSED AS FOLLOWS:

	METHOD	DETECTION LIMIT
AU OZ/TON	FA	0.001
S %	XRF	0.010
FE %	XRF	0.010
CU PPM	DCP	0.500
ZN PPM	DCP	0.500
AS %	XRF	0.010
AG OZ/TON	FA	0.100
PB PPM	DCP	2.000

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JUN 17

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X-RAY ASSAY LABORATORIES LIMITED

DATE 10-JUN-86

CERTIFIED BY ....

SAMPLE	AU OZ/TON	S %	FE %	CU PPM	
YVM-100/5C1	0.110	16.7	14.7	710.	x2
YVM-100/5C3	0.026	3.21	4.00	770.	x 2
YVM-100/5T	0.007	0.80	3.30	390.	x 1

SAMPLE	ZN PPM	AS %	AG OZ/TON	PB PPM	
YVM-100/5C1	>4000.	5.50	1.93	>4000	x2
YVM-100/5C3	>4000.	1.58	0.44	>4000	x2
YVM-100/5T	1100.	0.39	TRACE	1900	x1

> - CONCENTRATION TOO HIGH FOR TREATMENT BY GEOCHEMICAL METHOD

SAMPLE	BA PPM	W PPM	HG PPB	PB PPM
YVM-100 HD-1	80	3	140	>4000
YVM-100 HD-3	120	<3	120	>4000

> - CONCENTRATION TOO HIGH FOR TREATMENT BY GEOCHEMICAL METHOD

SAMPLE	MN PPM	FE %	NI PPM	CU PPM	ZN PPM
YVM-100 HD-1	>4000	5.92	4	230.	>4000.
YVM-100 HD-3	>4000	5.95	3	230.	>4000.

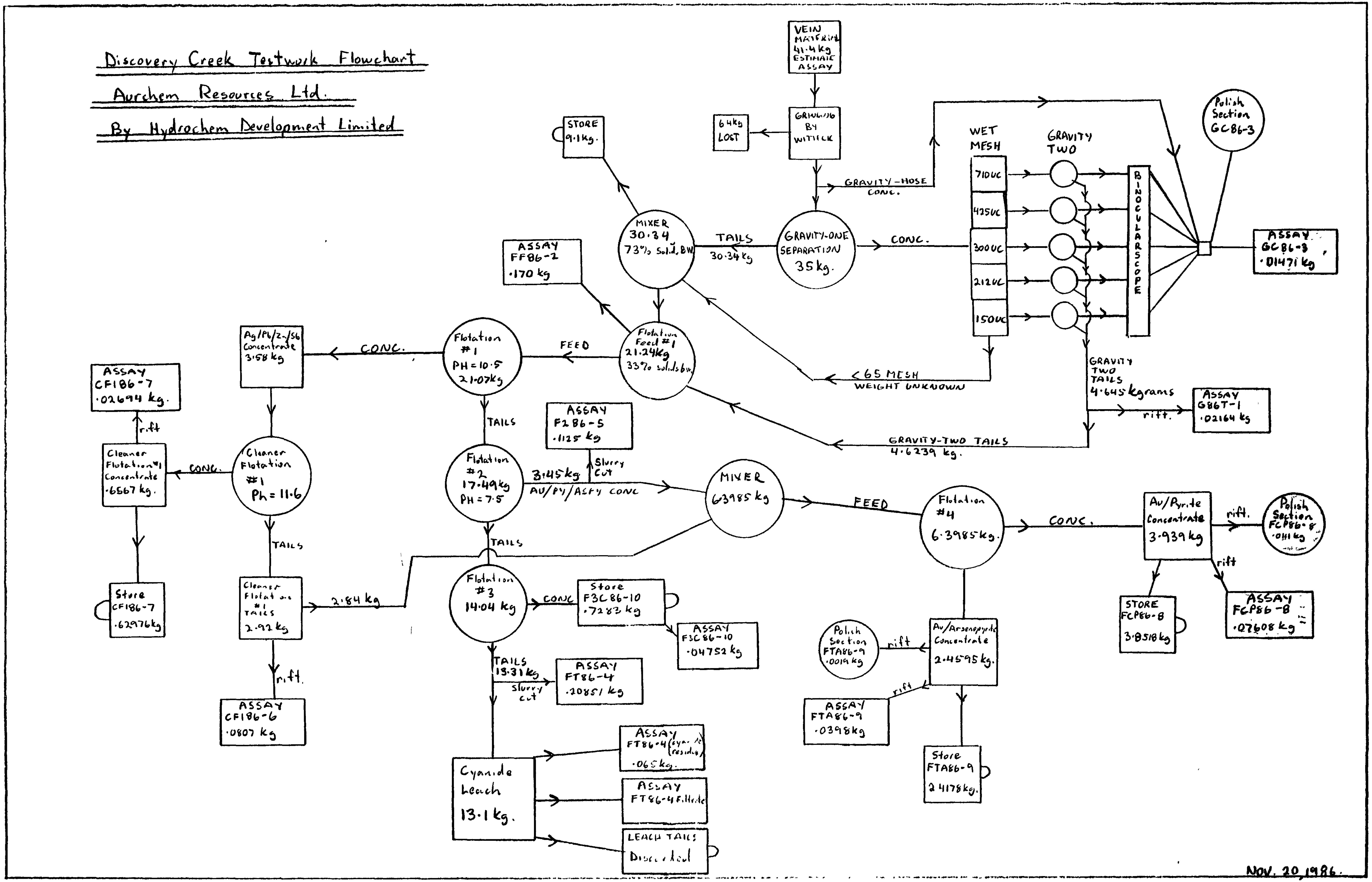
> - CONCENTRATION TOO HIGH FOR TREATMENT BY GEOCHEMICAL METHOD

SAMPLE	AS %	SE PPM	AG OZ/TON	SN PPM	TE PPM
YVM-100 HD-1	1.48	0.2	TRACE	<2	<0.1
YVM-100 HD-3	1.57	0.4	0.41	<2	<0.1

Discovery Creek Testwork Flowchart

Aurchem Resources Ltd.

By Hydrochem Development Limited



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ECONOMIC RESEARCH

Metallurgical Testwork

Material

A) Gravity and Flotation Work

Vein rock samples were gathered to form, a larger compiled sample for the metallurgical testing. Known and estimated assays were used to form a probable base for a head assay.

Rock #	Weight (lbs.)	Au (ppb)	Ag (ppm)	Au (lb. ppb)	Ag (lb. ppm)
FIR86-7A	11	5010	40	55,110	440
FIR86-9A	8.5	5260	12	43,395	99
W886-L57	17.5	9800	85.2	171,500	1,491
W886-L73	2.5	3840	8.4	9,600	21
W886-L63	5.5	5230	85.2	28,765	469
TW886-L1	10.25	1500	333	15,375	3,413
Total	55			323,745	5,933

A total of 36 lbs. of vein rock material was added to the above sample. Assays for this were estimated at Au .08 oz/t and Ag 2.0 oz/ton.

The feed material of 91 lbs. should assay as follows:

Au = .14 oz/ton
Ag = 2.7 oz/ton
Cu = .1%
Zn = .5%
Pb = 2.0%
Sulphides = 15-20% by volume (20-40% by weight)

B) Cyanide Heap Leach Sample

A 9 lb. sample of 10 mesh material was gathered from the above known assay samples to give: Au = .15 oz/ton, Ag = 2.8 oz/ton.

Material Preparation

The 91 lb. sample was sent to Witteck Development Inc. and prepared by plate pulverizing as follows:

about 36th of 5 inch material

↓ (jaw and cone crush)

36th of 10 mesh material ↘ ↙ 54th of 10 mesh

Blend

↓

Plate Pulverize

There appeared to be some loss of material during processing and/or transport as when the material returned it weighed 77 lbs.

Material Sent	-	Material Returned	=	Material Lost
(91 lbs.)		(77 lbs.)		(14 lbs.)

Gravity Testwork

Goal

A) To separate the heavy mineral fraction and thus any large size "free gold" particles. Past work to date had indicated a free gold (+150 mesh) of 5-40% of total gold.

B) To access the heavy mineral fraction through the use of the binocularscope and to prepare a sample for polished section microscope work.

Set-up - Gravity-One

The gravity-one set-up is shown on "sketch II" in Appendix I.

The feed material of 77 lbs. formed a heavy mineral concentrate of 4,623.9 grams or 10.2 lbs.

Notes and observations on the gravity-one separation are in Appendix I.

The gravity-one concentrate was used as feed for the gravity-two process in the small column.

Gravity-Two Preparation

In order for the small column to function efficientlv, the feed material must be of a common size. For this reason the 4,623.9 gram feed sample was wet meshed at:

Mesh	Size
#20	(850µC)
#24	(710µC)
#35	(425µC)
#48	(300µC)
#65	(212µC)

Each mesh size fraction was then individually run through the column. A description of the heavy concentrate of each size fraction is given in Appendix II.

The concentrates of the mesh sizes were then added together with the "hose" concentrate to give a 14.7 gram sample for total assay. (GC86-3)

The remainder of the gravity-one concentrate (or gravity-two tails) was dried and a rifted sample of 21.64 grams was taken for assay. (G86T-1)

The remaining 4,623.9 grams was added to the flotation feed.

A "selected grain" fraction was compiled during the binocularscope study of the "gravity-two" concentrate to produce a polished section for metallic mineral identification and possible gold association. See Appendix III under "Polished Section 1" for a description of selected minerals or grains.

Flotation

Goal

A number of flotations were done to gain a relative idea of concentrates that could be produced to determine their possible value. For example, are the zinc/lead/antimony contents sufficient or workable to make their presence important? and/or how much of the gold is directly related to the silver fraction? The flotations separated the material into basic fractions with the aim being to see which fractions or associations carried what amounts of gold. This would also give an idea of the refractory nature of the gold.

Flotation General

Table I shows the flotation procedure tests # 1 and #2, the reagents used, the pH maintained, etc., for flotations. Actual flotation notes are shown in Appendix VI. The general procedure followed was very close to one used on

General Procedure - Flotation #1 and #2 Table 1

		Flotation #1			Flotation #2				
Total Reagents	Preparation	Conditioning & Flotation (A)	Flotation (B)	Flotation (C)	Conditioning	Flotation (D)	Flotation (E)	Flotation (F)	Total Reagents
(lb/ton) amount									
Ca(OH) (pure)		(2.0) 22 gr							22 grams
H SO4 (10% b.w.)					(2.0) 211 ml				211 ml
CuSO4 .5H O (pure)		(.2) 2.2 gr			(.5) 5.5 gr				7.7 grams
Na Isobutyl (10%) (Xanthate)		(.04) 4.2 ml	(.04) 4.2 ml	(.04) 4.2 ml		(0.5) 5.25 ml	(.05) 5.25 ml	(0.5) 5.25 ml	28.35 ml
Aerofloat 25 (1%)								(.024) 26.5 gr	26.5 grams
MIBC (1%)		(.015) 16.6 ml	(.015) 16.6 ml						33.2 ml
TIME (minutes)	20 minutes	60 8		22	7	8	8	23	
pH	8.0	10.5 10.4		10.4	7.4	7.06	7.3	7.5	

Huestis and Webber material by Britton Research Limited in 1967 (submitted by Mount Nansen Mines Ltd.). We used their basic approach with some variations based on their results and your goals.

Flotation Reagents for #1 and #2

Calculations of the amounts of reagents used in the flotations #1 and #2 are contained in Appendix V.

These flotations basically separated the feed into three fractions:

1) A Ag/Pb/Zn/Sb concentrate fraction.
2) A gold - Fe - As sulphide/gange material tails
3) Tails

The feed material was taken as a cut of a mixture of the gravity-one and gravity-two tails (G86T-1) to make a slurry of 33% by weight.

A sample of the Flotation #1 feed material was taken for assay, (FF86-2) weighing 170 grams.

The procedure and calculations to form the Flotation #1 feed are shown in Appendix IV. The results show that 48.74 lbs. of dry solids were used for the flotation #1 material. The remaining 20.26 lbs was stored for possible future use. A 112.6 gram sample (F286-5) was taken from the flotation #2 concentrate for assay.

Flotation #3

Using the tails of Flotation-Two, a further "scavenger" flotation #3 was done. This flotation basically added sodium sulphide (Na_2S) to "coat" oxidized surfaces on the sulphides to give them greater floating capabilities. Through binocularscope work it was seen that a small percentage (<10%) of the sulphides had oxidized surfaces to some extent.

After flotation #3, a sample of the tails (FT86-4) of 208.51 grams was taken for assay. The remaining tails were used as feed material for the cyanide leach.

The concentrate formed in flotation #3 was 728.3 grams in weight and a sample was rifted for assay.

Cleaner Flotation #1

The Ag, Pb, Zn, Sb concentrate from Flotation #1 was used as the feed. The goal here was to re-float our concentrate at a higher Ph to "clean up" and make a richer concentrate. The pH was maintained between 11.6 and 11.9.

The resultant tails (CF186-6) and concentrate (CF186-7) were both assayed. The concentrate assay sample was rifted off and the remainder stored. The tails sample had a slurry cut taken and the remainder of CF186-6 was mixed with the Flotation 2 concentrate as feed for Flotation #4.

The data and comments of the Cleaner Flotation can been seen in Appendix VIII.

Flotation #4

This flotation was a basic separation of pyrite into the concentrate and arsenopyrite into the tails. It is believed that with the addition of potassium permanganate, the surfaces of the realgar and arsenopyrite are oxidized or coated with MnO . This causes them to be depressed (hyrophyllic) while the pyrite floats (hydrophobic).

The basic aim of this float was to see which direction the gold followed; the pyrite or the arsenopyrite fractions.

Notes and calculations on Flotation #4 are shown in Appendix VIV.

Cyanidation - Flotation #3 Tails

A further "scavenging" of the flotation tails for gold was carried out using a cyanide leach method. (See Kwasi Donyina for details.)

The cyanide/feed slurry was agitated for about 48 hours and then a sample of the solution and tails were taken. The total weight of the solids used were calculated from the pulp density and gave 10.12 kg. The tails assay sample FT86-4 (cyanide residue) weighed 65 grams. The bottle of solution was labeled FT86-4 (filtrate).

Experimental Procedure

Cyanide leaching was performed in a 24' x 12" x 12" tank for 48 hours. The pH of the slurry was adjusted to 10-11 with calcium hydroxide. Cyanide level was maintained at approximately 2lb/t of solution.

6th Flotation; Yukon Vein Material (-100 Mesh)

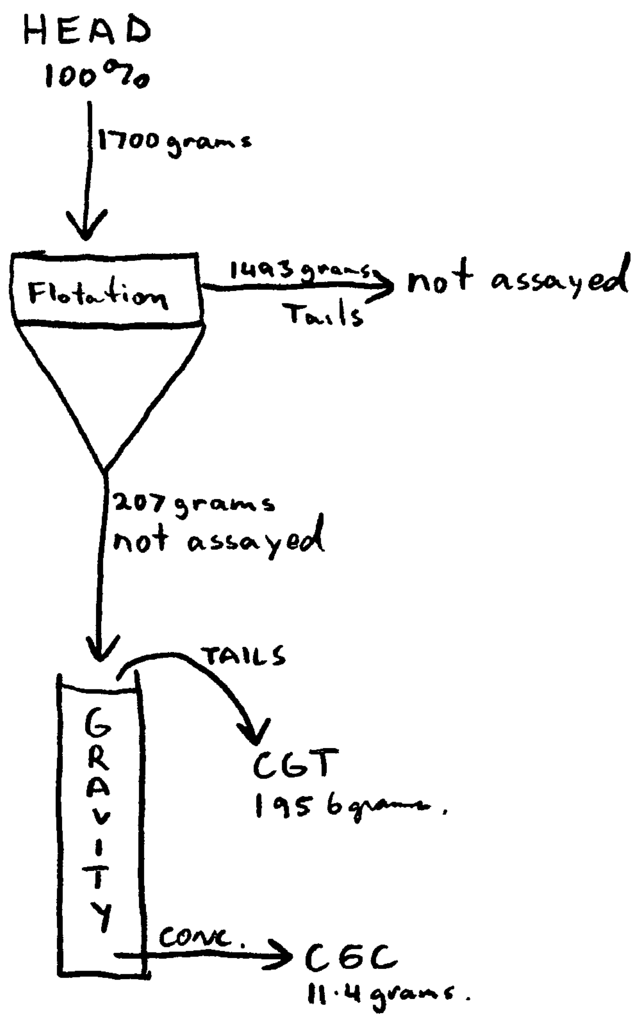
General

This flotation was done on the 1985 vein material and not the feed material used in the previous tests.

Goal

This was an attempt to isolate the carbon constituent by floating it into a concentrate. The flowchart is seen on the next page. No reagents were added in the flotation.

A gravity concentration was then done on the flotation concentrate to see which way the carbon would go.



6th Flotation Flow-Chart

- 1985 vein feed used

Results - Cyanide Leach Test #1

1. Cyanidation of Tails

Flotation tailings FT86-4 were cyanided for 48 hours. Gold extraction varied between 70 and 75% and silver 60-70%. Sodium cyanide consumption was 7.9 lb/ton of dry solids and calcium hydroxide 0.89 lb/ton of dry solids.

(a) Into Cyanidation:

10, 120 grams of tails at 0.033 oz/ton Au and .7oz/ton Ag.
Au units = 333.96
Ag units = 7084

(b) Out of Cyanidation
10, 120 grams of 48 hr residue at .010 oz/ton Au and .28 oz/ton Ag.
Au units = 101.20
Ag units = 2833.6

42, 250 ml of 48 hr filtrate at .007 oz/ton Au and .15 oz/ton Ag.

Therefore total units out are;
Au = 396.95
Ag = 9365.5

(c) Recoveries

Au

% Au extraction: $\frac{295.75}{396.95} \times 100 = 74.5\%$

or $1 - \frac{.010}{.033} \times 100 = 69.7\%$

- Gold extraction of 70 - 75%

% silver extraction: $\frac{6531.9}{9365.1} \times 100 = 69.7\%$

or $\left(1 - \frac{.28}{.70}\right) \times 100 = 60\%$

- Silver extraction of 60 - 70%

(d) Reagent Consumption

NaCN: $\frac{40 \times 2000}{10120} = 7.9$ lb/ton dry solids

$Ca(OH)_2$: $\frac{4.52 \times 2000}{10120} = .89$ lb/ton dry solids

Discussion

The fairly high extraction for Au and Ag is very encouraging and reagent consumption is not bad. A test using CIP cyanidation on the original feed material before flotation should be carried out in view of these results.

For comparison's sake a similar test on ore from the McLaughlin Mine gave initial results of 5 - 80% recovery depending on the variety of ore. This was improved to 90% by adding sulphuric acid followed by high temperature/high pressure oxidation in autoclaves.

A crude conclusion from our testwork as a whole is that it is possible that in our test sample about 45% of the gold may occur as individual small grains or as electrum possibly associated with the galena fraction. A similar association was found at the McLaughlin Mine as gold occurred with Ag - Sb sulphosalts.

2) Gravity-One Results

(a) Concentrate - 4.645 kg assayed .28 oz/ton Au and 6 oz/ton Ag.
- This contained 35% of the gold in 13.7% of the total sample by weight.

The assays stated are from the gravity-one concentrate after the concentrate of gravity-two was removed. If we put the gravity-two concentrate back in we get;

Au - .304 oz/ton
Ag = 6.08 oz/ton
Ag/Au = 20.1

Discussion See Gravity-Two discussion

3) Gravity-Two Results

(a) Concentrate - this gravitational test was basically to get a "super-concentrate" for polish section and binocularscope studies of the heavy mineral fraction. This gave a .01471 kg concentrate assaying;

Au $\simeq$ 7.74 oz/ton
Ag = 31.02 oz/ton
(3% of total Au and .3% of total Ag)

Discussion

The results show that for this particular sample, a saleable product was not produced using a gravitational method. Only 3% of the total gold content was recovered in the gravity-two concentrate although 35% of the gold was in the feed or gravity-one concentrate.

An explanation for the results can be seen in the binocularscope study of the individual concentrates of the mesh fractions. Visible gold was found in the 100 - 400 micron size with the bulk being 250 microns or less. When the wet meshing was done for the gravity-two work, any particles of 212 microns or less were discarded into the tails of gravity-two. If we assume that all the gold in gravity-one concentrate was free gold, it means that we wet meshed 29% of the total gold back into the tails, the particles being all less than 212 microns.

For comparison, in the McLaughlin Mine, gold was found to be very fine grained and disseminated usually as electron averaging 20 microns in size.

Further evidence was seen in the 1985 material testwork (6th flotation) that showed the bulk of the gold in the 150 micron size fraction. This was the finest fraction used in that testwork.

Overall the work shows that it is possible that the rock sample may contain a very high content of free gold but as very fine particles. This accompanied by the relatively high content of larger sized sulphide material make a gravity separation of these fine gold particles almost impossible as they would float out in the tails before many of the sulphides.

4) Flotation #1 Results

Individual assays and distribution percents can be seen on the results chart.

The attempt here to form a Ag-Pb-Zn-Sb concentrate with no As was quite successful. While the concentrate contained 16.8% by weight of the total sample, it contained 77% of the silver, 53% of the Pb and 63% of the zinc. The antimony did not go into the concentrate which may indicate some oxidation of the Sb minerals. The unexpected surprise in this test was that 46% of the total gold also went into the concentrate indicating a possible high free gold content that is not associated with the arsenic as originally believed. The Pb and Zn recoveries were fair but could be improved. The pH used in this flotation of 10.5 is very close to the borderline for galena to float and is a little better, but not by much for sphalerite. By lowering the pH we would probably float much more of the Pb and Zn but would also risk the possibility of floating more of the As.

5) Cleaner Flotation #1 Results

A cleaning flotation of the concentrate from Flotation #1 was done in order to determine how good a concentrate could be formed and its possible commercial value if any. The flotation was both a success and a failure.

It was a failure in respect to the recovered percentage of the total amount of the minerals. This was probably due to two main reasons:

TEST & PRODUCT	Assays										Distribution						Individual Distribution %						
	Wt %	Au oz/t	Ag oz/t	Pb %	Zn %	As %	Sb %	Cu %	Total C %	C (NOCO3) %	Au	Ag	Pb	Zn	As	Sb	Au	Ag	Pb	Zn	As	Sb	Wt. %
GRAVITY																							
Head Gravity	100	.109	4 16														100	100					100
Gravity-One Tails	86.66	.086	3.9														64.8	80.8					86.66
Gravity-Two Tails	13.3	.264	5.92														32.2	18.9					13.3
Gravity-Two Concentrate	.043	7.74	31.02														3 0	.3					.043
Gravity-One Concentrate	13.7	28	6.00														35.2	19.2					
Head	100	.109	4.16								100 0	100 0					100	100					
Gravity Concentrate	.043	7.74	31.02								3 0	3					3 0	.3					
Flotation #1 Feed	99.96	.125	4 34	3.04	1 17	1 98	.06		5 1	3 52	97 0	99 7	100	100	100	100	100	100	100	100	100	100	100
Flotation #1 Concentrate	16.8	.337	19 84	9.63	4 39	1 53	13				44.1	76 8	53 4	63 2	13.0	36.5	45.5	77.0	53.4	63.2	13.0	36.5	16.8
Flotation #1 Tails	82 8	.082	1.17	1.72	.52	2 09	.05				52 4	22 1	46 6	36 6	86.9	68.6	54.0	22.2	46.6	36.6	86.9	68.6	82.8
Cleaner Flotation Feed	16.8	.337	19.84	9 63	4 39	1.53	13				44 1	76 8	53 4	63 2	13 0	36 5	100	100	100	100	100	100	100
Cleaner Flotation #1 Tails	13.7	.242	13.95	7.60	2 12	1 73	07				25 8	44 1	34 4	24 9	12 0	16 0	58 5	57.4	64 4	39.4	92.3	43.8	
Cleaner Flotation #1 Concentrate	3.1	.762	46 09	18.68	14 52	.67	.39	1 09			*18 3	32 7	12 3	38.4	1.0	20 1	41 5	42 6	23.0	60.8	7.7	55.1	18.5
Flotation #2 Feed	82.3	.082	1 17	1 72	.52	2 09	.050				52 4	22 1	46 6	36 6	86.9	68 6	100	100	100	100	100	100	100
Flotation #2 Concentrate	16 2	310	6 88	3 80	2 44	5 82	070				39 1	25 6	20 3	33 9	47 7	19 0	74 6	115 8	43.6	92.6	54.9	27.7	19.7
Flotation #2 Tails	66.1	.026	85	1 21	05	1 17	.045				13 31	12 9	26 3	2 8	39 1	49.6	25 4	58 4	56.4	7.7	45.0	72.3	80.3
Flotation #4 Feed	30 1	.266	8 79	4 90	2 16	4 40	07				62 2	60 8	48 6	55 6	66 9	35 1	100	100	100	100	100	100	100
Flotation #4 Concentrate	18.5	.225	9 04	6 16	2 48	3 10	07				*32.4	38 5	37 6	39 3	12 2	21 6	52 1	63 3	77.4	70 7	43 5	61 5	61 5
Flotation #4 Tails	11.6	324	5.82	2 88	1 64	6 48	.07				*29 1	15 5	11 0	16.2	37 8	13 5	46 8	25 5	22.5	29.1	56.5	38 5	38.5
Flotation #3 Feed	66 1	109	85	1 21	05	1 17	.045				55 9	12 9	26 3	2 8	39.1	49 6	100	100	100	100	100	100	100
Flotation #3 Concentrate	3.5	1.50	3.53	2 76	32	3 62	.06				*39 9	2 8	3 1	0.9	6.3	3 4	71.4	21.7	11.8	32.1	16.1	6.9	5.3
Flotation #3 Tails	62.6	.033	7	1 13	04	1 04	044		3.10	27	16 1	10 1	23 3	2 1	32 9	46 0	28 8	78.3	88.6	75.0	84.1	92.7	
Total Recovered	21 7										122 7	89 8	64 0	94 8	57 3	58 6							
Cyanide Feed	62 6	.033	7	1.13	04	1 04	.044	3 10	3 10	27	16 1	10 1	NaCn Consumption = 7 9 lb/t dry solids				100	100					62.6
Cyanide Residue	62 6	.010	28								4 8	4 0	Ca(OH)2 Consumption = 89 lb/t dry solids				29 8	39.6					62.6
Cyanide Filtrate		.25ppm	9 3ppm														70-75	60-70%					

Note A line under the assay indicates calculated assay

1) The flotation was not carried out long enough and only recovered 18.5% by weight of its feed material.

2) Once again their was a problem with the pH used as discussed in Flotation #1. In this cleaner flotation the pH was increased from the 10.5 used previously to a range of 11.6 -11.9. The reason for this increase in pH was to clean the concentrate of As by depressing it, but it also put Pb below the floatable range and Zn as a borderline case. This showed up as only 23% of the Pb and 61% of the Zn went into the concentrate. The Au showed only 42% recovery from the feed which may indicate that some of the gold (possibly free gold) is caught up in the galena and/or sphalerite.

To produce a concentrate as we have done, at the recoveries shown, it can be "crudely" estimated that it's value would be in the range of $30 - $35 per ton of ore at present day prices. Further Pb - Zn separations may also have to be done. From a ton of ore of similar grade we would get a 62 lb. concentrate at;

Au -- .762 oz/ton
Ag -- 46 oz/ton
Pb -- 18.5%
Zn -- 14.5%

With further tests, it can be calculated that the concentrate could be improved to getting a concentrate of 94 lbs at;

Au -- .78 oz/ton
Ag -- 66 oz/ton
Pb -- 32%
Zn -- 15%

Obviously if future geological work indicates the probability of an ore body, further flotation tests should be done.

6) Flotation #2 Results

The aim to produce a Au/pyrite/arsenopyrite concentrate met with mixed success. From the feed material it can be seen that a good percentage of the Au, Ag and Zn went into the concentrate but Pb, As and Sb showed poor recoveries. We ended up with a tails that contained, from the total feed, 13% of the gold and silver, 26% of the lead, 39% of the arsenic and 50% of the antinony. It was postulated that the poor recoveries were caused by the ore material being partially oxidized as seen in the binocularscope study, thus causing the sulphides to become depressed. During the testwork it could be seen that the tails contained a large amount of sulphides so Flotation #3 was set up to investigate this further.

7) Flotation #3 Results

This flotation was in effect a further "scavenging" of the material in an attempt to float the possibly oxidized metallics into a concentrate.

The results show that 71% of the gold was recovered in 5.3% by weight of the feed. The 1.5 oz/ton Au was the highest concentrate value of gold obtained in the flotations. All of the other constituents of Ag, Pb, Zn, As and Sb gave very poor recoveries. This indicates the possibility of the following:

1) The Au definitely did not follow the As or any of the other minerals in this flotation. The gold may be associated with pyrite but Fe was not assayed. The gold could also occur as free gold that had oxide (or possibly manganese oxide) coatings so that it did not float in earlier flotations.

In possible future flotation tests, it may be a better approach to add the Na_2S to the initial flotation (flotation #1). This may prove to give a much higher gold recovery in our initial Ag-Pb-Zn (Au) concentrate.

8) Flotation #4 Results

This flotation was aimed at separating the material into a pyrite rich concentrate and an arsenopyrite rich tails to determine which group the gold followed.

The results show inconclusive evidence. Although the tails/concentrate did assay As as 6.5%/3.1%, the difference in weight gave only 56.5% of total As of the feed in the tails. The gold assays of tails/concentrate were .324/.225 (oz/ton) which appears to make gold follow the As trend. Contrary to this the tails contained only 46.8% of the gold from the feed while it contained 56.5% of the As.

Petrographic Study

The petrographic study was sub-contracted out to Brent McInnes of McMaster University. The report can be seen in Appendix XI.

From the gravity sample (GC86-3) no gold was seen although a free gold particle was put in the sample. It is believed that the gold "ball" was smeared and broken during polished slide preparation and may have gotten smeared onto the polisher. From personal conversation with Mr. McInnes he stated that there was a lot of very fine particles of what he though may be gold but were too small for positive identification.

Some of the gangue minerals listed in the rough notes were quartz and/or other silicates, plagioclase feldspars, zircon, and barite. Most of these contained small inclusions of pyrite, galena and/or magnetic. One other gangue mineral which I though was present from the binocularscope study was flourite with chlorite and magnetite inclusions. Mr. McInnes identified this as a possible silicate or feldspar with magnetite inclusions. (Note: polished sections cannot properly identify transparent or translucent minerals such as flourite, quartz, feldsparr, etc.)

The two other slides of the flotation #4 pyrite concentrate and arsenopyrite tails also showed no gold although their assays indicate .225 and .324 oz/ton Au respectfully. This could be for a number of reasons such as (a) the gold was too fine to be seen, (b) the polished section fractions contained no gold or (c) the gold is as free gold particles which did not get on the polished slides.

A comparison of the point counting method for estimating the percentages of minerals and the assays show the following: (Note: assay in brackets)

(a) FCP86-8
pyrite = 84%
arsenopyrite = 3.9% (As = 3.1%)
galena = 4.5% [(pb = 6.16%) -- 8% galena]
sphalerite = 6.9% [(Zn = 2.5%) -- 5% sphalerite]
Chalcoprite = .4%
Other = .3%

(b) FCP86-9
pyrite = 57%
arsenopyrite = 28.1% (As = 1.04%)
galena = 3.5% [(Pb = 1.13%) -- 1.5% galena]
sphalerite = 4.1% [(Zn = .04%) -- .08% sphalerite]
chalcopyrite = .5%
other = 6.9%

The methods show a great difference in the results. This may be to; (a) unrepresentative samples in either the assay or polish section samples (b) in the point counting method only the larger particles are counted. The groundmass may have a preferred type of mineral changing the percentages. (c) in the point counting method many of the particles were found to be a mixture of minerals so the clast was counted as the dominant mineral present.

One major observation from the polished section report is that supergene activity was seen in the slides. Other companies working in the Mount Nansen area have found that this supergene zone may go 200-300 feet in depth and in this zone cyanidation tests show an improved recovery rate for gold varying from 60 - 90%.

Cyanide Leach Test #2 - Results

Based on the positive results from he first cyanide leach test on the flotation tails, a second test was set up using the original feed material.

Into Cyanidation

	Au	Ag
1293 grams of ore at .105 oz/ton Au	135.81	5807.4

Out of Cyanidation

	Au	Ag
31.4 g of 45 hr. carbon at 3.48 oz/ton Au	109.06	
40.9 oz/ton Ag		1281.8
31.04 g 115 hr. carbon at 0.12 oz/ton Au	3.72	
4.05 oz/ton Ag		127.7
30 g of 210 hr. at .089 oz/ton Au	2.67	
10.1 oz/ton Ag		303.0
1293.4 g of 210 hr. residue at .055 oz/ton Au	71.14	
2.43 oz/ton Ag		3143.0
TOTAL	186.59	4853.5

Gold Extraction

(a) 48 hours $\frac{109.06}{186.59} \times 100 = 58.4\%$

(b) 115 hours $\frac{112.78}{186.59} \times 100 = 60.4\%$

(c) 210 hours $\frac{115.45}{186.59} \times 100 = 61.9\%$

Silver Extraction

(a) 48 hours $\frac{1281.8}{4853.5} \times 100 = 26.4\%$

(b) 115 hours $\frac{1407.5}{4853.5} \times 100 = 29.0\%$

(c) 210 hours $\frac{1710.5}{4853.5} \times 100 = 35.2\%$

Discussion

Gold extraction was fairly good at 61.9% which still indicates that 50% of the gold may be as fine free gold particles. The 40% Au not recovered may be refractory gold that went into the arsenopyrite/pyrite concentrate of flotation #2.

The silver recovery at 35% was poor. Our flotation #1 concentrate extracted 77% of the silver with very little arsenic. The silver is believed to be associated mainly with the galena. The poor silver recovery in the cyanidation may be the result of lack of oxygen due to O_2 competition by the cyanide consumers. A pre-oxidation of the feed may bring the silver recovery up (possibly to 70%) so; further testwork is needed.

The graph on the next page shows the cyanide ~~and lime~~ consumption of the test. Consumption ~~of both~~ is quite high. A pre-oxidation of the feed would probably also lower this consumption.

6th Flotation Results

In other reports on the Mount Nansen area, it has been suggested that the gold may be associated with the carbon in the veins. Assays on Aurchem's vein material have shown organic carbon values (carbonate not included) of up to 3%. The purpose of this flotation was to see; (a) if carbon could be concentrated; (b) if the gold followed the carbon; and (c) to see how carbon reacts to a gravity test.

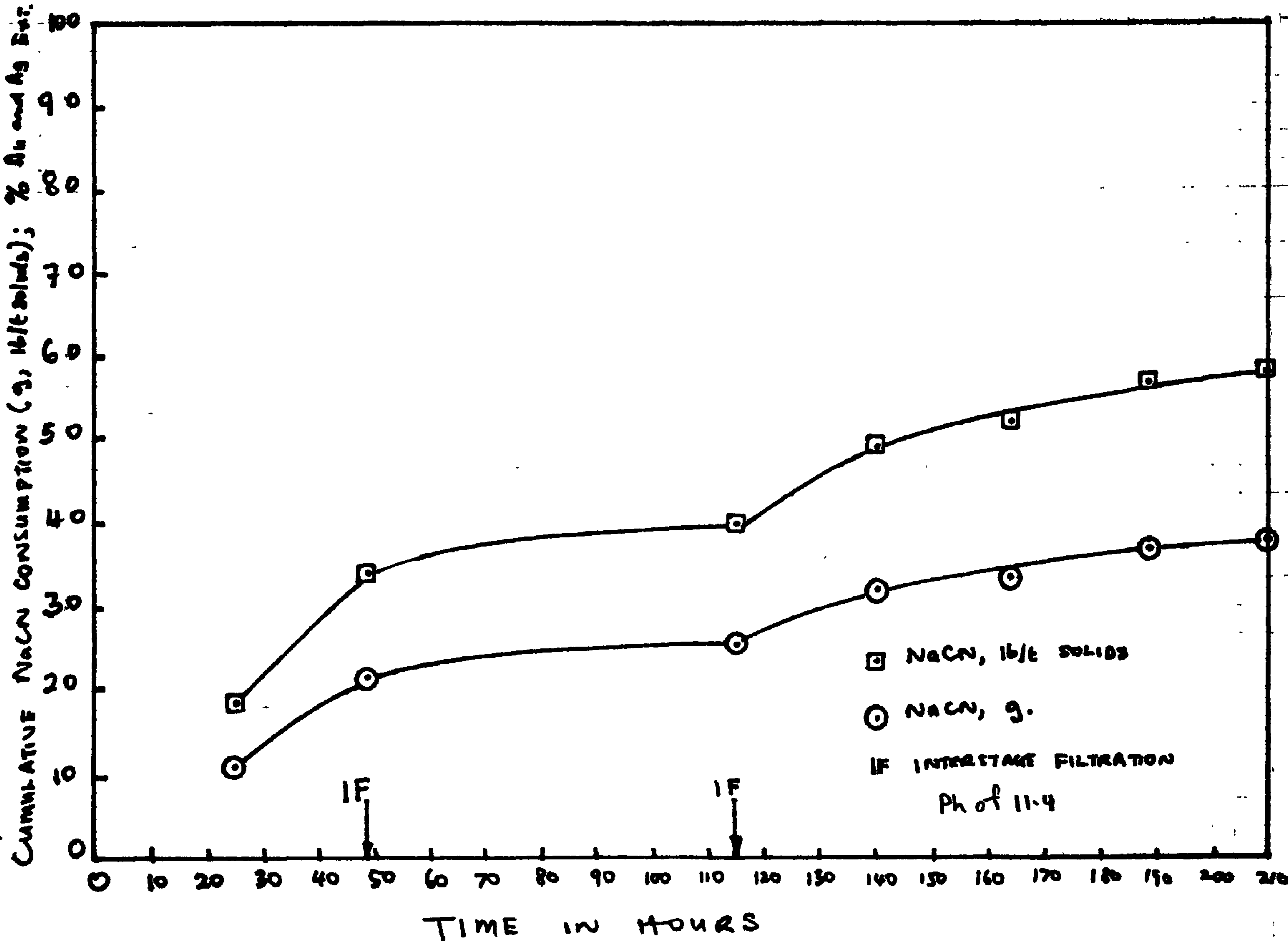
The results chart shows that the organic carbon did go into the concentrate but that the gold did not follow. This shows that the gold is not directly tied to the carbon but the presence of organic carbon may still indicate the presence of gold.

The gravity test shows a slight concentration of organic carbon in the concentrate but again not in proportion to the gold content.

The results show on the next page inconclusive evidence but generally indicate that for recovery processes, the gold-carbon relationship may not be important. The carbon content may show up as a factor in the cyanide leach tests as the carbon may grab gold and silver out of solution.

DEC 22, 1986

YUKON 2: GOLD AND SILVER EXTRACTION, CYANIDE CONSUMPTION AS FUNCTION OF TIME.



6th Flotation Results

Assay

Sample #	Weight (gr)	Au oz/ton	Total C %	CO %	C %	Ag oz/ton
6-CGC	11.4	.140	5.58	25.3	.52	5.36
6-CGT	195.6	.026	7.45	35.9	.27	2.38
HEAD	54.7	-----	7.47	36.8	.11	----

Head sample assays Au -- .037 oz/ton
Ag -- .31 oz/ton

Stream	Weight (gr)	% b.w.	Au oz/t	Total C %	CO %	C %	Ag oz/t	Au %	Total C %	CO %	C %	Ag %
Head	1700	100	.037	7.47	36.8	.11	.31	100	100	100	100	100
6-Flot Ft-Tails	149.0	87.8	.037	7.49	37.0	.09		87.8	88	71.8	67.9	0.0
6-Flot FC Conc.	207.0	12.8	.032	7.35	35.3	.29	2.54	10.5	12	11.7	32.1	100
6-CGC	11.4	0.67	.140	5.58	25.3	.52	5.36	2.5	.5	.5	3.1	11.6
6-CGT	195.6	11.5	.026	7.45	35.9	.27	2.38	8.0	11.5	11.2	28.2	88.5

From the Gravity Test

The 6th flotation put 32.1% of the total C in the concentrate. This used as feed for the gravity test gave 90.35% of C in the concentrate and 9.65% C in the tails.

Appendix I

Notes and Observations on Gravity-One (Large Column)

The feed slurry was agitated and pumped the into column via value 3 when the column was full of water and agitator was on slow speed. The very fines floated off as tails during the material filling process.

Phase 1

In this case the water feed was varied and the agitator motor speed was kept constant at 62.

(a) Water rate (slow = 1.4 l/min or .16 ft/min in column
- Water entered via valves 3 and 2 (not on full).
- Only the very fines were coming off as quartz and calcite (no sulphides).

(b) Water rate (fast) = 4.7 l/min
- Water entered via valves 1 and 2 (not at full water pressure).
- Material coming off contained 2% sulphides mostly as sulphides within quartz (still fine material).

(c) Water rate (full) = 82.5 l/min.
- Water entered at full pressure via valves 1, 2 and 3.
- Coarser material coming off with 8% sulphides.

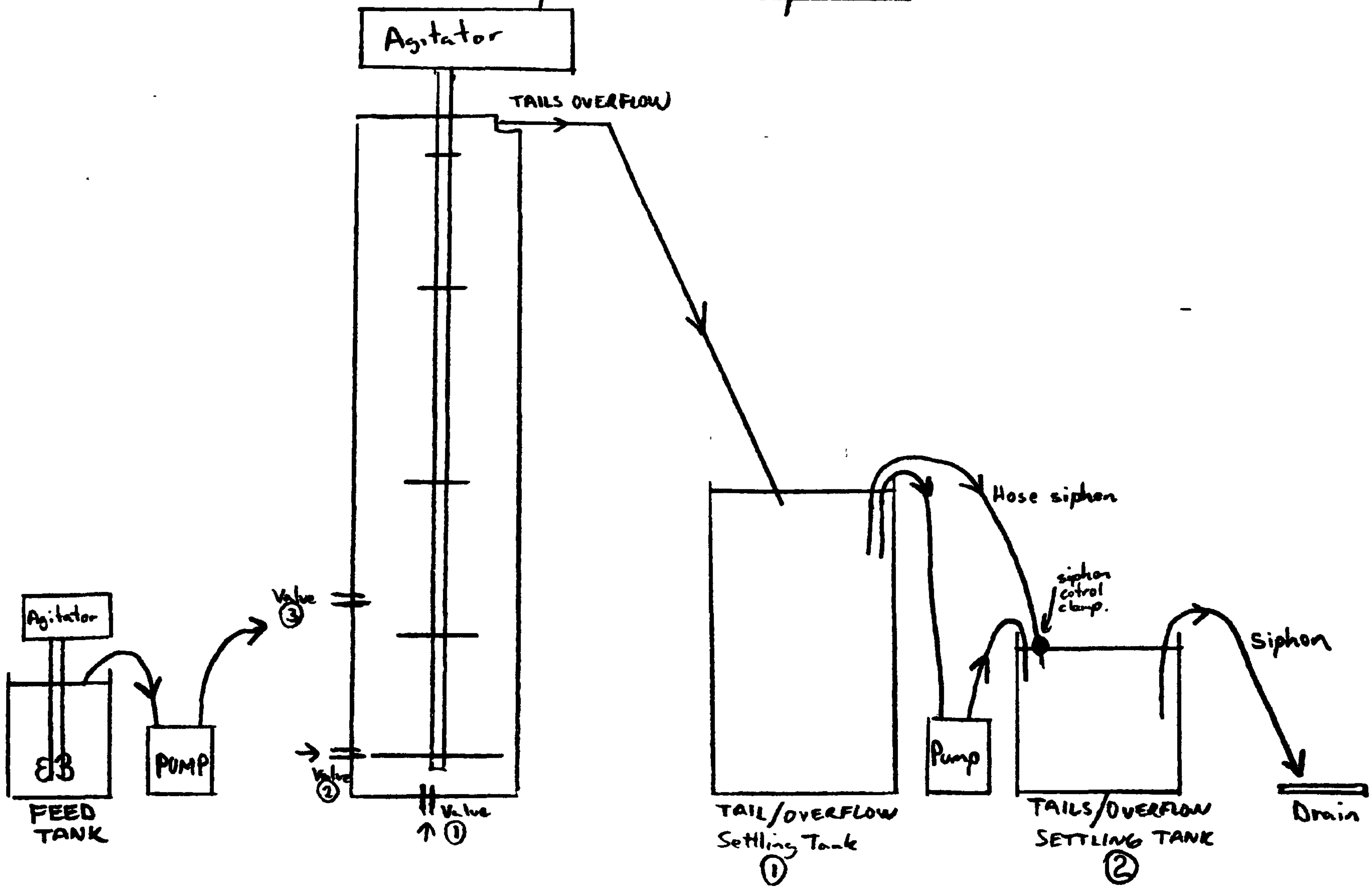
Phase 2

In this case the water rate was kept constant and the agitator motor speed was varied. (Water rate at 82.5 l/min through valves 1, 2 and 3 at full water pressure.)

Agitator	Float Lithology		Particle Size	
Motor Speed	Sulphides	Qtz/Calcite	Sulphides	Qtz/Calcite
22	2%	98%	350-400UC	450UC
42	8%	92%	400UC	500UC
45	10%	90%	350UC	600UC
54	20%	80%	400UC	450UC

Note: Sulphides were mostly as quartz with sulphide particles and not solid sulphides.

Sketch II - Gravity-One Setup



Important

An important observation from the process is that during pumping of the feed into the tank a very good gravity separation or pseudo-panning of the feed was made by the pump and hose. The pulsing pump feed through the elevated hose produced a concentrate in the hose of the coarsest material which contained the bulk of the visible gold. This should be remembered during future tests.

About 30% of our "hose" sample was lost before we realized its significance and presence.

Other Notes

During flotation the sulphides were concentrated in the lower parts of the column increasing in percentage and size down the tank. The "gange" (qtz/calcite) material showed a decrease in size up the column and also a greater degree of roundness.

During high agitation and full 3 valve water rate the coarse sulphides on the bottom of the column showed a "dancing sulphide" fraction. At lower water rates this fraction settled to the bottom

Appendix II

Description of Gravity-Two and Hose Sample Concentrates

- Total of all samples combine to form sample GC86-3

Screen Mesh	Size (UC)	% Sulphides	% qtz/calcite	Composition Sulphides	Visible Gold	Other Comments
65 mesh	212-300	98%	2%	60% galena 20% marcasite 10% pyrite 8% Aspy. <1% pyrrhotite	6	<2% brassy coatings or oxidized surfaces
48 mesh	300-425	97%	3%	50% galena 23% marcasite 15% pyrite 10% Aspy <1% pyrrhotite	1	3% have oxidized coatings. 2% of marcasite have brassy-yellow coatings.
35 mesh	425-710	97%	3%	50% galena 25% marcasite 15% pyrite 7% Aspy	none seen	small oxidized stains on 40% of Fe sulphides 1-2% brassy-yellow marcasite.
24 mesh	710-850	75%	25% galena & pyrite in qtz clast.	8% galena 20% marcasite 40% pyrite 3% Aspy *4% Sphalerite	none seen	50% of Fe sulphides have minor oxidized patches. No brassy-yellow marcasite.
20 mesh	>850	11%	89% 60% qtz. 20% sil. calcite 8% flourite 1% other	2% (galena & sphalerite & Aspy) 6% pyrite 2% pyrrhotite 1% malachite (tuberous)	none seen	flourite has pyrrhotite and/or magnetite in it.
Hose Conc.	Largest size	>99%	1%	45% galena 22% marcasite 18% pyrite 10% Aspy 5% pyrrhotite	6-15 (100-400 UC size)	50% of Fe sulphides gave oxidized patches. 25% of marcasite have brassy-yellow surfaces

Note: In all above cases the galena was the finer fraction.

Appendix III - Polished Sections

1) Selected Particles for Polished Section #1

a) From hose sample
- 3 samples of rusty coated pyrrhotite, magnetic, black and rusty coatings, silvery white inside, irregular shapes, 1 tubular, tarnishes to a deep blue colour, form small round balls also, 1 large sample 3/4 cm across.

b) From hose sample
- possibly a silver alloy, 1 good cleavage, silvery colour with minor oxide coating, brittle, platy cleavage.

c) From hose sample
- a steel grey metallic, no oxides, dull lustre, possible arsenopyrite, two samples.

d) From hose sample
- 2 samples chalcopyrite, brassy yellow colour.

e) From hose sample
- 2 samples, tetrahedrons, silver grey colour, 1 good cleavage and two poor, marcasite or arsenopyrite.

f) From hose sample
- 2 samples, pyritic agglomerations, dull pyritic colour.

g) From hose sample
- 2 samples of cubic galena.

h) From hose sample
- 2 samples, silvery-white colour, choncoidal fracture, probably marcasite.

i) From hose sample
- 2 samples, quartz with pyrite and galena within.

j) From hose sample
- 2 sample, quartz with pyrite agglomeration with a platy sheared surface.

k) From hose sample
- 1 sample, native gold, ball shaped.

l) From the 850 UC sample
- 3 samples, mixture of quartz and sulphides.

m) From the 850 UC sample
- 2 samples, cloudy white/green material not metallic, magnetic probably flourite with chlorite and magnetite (or pyrrhotite), minor sulphides within.

n) From the 850 UC sample
- 1 sample, purplish/pink mineral with minor disseminaqted sulphides within, rhodachrosite??

o) From the 850 UC sample
- 1 sample, possibly a piece of organic carbon, could be a piece of rubber hose, gas bubbles thoughout, black colour, spongy appearance, elastic and very soft, bounces back to shape.

p) From the 850 UC sample
- 1 sample, silvery white or tin white colour, magnetic strained appearance of curved cleavage, possibly strained pyrrhoitite.

r) From the 710 UC sample
1 sample, sphalerite with quartz and calcite, black-red colour.

s) From the 710 UC sample
- 1 sample, purplish/pink sphalerite with quartz.

t) From out of flotation air hose
- 1 sample, platy wire metallic, possible shaving of foreign material from a lathe, brassy coating, silvery white and/or transparent under brassy coating, magnetic, malleable, can be bent and squashed, unknown.

2) A 1.9 gram sample of FTA86-9 (Flotation #4 tails) was sent.

3) A 11.1 gram sample of FCP86-8 (Flotation #4 concentrate) was sent.

Appendix IV

Flotation #1 Feed - Pulp Density

The flotation was conducted in a 24' x 12" x 12" tank. Two samples from the holding tank were taken to determine the % solids by weight in order to be able to determine the amount of the slurry needed to form a 33% by weight slurry in the flotation tank.

(1) -124 ml of slurry
mass (wet) = 233.6 grams
mass (dry) = 170.6 grams
233.6 - 170.6 = 63 grams water.

$$\text{wt \% of } \frac{\text{solids}}{\text{slurry}} = \frac{170.6 \text{ grams}}{233.6 \text{ grams}} = 73\% \text{ by weight}$$

(2) -480 ml of slurry
mass (wet) = 932.9 grams
mass (dry) = 682.5 grams
932.9 - 682.5 = 250.4 grams of water

$$\text{wt \% of } \frac{\text{solids}}{\text{slurry}} = \frac{682.5 \text{ grams}}{932.9 \text{ grams}} = 73\% \text{ by weight}$$

Need dry gravity of solids;

a) Gravity-One Tails = 66.8 lbs _ 30 kg.
b) Gravity-Two Tails = 4.65 kg.
c) Density of galena = 7.6
d) Density of pyrite/arsenopyrite/qtz = 5.3
e) Observed material is 50% galena by volume.

$$\text{\% galena by weight in Gravity-Two Tails} = \frac{\%(e)C7.6}{\%(e)C7.6 + (1 - \%)\ C\ 5.3} = \frac{.5(7.6)}{.5(7.6) + 5(5.3)} = 59\%$$

Gravity of Gravity-Two Tails = % (galena) (7.6) + % (other) 5.3 = 59(7.9) + 41 (5.3) = 6.6

- assume gravity of Gravity-One tails = 2.7

$$\text{Dry Gravity of Mixture (Flot'n Feed)} = \left(\frac{30}{30 + 4.7}\right)(2.7) + \left(\frac{4.7}{30 + 4.7}\right)(6.6) = 3.2$$

- Calculation of Pulp Density at a dry density of 3.2 and a slurry of 73% solids by wei

$$C\text{slurry} = \frac{C\text{ dry}}{\frac{\text{wt solids}}{\text{wt slurry}} + \left(1 - \frac{\text{wt solids}}{\text{wt slurry}}\right) C\text{ dry}} = \frac{3.2}{.73 + (1 - .73)\ 3.2} = 2.0$$

* a check of Cslurry using the Marcey Density Scale gave 2.05
Slurry Gravity = Specific Gravity x density of water

= 2.0 x 62.4
= 125.3 lbs/ft

- Calculations of volume of slurry in holding tank;

Vol = πr^2 x height
= 3.14 x $(18")^2$ x 5.125 = 5213.97 cubic inches
- 5213.97 - 12^3 = .75 ft^3

wt of slurry = density x vol
= 125.3 x .75
= 94.56 lbs.

wt of dry solids in slurry = 94.56 x 73%
= 69 lbs.

- Calculation of Slurry to add to Flotation Tank to get 33% solids by weight.

Flotation tank = 24" x 12" x 12" (fill up to 22")

need (22" x 12" x 12") - 12^3= 1.83 cu. ft. of material at 33% b.w.
Cd = 3.2 (preciously calculated)

$$\text{Cslurry} = \frac{\text{Cd}}{\% + (1-\%)\text{Cd}} = \frac{3.2}{.33 + .67\ (3.2)} = 1.29$$

wt of = (sp. gr. x vol.) vol
slurry = (1.29 x 62.4) 1.83 = 147.7 lbs

to get the weight of the dry solids to be added, we have;

$$\frac{\text{wt. dry solids}}{\text{wt. of slurry}} = 33\%$$

wt of dry solids = .33 x 147.7 = 48.74 lbs
needed in flotation tank

$$\text{must add; } \frac{\text{weight of solids needed}}{\%\text{ by weight of solids}} = 66.7 \text{ lbs of slurry}$$

available at 73% C = 2.05 -- 127.9 lbs/cu ft.

$$66.7 \text{ lbs of slurry} = \frac{\text{wt of solids needed}}{\text{slurry gravity}}$$

$$= \frac{66.7 \text{ lbs}}{125.3 \text{ lbs cu ft}} = .53 \text{ cu ft of slurry}$$

$$Vol = .53 \text{ cu ft} -- = .53 \times 28.3 \text{ l/ft} = 14.7 \text{ l}$$

14.7 litres of slurry were put into the flotation tank and topped up to 22 inches in height with water.

Appendix V

Calculations of Reagents for Flotations #1 and #2

All reagents were used at the same concentrations as in the "Britton Research Limited" report.

(a) H_2So_4 · (10% by weight)
- used at 2.0 lbs/ton = 2 -2000 = .001 (no units)

grams of reagent = .001 x 22.128 grams
= 22.13 grams

- but reagent (H_2SO_4) is at 10% by weight.

$$\text{wt of acid sol'n} = \frac{22.13 \text{ grams}}{0.1} = 221 \text{ grams of prepared acid}$$

$$C = \frac{1.84 \text{ (pure)}}{.1 + (.a)\ 1.84} = 1.0478 \text{ kg/1}$$

$$\text{Vol acid} = \frac{\text{mass of acid}}{\text{density of acid}} = \frac{.221}{1.0478} = .211 \text{ 1 or } 211 \text{ ml}$$

(b) Lime Ca(OH) (pure)
- used at 2.0 lbs/ton
same as in (a) = 22.13 grams

Note; the lime was added as needed to maintain the required pH of each flotation.

(c) $CuSO4\ .5H_2O$ (pure)
- used at .2 lbs/ton and .1bs/ton
- as in above -- 22.13 grams for 2.0 lb/ton
.21bs/ton - 22.13 - 10 = 2.213 grams pure $CuSO_4.5H_2O$.
- for .5 lbs/ton;

$$2.213 \text{ grams} \times \frac{.5}{.2} = 5.53 \text{ grams}$$

(d) Na Isobutyl (10% by weight)
- used at .04 lbs/ton and .05 lbs/ton.
.04 lbs/ton = .04 - 2000 = 2×10^{-5} (no units)
wt needed = $(2 \times 10^{-5}) \times 22{,}128$ = .44 grams
- assume C = 1.05 kg/1
- solution is 10% by weight

$$\text{Vol needed} = \frac{\text{Mass}}{\text{Density}} = \frac{.44}{1.05 \times 10^{-3}} = 4.2 \text{ ml}$$

for 0.5 lbs/ton;

$$\frac{.04}{4.2} = \frac{.05}{x} \quad x = 5.25 \text{ ml}$$

(e) MIBC (1%)
- used at .015 lbs/ton

.015 lbs/ton = .015 - 2000 = 7.5×10^{-6} (no units)
wt of = reagent (7.5×10^{-6}) 22, 128 = .166 grams of 100%
But the solids are at 10%, not 100%
We need 100 times the amount or 16.6 grams solution at 1%

(f) Aerofloat 25 (1%)
- need .025 lbs/ton
- assume C = 1 kg/l

.024 lbs/ton - .024 - 2000 = 1.2×10^{-5} (no units)
Grams of reagent needed = $1.2 \times 10^{-5} \times 22{,}128$ = .2655 grams at 100%
But our reagent is at 1%, not 100%
Grams of reagent needed = 100 x .2655 = 26.55 grams at 1%

wt of solids = 48.74 lbs
Pulp Density = 33%
Tank = 12" x 12" x 24"
Settings; Agitator Motor - 100
Air - 25 gf%

$Ca(OH)_2$ = pure
H SO4 = 10% b.w.
$CuSO_4 .5H_2O$ = pure
Na Isobutyl = 10% b.w.
Aerofloat = 1% b.w.
M.I.B.C. = 1% b.w.

Time	RPM	PH	% Air	Comments	Ca(OH)	H2SO4	CuSO4.5H2O	Na Isobuityl	Aerofloat 25	MIBC
(FI)										
10:40	80	7.9	5	Agitating Solution						
11:30	100	8.0	25							
11.31	100	8.0	25							3 ml
										3 ml
				- realgar coming off in float						3 ml 16.6 ml
										3 ml
										3 ml
11:32										1.6 ml
(FIA)										
11:39	100	10.4	25	- add lime for pH of 10.4 (froth thickened was added).	13.2					
11:42		10.7		- added lime	2.8					
11:45				- added CuSO4 .5H O			2.26			
11:47		10.6		- added Ca(OH)	2.5					
11:48		10.5		- froth loaded				4.2		
(FIB)										
11:53		10.5		- overflow looks like galena	1.5			4.2		16.6
11:55	100		25	- good overflow						
(FIC)										
12:00		9.9		- added xanthate which				4.2		
12:05		10.2		- added 1500 ml water						
12:09				- added 1500 ml water						
12:10		10.06								
12:12		10.47		- added lime increased frothing	1.7 gr.					
12:17		10.43		- lime thickened froth	1.0 gr.					
(F2)		9.2								
12:22		8.8		- add acid		211 ml	5.59 gr			
12:24				- add 1200 ml of water						
12:26		7.6					5.5 gr			
12:28		7.2		- add acid for lower pH		50 ml				

(F2D)

Time	RPM	PH	% Air	Comments	Ca(OH)	H2SO4	$CuSO_4.5H_2O$	NaIsobuityl	Aerofloat 25	MIBC
(F2D)										
12:29		7.1					31 ml	5.25 ml		
12:30		7.06					31 ml	5.25 ml		
12:32				- froth overflowing						
(F2E)										
12:37	100	7.45	25	- added 1500 ml water				5.25 ml		
12:38		7.1		- added acid		35 ml				
(F2F)										
12:45		7.43		- added 750 ml water - much smaller bubbles				5.25 ml	26.66 ml	
12:50		7.58		- added 1900 ml water						
12:58		7.64	20	- added 750 ml water - slurry turning very light coloured						
12:59				- stopped flotation						
TOTAL REAGENTS USED				9,100 ml water	22.7 gr	327ml	13.35 gr	28.35 ml	26.55 ml	33.2 ml

Appendix VII

Flotation # 3

Tank = 12" x 12" x 24"
Agitator Motor = 100
Air = 25 gf %
Feed = tails of Flotation #2

N Isobutyl (10% b.w.)
MIBC (1% b.w.)
Na_2S (60% b.w.) -- mixed 10 grams in 335 ml water
$Ca(OH)_2$ (pure)

Time	RPM	PH	% Air	Comments	Ca(OH)2	Na2S	Na Isobutyl	M.I.B.C.
1:08	100	7.71	25					
1:10		9.06		Added Na_2S	2.5 grams	200 ml		
1:13		9.9		Added Na_2S		135 ml		
1:14				Added other reagents			1 ml	1 ml
1:16							3 ml	3 ml
1:18		9.4						
1:21		9.35		Added 200 ml water				
1:29				Added 200 ml water				
1:35		9:15		Stopped Flotation				
TOTAL REAGENTS USED				4000 ml water	2.5 grams	335 ml or 10 grams of 60% b.w.	4 ml	4 ml

Note: Remaining slurry was 22" in height

CLEANER FLOTATION APPENDIX VIII

Tank = 12" x 12" x 24" lbs
Agitator Motor = 75
Air = 25 gf %

$Ca(OH)_2$ = pure
Feed - Flotation #1 concentrate (Ag, Pb, Zn, Sb)
pH = 11.6 to 11.9

Time	RPM	PH	% Air	Comments	Lime ($Ca(OH)_2$)	
4:03	75	9.04	25	Thin froth		
4:06	75	11.6	25		1.4 grams	
4:10		11.9			1.3 grams	realgar and stibnite were seen in the Cleaner Flotation
4:17	75	11.76	7.5	Thick Froth		tails under the binocularscope
4:22				Overflow started		
4:24	75	11:68	10			
4:28	75	11:58	6			
4:31		11:46				
4:32		11.8			.9 grams	
4:39		11.6		Overflow stopped		
TOTAL REAGENTS USED				Water continually supplied from tap.	3.6 grams	

Calculation of Weight of Cleaner Flotation Tails

From the tails a slurry cut was taken for assay sample CF186-6. From this cut the total weight the total weight of the tails can be calculated. The tails were then mixed with the Flotation #2 concentrate to be used as feed for Flotation #4.

Knowns;

- slurry cut weighed 302.4 grams wet
- dry weight of cut = 80.7 grams
- volume of cut = 225 ml
- 1 litre = .03531 cu. ft.
- volume of slurry holding tank = 11 3/4" diameter x 1" depth.

Volume of slurry = depth x ¶r
= 1" x 3.14 $(5.875)^2$
= 108.4 cu inches or .061 cu ft

Total volume of slurry = .061 cu ft + 5 litres + 2000 ml
= .061 + .1765 + .071
= .309 cu ft -- .309 - .03531 = 8.75 1

$$C_{\text{of slurry}} = \frac{C\ \text{dry}}{\frac{\text{wt solids}}{\text{wt slurry}} + \left(1 - \frac{\text{wt solids}}{\text{wt slurry}}\right) C\ \text{dry}}$$

$$= \frac{4}{.267 + (1 - .267)\ 4} = 1.25\ \text{kg/l}$$

Mass of slurry = Density x Volume
= 1.25 x 8.75
= 10.94 kg

$$\text{Wt of dry solids} = \text{wt of slurry} \times \frac{\text{wt of solids sample}}{\text{wt of slurry sample}}$$

$$= 10.94 \times \frac{80.7}{302.4} = 2.92\ \text{kg}$$

- Cleaner Flotation Tails are 2.92 kg in weight
  2920 - 80.7 = 2.84 kg added to flotation #4 slurry

Appendix VIV

Flotation #4 Calculations and Data

To form a slurry for this flotation the estimated weight of the dry solids for the feed was required. The feed is coming from (a) Flotation #2 concentrate (b) Cleaner Flotation #1 tails.

From Appendix VIII we can get that (b) = 2.84kg dry solids.
Need weight of (a)

Known;

Volume of Flotation #2 concentrate holding tank = 4.75" x 11.75 dia.
slurry sample = 312.8 grams wet
slurry sample = 112.6 grams dry
assumed C dry = 4

$$\begin{aligned}\text{Volume of slurry (a)} &= \pi R \times \text{depth} \\ &= 3.14 \times 5.875 \times 4.75 \\ &= 513.925 \text{ cu inches} \\ &= .2974 \text{ cu ft} \text{ -- - } .03531 = 8.42 \text{ liters}\end{aligned}$$

$$\underset{\text{(of slurry)}}{\text{C kg/l}} = \frac{\text{C dry}}{\frac{\text{wt solids}}{\text{wt slurry}} + \left(1 - \frac{\text{wt solids}}{\text{wt slurry}}\right)\text{C dry}}$$

$$= \frac{4}{\frac{112.6}{312.8} + \left(1 - \frac{112.6}{312.8}\right) 4} = 1.37 \text{ kg/l}$$

$$\begin{aligned}\underset{\text{of slurry}}{\text{Mass}} &= \text{Density} \times \text{Volume} \\ &= 1.37 \times 8.42 \\ &= 4.15 \text{ kg}\end{aligned}$$

$$\underset{\text{solids}}{\text{Mass of}} = \underset{\text{slurry}}{\text{Mass of}} \times \frac{\text{wt of solids}}{\text{wt of slurry}}$$

$$\begin{aligned}\text{Total solids} &= \text{solids (a)} + \text{solids (b)} \\ &= 2.84 + 1.5 \\ &= 4.64 \text{ kg}\end{aligned}$$

This total weight of solids was needed in order to determine the amount of the reagents needed for the flotation #4.

*Note: In calculating the total solids weight before the flotation, a mathematical error was made which gave the total solids a weight of 7.07 kg rather than the 4.64 kg shown above. The reagents for the test were calculated using the 7.07 kg. After flotation #4 was completed, the tails and concentrate (see later) were dried and weighed so the true weight is known after the fact. The true weight is 6.399 kg. Therefore the 7.07 kg weight was the better value to be used in the calculations for of the reagents. A 10% oversupply of reagents was therefore used.

Calculation of Reagents (Using 7.07 kg as wt of solids)

Pot. Permanganate	=	.36 grams/kg of solids (added dry)
Zanthate	=	.24 grams/kg of solids (mixed H_2O)
Dowthroth	=	.34 grams/kg of solids (mixed in H_2O)

Pot. Permanganate	= 2.55 grams
Zanthate	= 1.69 grams
Dowthroth	= 2.40 grams

Flotation #4 Procedure

- the pH was kept at 8.0
- the agitator motor was set at 2500 rpm
- the slurry was 1" below lip of column

(a) agitated for 4 minutes
(b) air rated for 135 minutes (had problems of reagents of flotation #1 and #2 causing a flotation during this air rating period.)
(c) added 2.55 grams of $kMnO4$ and waited 49 minutes.
(d) added 1.69 grams of Na Isobutyl (Xanthate) and 2.4 grams of Dowfroth and floated for 30 minutes.

Both the concentrate and tails were dried, weighed, and assay samples were rifted. Also a sample of each was sent for a polished section study.

	Total Weight	Assay wt	Polished Sect. wt
Aspy (tails) FTA86-9	2459.5 grams	39.8 grams	1.9 grams
Py (concentrate) FCP86-8	3939.0 grams	79.08 grams	11.1 grams

Appendix X

Assay Samples

GC86-3

- 14.71 grams
- total assay
- assay for Au, Ag, Pb
- small column gravity-two concentrate

G86T-1

- 21.64 grams
- rifted sample from gravity-one concentrate after gravity-two concentrate was removed (i.e. gravity-two tails)

FF86-2

- 170 gram (dry)
- sample of flotation #1 feed
- sample taken from agitated slurry

FT86-4

- 208.51 grams
- tails of flotation #3 (Cyanide Feed)
- sample cut from agitated slurry

CF186-6

- 80.7 grams dry
- tails of cleaner flotation of flotation #1 concentrate
- sample cut from agitated slurry

CF186-7

- 26.94 grams dry
- concentrate of cleaner flotation of flotation #1 concentrate

F286-5

- 112.6 grams dry
- sample of Flotation #2 concentrate
- sample cut from agitated slurry
- used as feed for Py/Aspy separation

FTA86-9

- arsenopytite tails of flotation #4
- 39.8 grams
- rift off dried sample

FCP86-8

- pyrite concentrate of flotation #4
- 76.08 grams
- rift off dried sample

<u>FT86-4</u> (Cyanide residue)
- 65 grams
- tails from cyanide leach

<u>FT86-4</u>
- bottle of solution
- taken from cyanide leach solution

<u>F3C86-10</u>
- 47.52 grams
- flotation #3 Concentrate
- rifted from dried solids

Appendix XI

PETROGRAPHIC STUDY OF PROCHEM SAMPLES

FCP 86-8, FTA 86-9 AND GC 86-3

K.A.H.
REC'D DEC 22 1986
M.L.
KJF.
GOS

December 18, 1986
Prepared by:
Brent McInnes

PURPOSE

The purpose of this study was to:

A) identify particles of free gold and associated minerals in order to establish the best gold recovery method
B) identify the metallic minerals and their relative percentages to assess the pyrite/arsenopyrite separation process
C) observe mineral intergrowths and textures in order to constrain the environment of precious metal deposition

METHOD

Samples FTA 86-9, FCP 86-8 and GC 86-3 were mounted on a glass slide and polished on one side to enable observation by reflected light microscopy. Two sections were made of FTA 86-9 and FCP 86-8 each to insure maximum sample use, and one of GC 86-3.

Modal percentage analysis was conducted using a grid system point counting scheme. Five hundred grains were counted on each slide, for a total of one thousand grains per sample. Statistically, this method has a probable error of less than 5%, and is quite effective in accurately estimating modal percentages of minerals.

During the point counting, notes were made and pictures taken of interesting mineral textures. Gold was carefully searched for at medium power (200X) and at high power (500X). It is possible to distinguish gold grains as small as 10 microns when in contact with another mineral and 25 microns when it occurs as singular grains. No gold grains were detected in any of the samples studied.

RESULTS

The following are the results of the point counting:

FCP 86-8 (Pyrite Concentrate)

	PYRITE	ARSENOPYRITE	GALENA	SPHALERITE	CPY	OTHER
FCP-8A	422	20	21	34	2	1
FCP-8B	418	19	24	35	2	2
TOTAL(%)	84.0	3.9	4.5	6.9	.4	.3

FTA 86-9 (Arsenopyrite Concentrate)

	PYRITE	ARSENOPYRITE	GALENA	SPHALERITE	CPY	OTHER
FTA-9A	275	143	17	16	3	46
FTA-9B	295	138	18	25	2	23
TOTAL(%)	57.0	28.1	3.5	4.1	.5	6.9

Marcasite was included with the pyrite during the point counting, however the modal abundance of marcasite does not exceed 1% in either sample. The category "OTHER" refers to non-metallic minerals, generally quartz and quartz-silicate (gangue) intergrowths, and Fe-oxides.

A total of 29 grains are present in GC 86-3. They are as follows:

GALENA	5
PYRITE	11
ARSENOPYRITE	4
GANGUE	6
SPHALERITE	1
FE-OXIDE	1
MAGNETITE	1

Rough notes made during identification are provided for your reference.

TEXTURES

Some of the common textures that occur in the samples have been photographed and appear in the following pages.

Pyrite and arsenopyrite are common constituents of all the samples. These minerals often appear fractured, some of which probably occurred during sample preparation, but other evidence such as subgraining and replacement along fractures suggest that some of the deformation may have been the result of faulting during ore emplacement. Pyrite and arsenopyrite are often intergrown and therefore obtaining pure separates of these minerals may be difficult (see photos 6, 7 and 8). Pyrite commonly contains inclusions of other minerals (see 1 & 2), generally galena, pyrrhotite and chalcopyrite, suggesting that all constituents were being precipitated simultaneously from solution.

Sphalerite occurs in both samples, often associated with galena or chalcopyrite. Internal reflections in sphalerite are reddish-brown (see 5), indicative of a Fe-rich variety commonly known as blackjack sphalerite. Some sphalerites have tiny blebs of chalcopyrite within them, a texture known as chalcopyrite disease. This phenomenon is an unmixing caused by the mineral separation of chalcopyrite from sphalerite at a lower temperature

... the higher temperature Cu-Fe-Zn sulphide precursor precipitated at (see 3 &4).

Marcasite (see 5) often appears disaggregated and is preferentially replaced by Fe-oxides along parallel cleavage planes. Chalcopyrite also replaces marcasite.

The presence of amorphous Fe-oxides (goethite) and replacement rims of chalcocite and covellite auggist that the samples have been partially exposed to surface waters and are from a supergene zone of alteration.

All minerals were observed intergrown with each other, therefore they probably represent a coeval suite, with no late stage depositional event.

CONCLUSIONS

1. No particles of free gold were observed during the study. This is not uncommon and has frustrated many researchers working on precious metal systems. An alternative route to identify the presence of gold would be to retry the technique used in this study on a larger heavy mineral concentrate, or to use a scanning electron microscope equipped with an energy dispersive unit.

2. The pyrite separate FCP 86-8 contained 84% pyrite and 4% arsenopyrite. The arsenopyrite separate FTA 86-9 contained 57% pyrite and 28% arsenopyrite. As pyrite and arsenopyrite are commonly intergrown a relatively "pure" separate may not be possible unless the samples are crushed to a finer grain size.

3. Mineralogically, the deposit contains a mineral suite similar to many of the known gold deposits in the Dawson Range. Deformation textures are common, and are probably the result of tectonic disturbance during the mineralization event.

Photo 1. Large grain of pyrite (P) containing inclusions of brownish pyrrhotite (Po) and yellow chalcopyrite. Smaller grain below shows galena remobilized into fractures within pyrite. Field of view is approx. 1 mm wide.

Photo 2. Large grain consisting of pyrite, galena and sphalerite. Note inclusions of galena and sphalerite within pyrite suggesting coeval precipitation. The two lower flanking grains are galena. Field of view = .8 mm wide.

Photo 3. Sphalerite (S) and galena (G) intergrowth with small blebs of chalcopyrite within sphalerite. This texture is referred to as chalcopyrite disease. Small gray minerals within sphalerite are Fe-oxides. Field of view = .5mm wide.

Photo 4. Two grains of sphalerite, one with chalcopyrite disease, the other massive. Galena at top. Field of view = 1 mm.

Photo 5. Two grains of marcasite (M) with other grains of arsenopyrite (A), pyrite (P) and sphalerite (S). Marcasite has well developed parallel cleavage where surface water has penetrated and reacted to form amorphous Fe-oxides. Chalcopyrite has replaced the right hand marcasite grain before oxidation. Grain of sphalerite shows reddish-brown internal reflections indicating Fe- rich composition. Field of view = .8 mm wide.

PHOTOS 6-9 ARE FROM GCA 86-3

Photo 6. Fine grained aspy mass with occasional grain of yellow pyrite. Grain is 0.7 mm long.

Photo 7. Arsenopyrite - pyrite intergrowth. The three grains are 1.2 mm by 0.7 mm long.

Photo 8. Aggregate of arsenopyrite, pyrite and quartz showing well developed crystal faces indicative of growth in free space. Grain to left consists of silicate minerals intergrown with magnetite (M). Dimensions of grain are .8 mm X .6 mm.

Photo 9. A very fine grained assemblage of pyrite with about 10 % arsenopyrite. Grain dimensions = 1.5 mm X .8 mm, ave. gn. size =

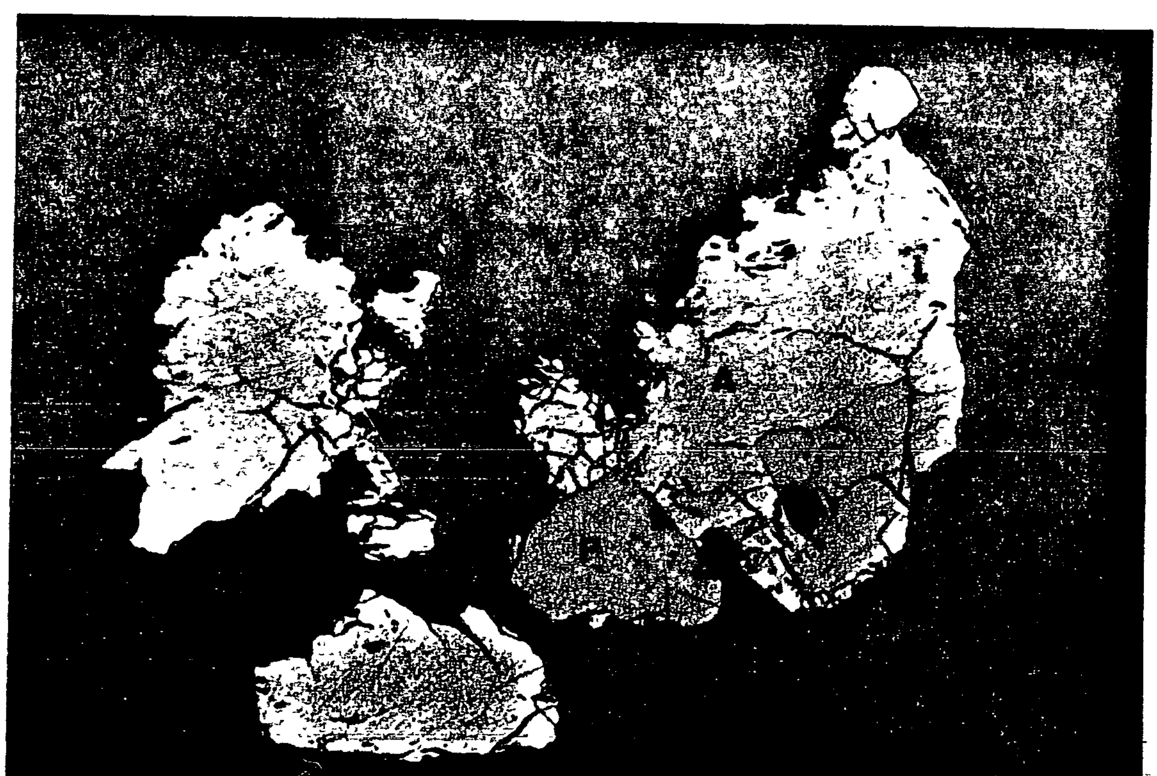
1

2

5

6

7



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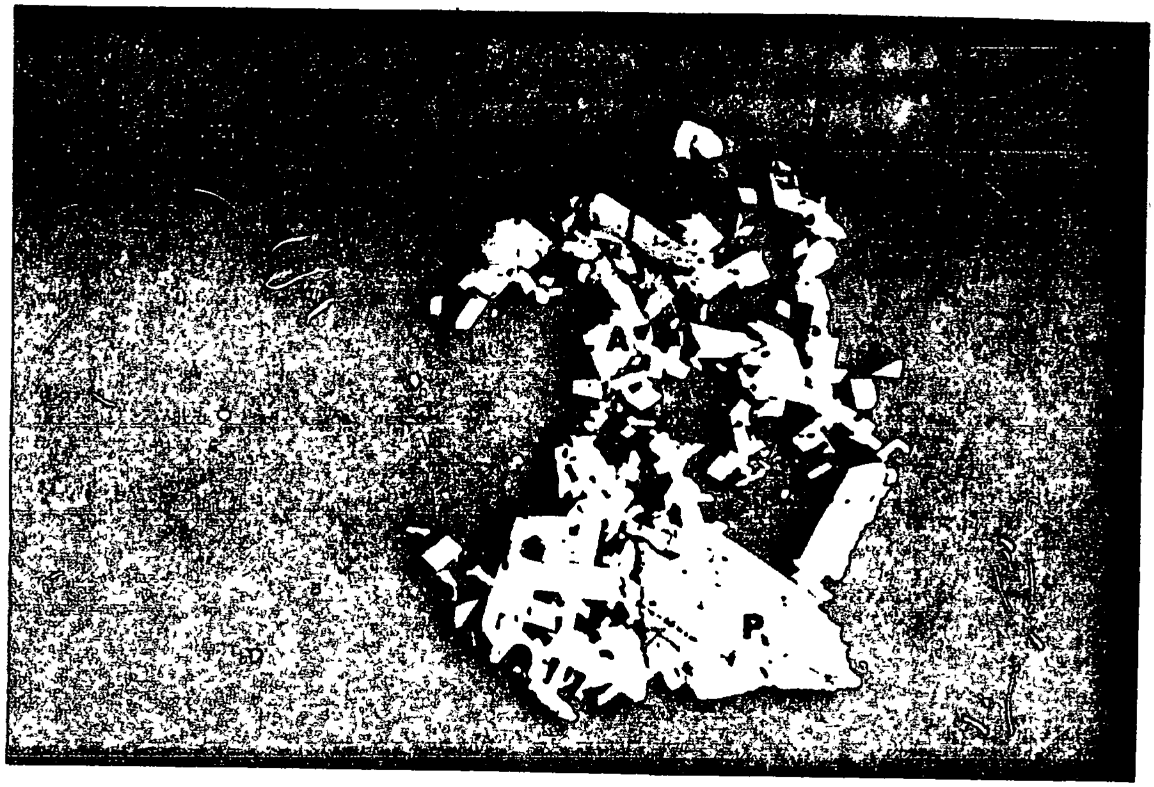

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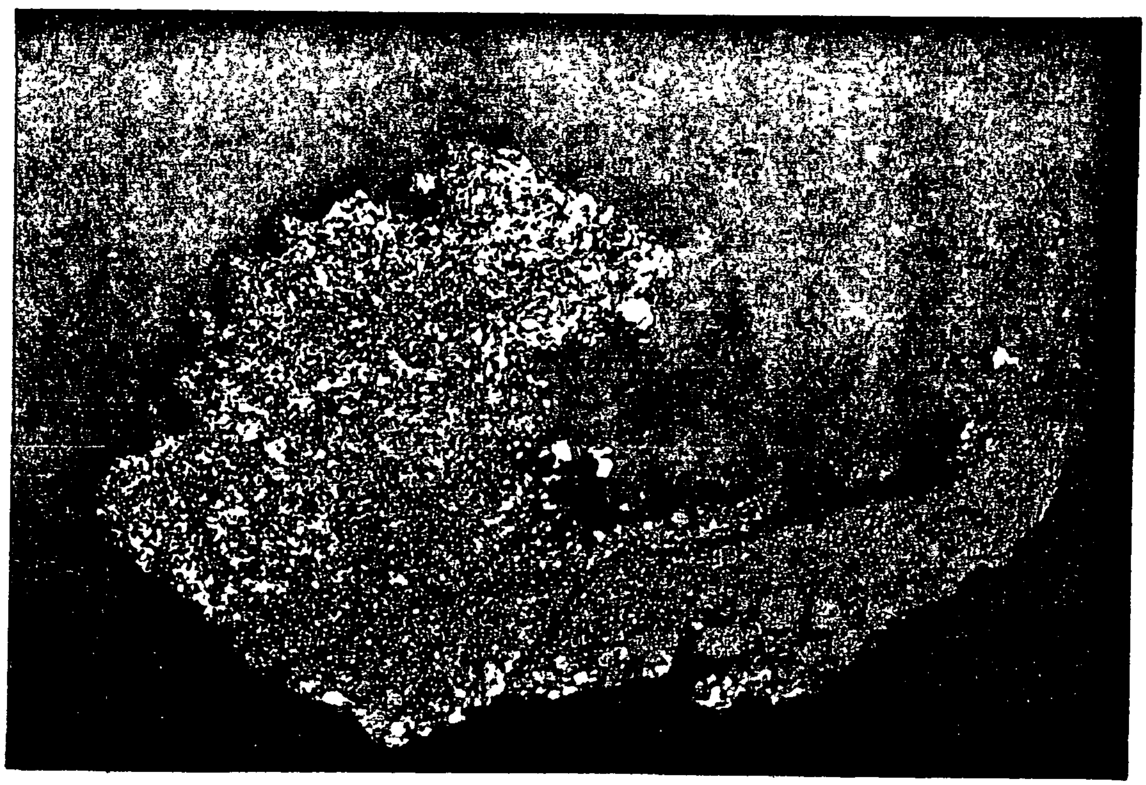


The following is a breakdown of costs incurred during preparation of this report. Cost per hour is $20.00.

Mineral Identification		2
Point Counting		7
Photography		1.5
Report preparation		3.5
TOTAL		14.0 hours
Labour costs	14 X $20 =	$280.00
Photo processing		$ 10.64
TOTAL COST		$290.64

Brian McInnes

Dec 18, 1986

CASH TOTAL
AMOUNT TENDERED 20.00
CHANGE

B. McINNES
DEPARTMENT OF GEOLOGY
McMASTER UNIVERSITY
HAMILTON, ONTARIO, CANADA

APPENDIX XII - ASSAYS

of Analysis

REPORT: 416-5306

PROJECT: Y86 PAGE 1

SAMPLE NUMBER	ELEMENT UNITS	Au O/T	Ag O/T	Pb PCT	Zn PCT	As PCT	Sb PCT	Cu PCT	C tot. PCT	C flx. PCT
GC 86-3		7.740	31.02							
C 86T-1		0.264	5.92							
FF 86-2		0.125	4.34	3.04	1.17	1.98	0.06		5.10	3.52
F2 86-7		0.319	6.88	3.80	2.44	5.82	0.07			
CF 186-7		0.762	46.09	29.09	14.52	0.67	0.39	1.09		
CF 186-6		0.242	13.96	7.73	2.12	1.73	0.07			
FTA 86-9		0.324	5.82	2.88	1.64	6.48	0.07			
FC[illegible] 86-8		0.225	9.04	6.16	2.48	3.10	0.07			
FT 86-4		0.033	0.70						3.10	0.27
FT 86-4 CYANIDE		0.010	0.28							
V 86-1		0.248	16.02							

Chief Chemist

of Analysis

PORT# 416-5306

PROJECT# YB6 PAGE 1

SAMPLE NUMBER	ELEMENT UNITS	Au O/T	Ag O/T	Pb PCT	Zn PCT	As PCT	Sb PCT	Cu PCT	C tot PCT	C Fix PCT
GC 86-3		7.740	31.02							
C 86T-1		0.264	5.92							
FF 86-2		0.125	4.34	3.04	1.17	1.98	0.06		5.10	3.52
F2 86-7		0.310	6.88	3.80	2.44	5.82	0.07			
CF 186-7		0.762	46.09	29.09	14.52	0.67	0.39	1.09		
CF 186-6		0.242	13.96	7.73	2.12	1.73	0.07			
FTA 86-9		0.324	5.82	2.88	1.64	6.48	0.07			
FCP 86-8		0.225	9.04	6.16	2.48	3.10	0.07			
FT 86-4		0.033	0.70						3.10	0.27
FT 86-4 CYANIDE		0.010	0.28							
V 86-1		0.248	16.02							

Chief Chemist

SAMPLE	AG OZ/TON	SB %	PB %
F3C 86-10	3.53	0.06	2.76

XRAL

SAMPLE	AU OZ/TON	ZN %	AS %
F3C 86-10	1.500	0.32	3.62

BONDAR-CLEGG

Certificate of Analysis

REPORT: 416-5301

PROJECT: Y86 PAGE 1

SAMPLE NUMBER	ELEMENT UNITS	Au PPM	Ag PPM	
ET 86-4 FILTRATE		6.05	9.3	by neutron act.
		.38	5.3	Nitric Acid Only
		.25		aquaregia solution (Ag not possible.)

Used 0.25 ppm Au & 5.3 ppm Ag for calculation as the gave better metal balance.

Chief Chemist

Certificate of Analysis

REPORT: 416-5307

PROJECT: 10A PAGE 1

SAMPLE NUMBER	ELEMENT UNITS	Au PPM	Ag PPM
ET 86-4 FILTRATE		0.25	5.3

Chief Chemist

	Zn %	As %	Sb %	Pb	Au	Ag
F3C 86-10	.32	3.62	.06	2.76		

8032 TORBRAM RD, BRAMPTON, ONT.

Carrier	
☐ ppd. ☐ col.	
WB#	
PROJECT#	
P.O.#	
SERVICE AGREEMENT#	

The samples of 4 listed below are to be analysed by: ☐ GEOCHEMICAL METHODS
: ☑ ASSESSMENT ASSAY METHODS

for the elements circled and/or as indicated in the listing. Report to: 1 ☑ Head office, 2 ☐ Field office, 3 ☐ Other (please specify) ______

Invoice to 1, 2 or 3. Unused materials: ☐ return to 1, 2 or 3, ☐ store at cost ____ days, ☐ discard after 90 days.

AUTHORIZED BY: [signature] DATE Dec. 15, 1986

Cu Pb Zn Ag Cd Ni Co Fe Mn Mo Au As Sb B U Th SnW S F Ba Pt Pd Whole Rock Analysis-(Si Al Ca Mg Na K Fe Mn Cr Ti P Sr Rb Zr LOI) 39 El.

SAMPLE NUMBER	Au	Ag	SAMPLE NUMBER		
			Au	Ag	
Yukon 2 - ~~HEAD~~	✓	✓	0.11 } 0.105	4.41 }	4.485 = 4.49
VEIN - C286 - 1	✓	✓	.10	4.56	
Yukon 2 - 48 Hr Carbon	✓	✓	3.48	40.90	
Yukon 2 - 115 Hr Carbon	✓	✓	0.12	4.05	

Au & Ag by Fire Assay
48 Hr RUSH

Due 17th 8 am

8031 TORBRAM RD. BRAMPTON, ONT.

The samples of 4 listed below are to be analyzed by: ☐ GEOCHEMICAL METHODS
: ☑ ASSESSMENT ASSAY METHODS

for the elements circled and/or as indicated in the listing. Report to: 1 ☑ Head office, 2 ☐ Field office,
3 ☐ Other (please specify)
Invoice to 1, 2 or 3. Unused materials: ☐ return to 1, 2 or 3, ☐ store at cost ____ days, ☐ discard after 90 days.
AUTHORIZED BY [signature] DATE DEC 22, 1986

Carrier
☐ ppd. ☐ col.
WB#
PROJECT#
P.O.#
SERVICE AGREEMENT#

Cu Pb Zn Ag Cd Ni Co Fe Mn Mo Au As Sb B U Th SnW S F Ba Pt Pd Whole Rock Analysis-(Si Al Ca Mg Na K Fe Mn Cr Ti P Sr Rb Zr LOI) 30 El

SAMPLE NUMBER	Au	Ag	SAMPLE NUMBER Au	Ag
YUKON 2- FILTRATE	✓	✓	N.L	NIL
YUKON 2- RESIDUE	✓	✓	.061	2.32
YUKON 2-V2-RESIDUE	✓	✓	.049	2.53
YUKON 2- FIN. ST. CARBON	✓	✓	.089	10.1

.055 2.43

Jan 0, '87

Au by Fire Assay
Normal Delivery.

SAMPLE	AU OZ/TON	AG OZ/TON
VEIN-C286-1	0.100	4.56
YUKON2-HEAD	0.110	4.41
YUKON2-48C	3.480	40.90
YUKON2-115C	0.120	4.05

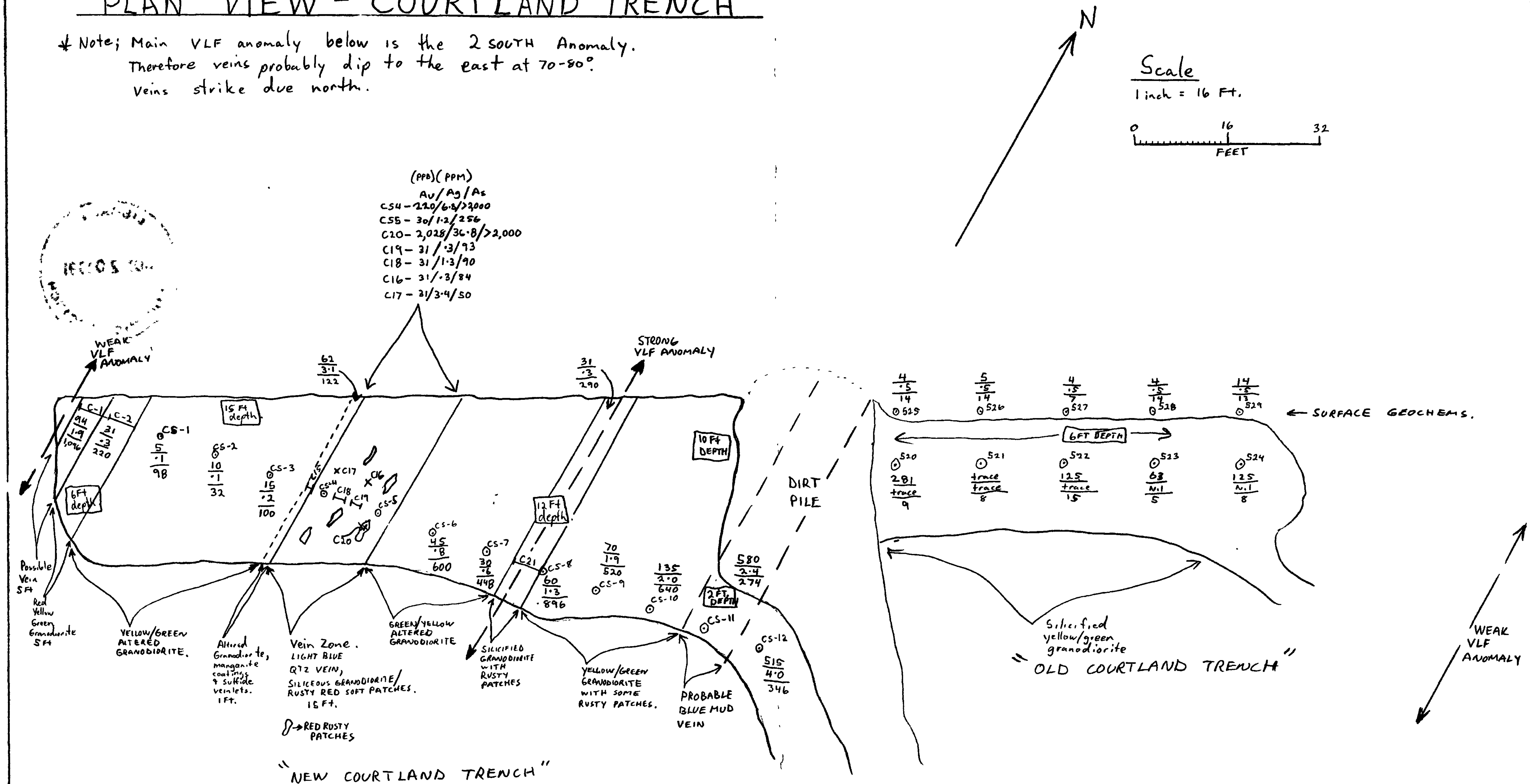
SAMPLE	AU OZ/TON	AG OZ/TON
YUKON-2- FILTRATE	NIL	NIL
YUKON-2- RES.	0.061	2.32
YUKON-2- VZ-RES.	0.049	2.53
YUKON-2- FIN.ST.CAR.	0.089	10.10

TEST & PRODUCT	Assays: Wt %	Au oz/t	Ag oz/t	Pb %	Zn %	As %	S6 %	Cu %	Total C %	C (NOCO3) %	Distribution: Au	Ag	Pb	Zn	As	Sb	Individual Distribution %: Au	Ag	Pb	Zn	As	Sb	Wt %
GRAVITY																							
Head Gravity	100	109	4 16														100	100					100
Gravity-One Tails	86 66	086	3 9														64 8	80 8					86.66
Gravity-Two Tails	13 3	264	5 92														32 2	18 9					13 3
Gravity-Two Concentrate	.043	7 74	31 02														3 0	3					043
Gravity-One Concentrate	13 7	28	6 00														35 2	19 2					
Head	100	109	4 16								100 0	100 0					100	100					
Gravity Concentrate	043	7.74	31 02								3 0	3					3 0	3					
Flotation #1 Feed	99 96	125	4.34	3 04	1 17	1 98	06		5 1	3 52	97 0	99 7	100	100	100	100	100	100	100	100	100	100	100
Flotation #1 Concentrate	16 8	337	19 84	9 63	4.39	1 53	13				44 1	76 8	53 4	63 2	13 0	36 5	45 5	77 0	53 4	63 2	13 0	36 5	16 8
Flotation #1 Tails	82 8	082	1 17	1 72	52	2 09	05				52 4	22 1	46 6	36 6	86 9	68 6	54 0	22 2	46.6	36 6	86 9	68 6	82 8
Cleaner Flotation Feed	16.8	337	19 84	9.63	4 39	1 53	13				44 1	76 8	53 4	63 2	13 0	36 5	100	100	100	100	100	100	100
Cleaner Flotation #1 Tails	13 7	242	13 95	7.60	2 12	1 73	07				25 8	44 1	34 4	24 9	12 0	16 0	58 5	57 4	64 4	39 4	92.3	43 8	
Cleaner Flotation #1 Concentrate	3 1	762	46 09	18 68	14 52	67	39	1 09			*18 3	32 7	12.3	38 4	1 0	20 1	41 5	42 6	23 0	60 8	7 7	55 1	18 5
Flotation #2 Feed	82 3	082	1 17	1 72	52	2 09	050				52 4	22 1	46 6	36 6	86 9	68.6	100	100	100	100	100	100	100
Flotation #2 Concentrate	16 2	310	6 88	3 80	2 44	5 82	070				39 1	25 6	20 3	33 9	47 7	19 0	74 6	115 8	43 6	92 6	54 9	27 7	19 7
Flotation #2 Tails	66 1	026	85	1 21	.05	1 17	045				13 31	12 9	26 3	2 8	39 1	49 6	25 4	58 4	56.4	7 7	45 0	72 3	80 3
Flotation #4 Feed	30 1	266	8 79	4 90	2 16	4 40	.07				62 2	60 8	48 6	55 6	66 9	35 1	100	100	100	100	100	100	100
Flotation #4 Concentrate	18 5	225	9 04	6 16	2 48	3 10	07				*32 4	38 5	37 6	39.3	12 2	21 6	52 1	63 3	77 4	70 7	43 5	61 5	61 5
Flotation #4 Tails	11 6	324	5 82	2 88	1 64	6 48	07				*29 1	15 5	11 0	16.2	37 8	13 5	46 8	25 5	22 5	29 1	56 5	38 5	38 5
Flotation #3 Feed	66 1	109	85	1 21	05	1 17	.045				55 9	12 9	26 3	2 8	39 1	49 6	100	100	100	100	100	100	100
Flotation #3 Concentrate	3 5	1 50	3 53	2.76	32	3 62	06				*39 9	2 8	3 1	0 9	6 3	3 4	71 4	21 7	11 8	32.1	16 1	6 9	5 3
Flotation #3 Tails	62 6	033	7	1 13	04	1 04	044		3 10	27	16 1	10 1	23 3	2.1	32 9	46 0	28 8	78.3	88 6	75 0	84 1	92.7	
Total Recovered	21 7										122 7	89 8	64 0	94 8	57 3	58 6							
Cyanide Feed	62 6	033	7	1 13	04	1 04	044	3 10	3 10	27	16 1	10 1	NaCn Consumption = 7 9 lb/t dry solids				100	100					62 6
Cyanide Residue	62 6	010	28								4 8	4 0	Ca(OH)2 Consumption = 89 lb/t dry solids				29 8	39 6					62 6
Cyanide Filtrate		25ppm	9 3ppm														70-75	60-70%					

Note A line under the assay indicates calculated assay

PLAN VIEW - COURTLAND TRENCH

* Note; Main VLF anomaly below is the 2 SOUTH Anomaly. Therefore veins probably dip to the east at 70-80°. Veins strike due north.



AURCHEM RESOURCE'S LTD.	
DISCOVERY CREEK CLAIMS	YUKON 115 I/3
OCT 9, 1986.	BY M. LANGDON.

622
.1
YE1A
86-026
v.3

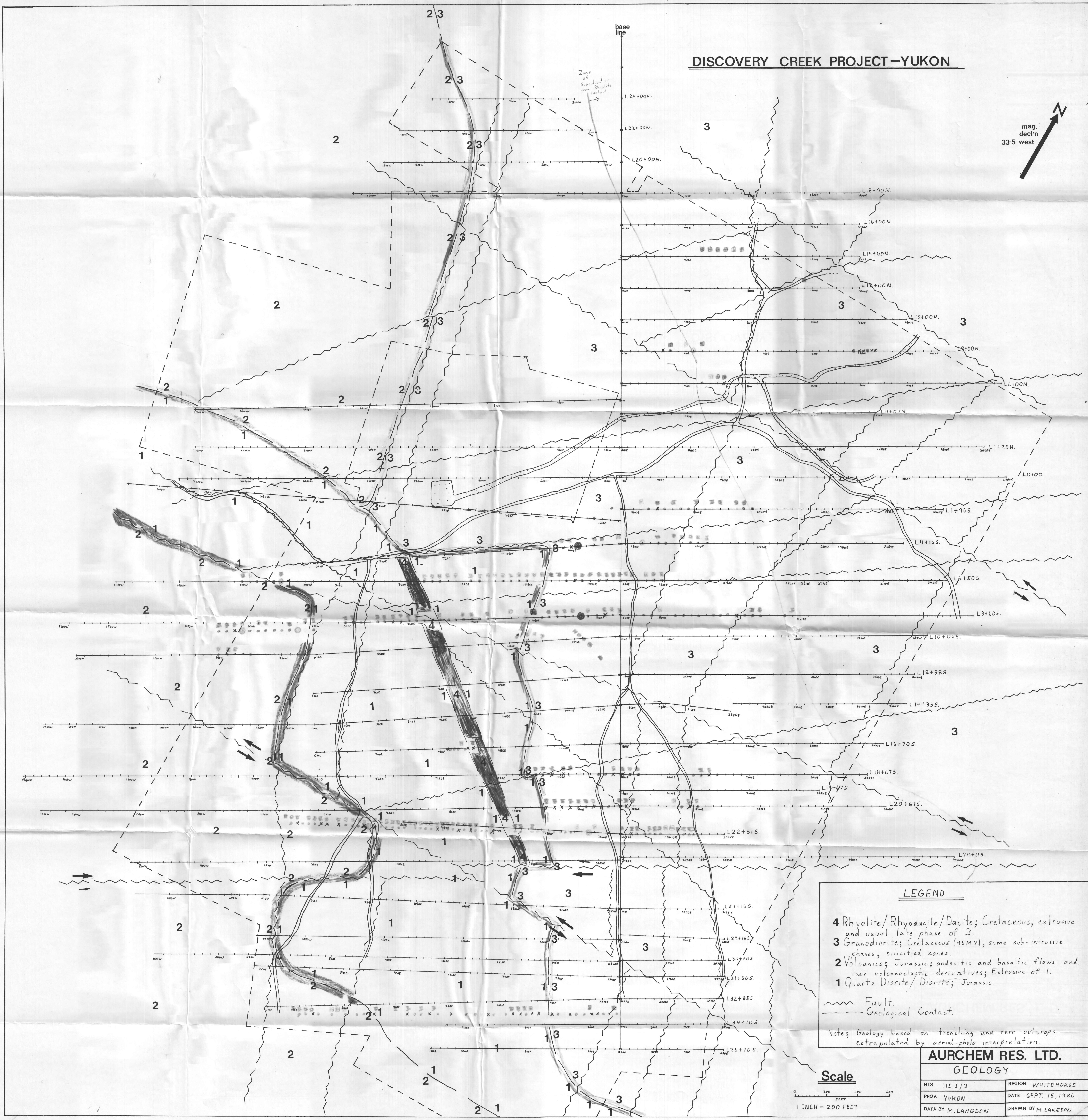
86-026

COURTLAND
TRENCH

LIBRARY
JUN 20 1991
ECONOMIC RESEARCH

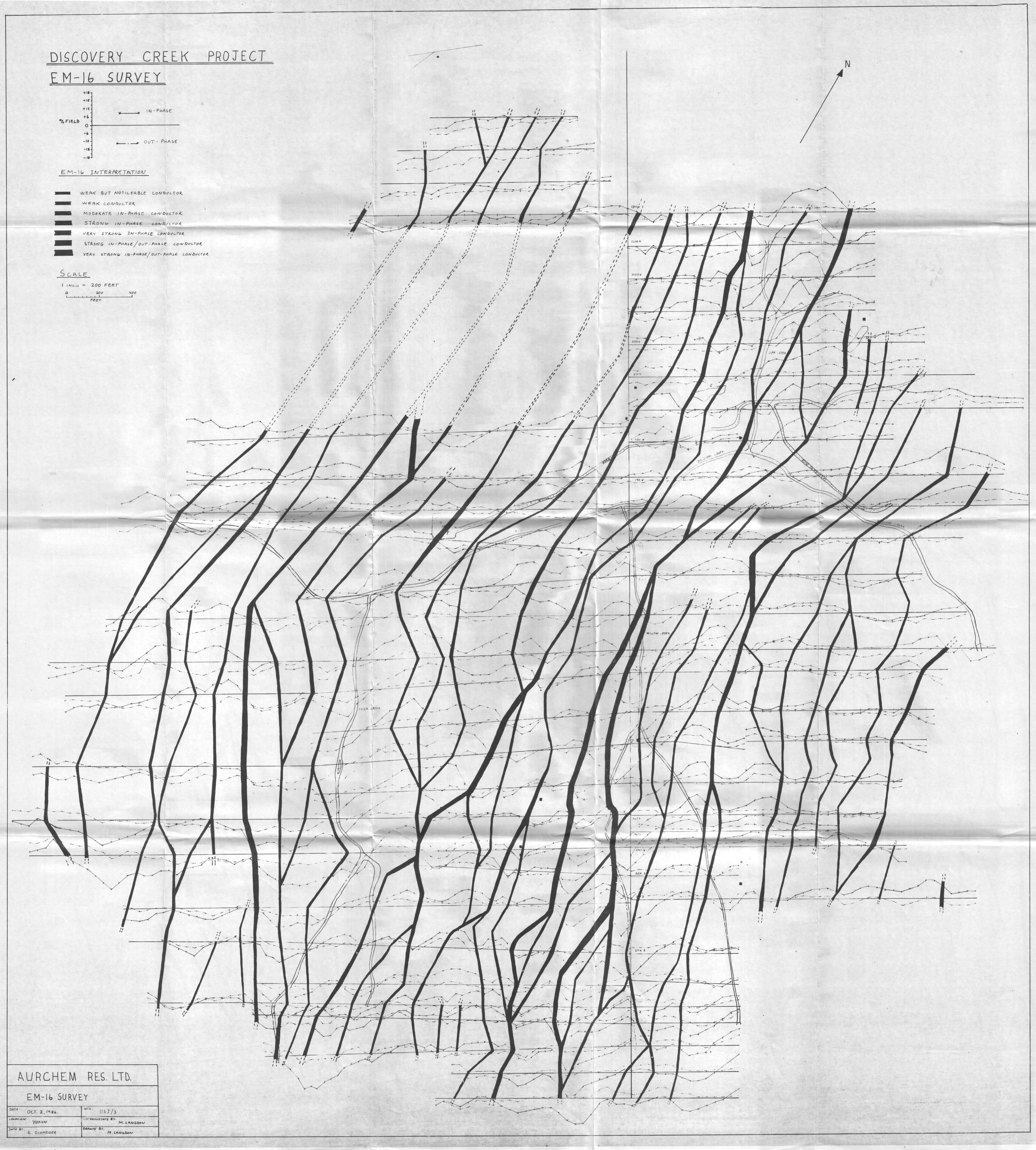
Schematic

Cross-Section.

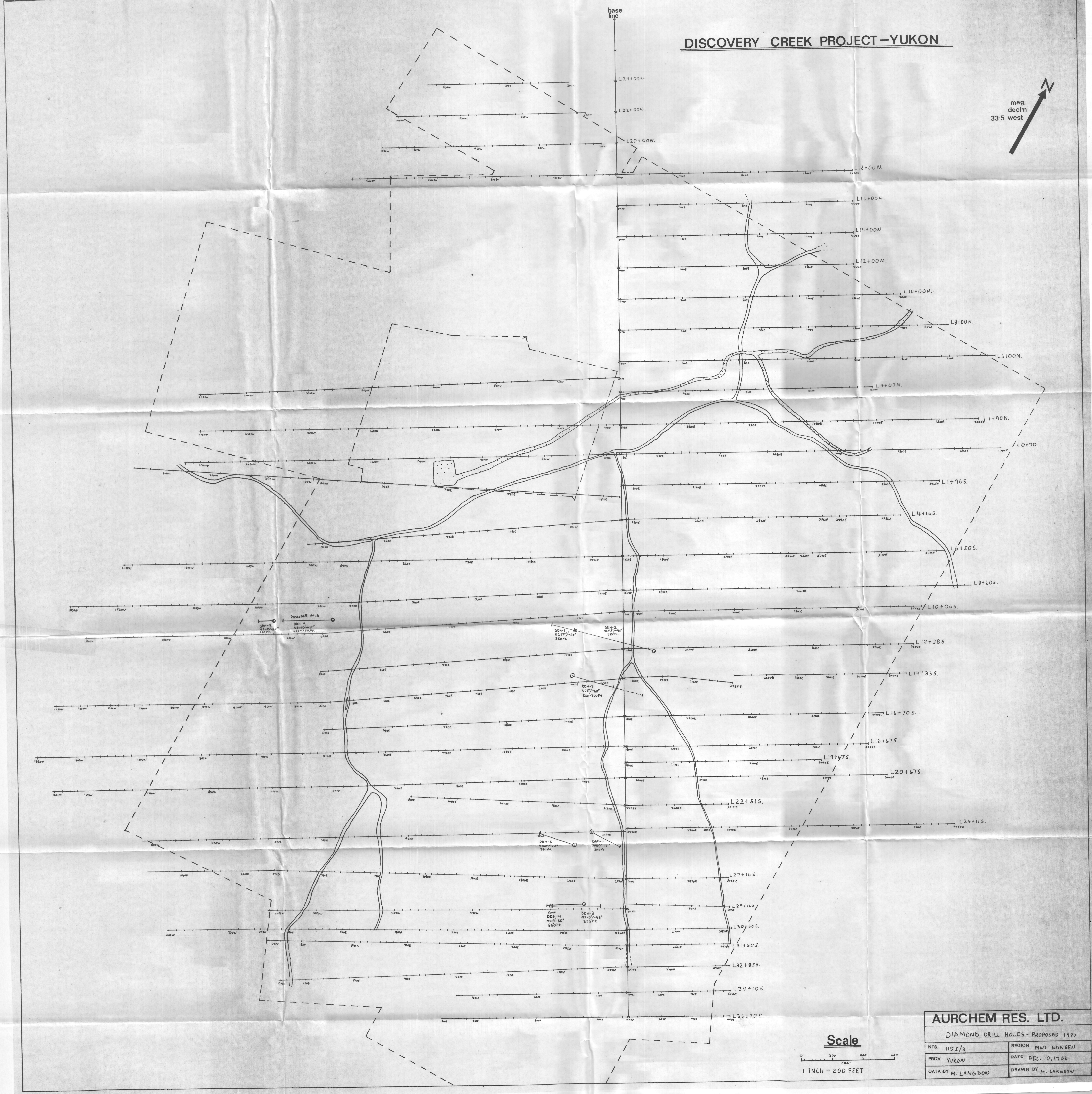


DISCOVERY CREEK PROJECT—YUKON
base line
mag. decl'n 33·5 west
N
L24+00N.
L22+00N.
L20+00N.
L18+00N.
L16+00N.
L14+00N
L12+00N.
L10+00N.
L8+00N.
L6+00N.
L4+07N.
L1+90N.
L0+00
L1+96S.
L4+16S.
L6+50S.
L8+60S.
L10+06S.
L12+38S.
L14+33S.
L16+70S.
L18+67S.
L19+67S.
L20+67S.
L22+51S.
L24+11S.
L27+16S
L29+16S
L30+50S
L31+50S
L32+85S
L34+10S
L35+70S
LEGEND
4 Rhyolite/Rhyodacite/Dacite; Cretaceous, extrusive and usual late phase of 3.
3 Granodiorite; Cretaceous (95M.Y.), some sub-intrusive phases, silicified zones.
2 Volcanics; Jurassic; andesitic and basaltic flows and their volcanoclastic derivatives; Extrusive of 1.
1 Quartz Diorite/Diorite; Jurassic.
Fault.
Geological Contact.
Note; Geology based on trenching and rare outcrops extrapolated by aerial-photo interpretation.
Scale
0 200 400 600
FEET
1 INCH = 200 FEET
AURCHEM RES. LTD.
GEOLOGY
NTS. 115 I/3
REGION WHITEHORSE
PROV. YUKON
DATE SEPT. 15, 1986
DATA BY M. LANGDON
DRAWN BY M. LANGDON

Geology


DISCOVERY CREEK PROJECT
EM-16 SURVEY
% FIELD
+18
+15
+12
+6
0
-6
-12
-15
-18
IN-PHASE
OUT-PHASE
EM-16 INTERPRETATION
WEAK BUT NOTICEABLE CONDUCTOR
WEAK CONDUCTOR
MODERATE IN-PHASE CONDUCTOR
STRONG IN-PHASE CONDUCTOR
VERY STRONG IN-PHASE CONDUCTOR
STRONG IN-PHASE/OUT-PHASE CONDUCTOR
VERY STRONG IN-PHASE/OUT-PHASE CONDUCTOR
SCALE
1 INCH = 200 FEET
0
200
400
FEET
N
AURCHEM RES. LTD.
EM-16 SURVEY
DATE: OCT. 2, 1986.
NTS: 115I/3
LOCATION: YUKON
INTERPRETATE BY: M. LANGDON
DATA BY: R. SCHNEIDER
DRAWN BY: M. LANGDON

EM-16 SURVEY

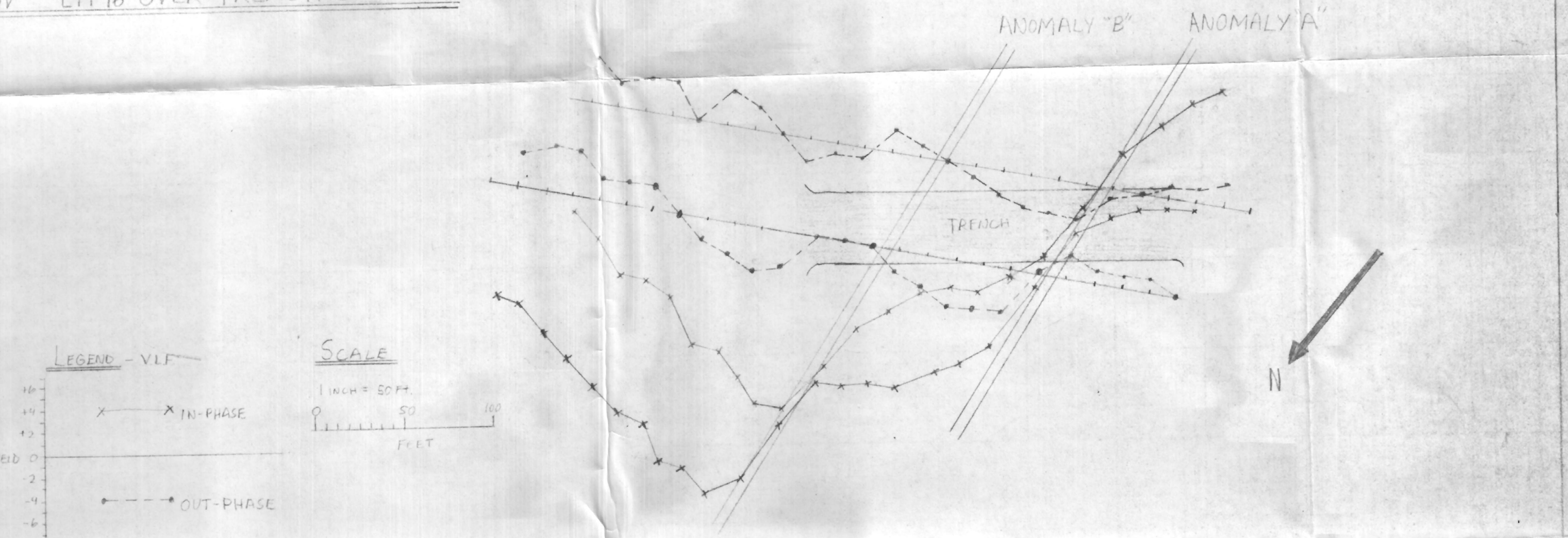

DISCOVERY CREEK PROJECT—YUKON
base line
mag. decl'n 33·5 west
N
L24+00N.
L22+00N.
L20+00N.
L18+00N.
L16+00N.
L14+00N.
L12+00N.
L10+00N.
L8+00N.
L6+00N.
L4+07N.
L1+90N.
L0+00
L1+96S.
L4+16S.
L6+50S.
L8+60S.
L10+06S.
L12+38S.
L14+33S.
L16+70S.
L18+67S.
L19+67S.
L20+67S.
L22+51S.
L24+11S.
L27+16S.
L29+16S.
L30+50S.
L31+50S.
L32+85S.
L34+10S.
L35+70S.
POSSIBLE HOLE
DDH-1
DDH-2
DDH-3
DDH-4
DDH-5
DDH-6
DDH-7
DDH-8
DDH-9
Scale
0 200 400 600
FEET
1 INCH = 200 FEET
AURCHEM RES. LTD.
DIAMOND DRILL HOLES - PROPOSED 1987
NTS. 115I/3
REGION MNT. NANSEN
PROV. YUKON
DATE DEC. 10, 1986.
DATA BY M. LANGDON
DRAWN BY M. LANGDON

Proposed Drill Hole
Locations.

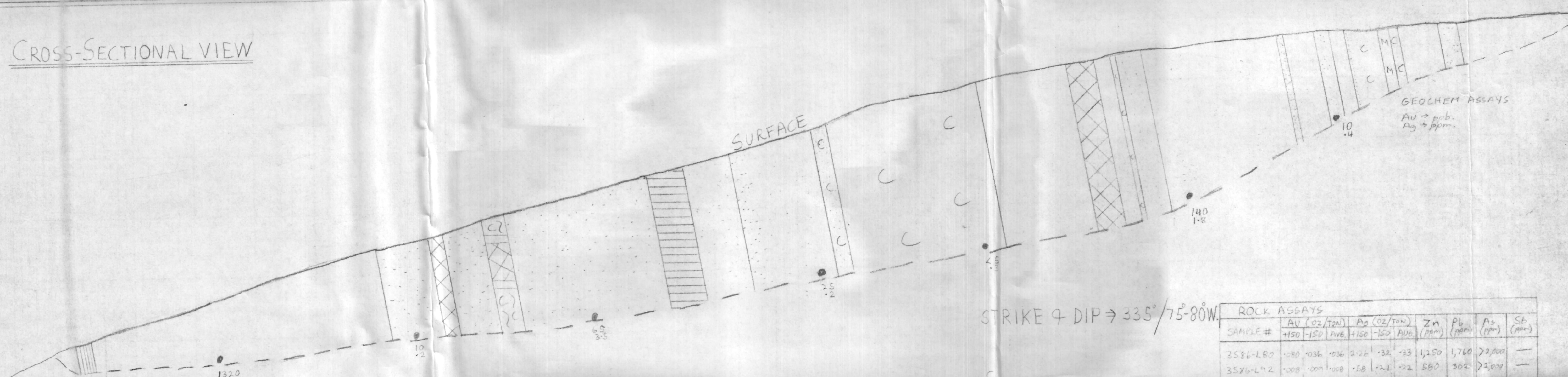
LGCS 1
MYRTLE
RICCO
WEDGE 6
COURTLAND
ACCESS ROAD
LAURA
WEDGE 5
W10
W15
W9
CAMP
DISCOVERY CREEK
W6
W5
W7
W8
5L106
5L90
IDA MAY
LGCS 3
MACK
WEDGE 10
WEDGE8
3L90
W7
W8
RAS 3
4L90
WEDGE 7
W10
W15
RAS1
RAS2
3L106
RAS 2
6L106
WEDGE 15
RAS 3
4L105
RAS 1
5L105
8L102
3L102
SUNSET
HAZEL
ANNE
RAS1 + RAS 2
RAS 4
KEY
Boundry of Surveyed Claims
Boundry of Staked Claims
Found Post
Post (approx. position)
SCALE 1" = 200'
0
200'
400'
600'
N
AURCHEM
RESOURCES LTD.
CLAIM MAP
DISCOVERY CREEK CLAIMS
DATE; SEPT. 17, 1986
DRAWN BY; R. Schneider

CLAIM MAP.

PLAN VIEW – EM-16 OVER TRENCH 3 SOUTH

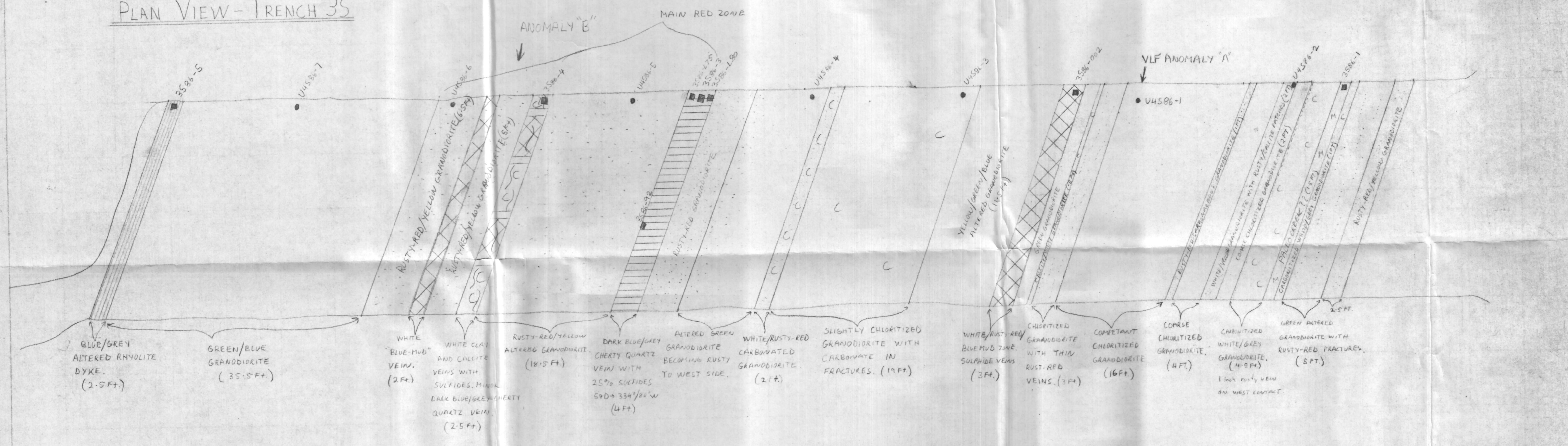


CROSS-SECTIONAL VIEW



ROCK ASSAYS										
SAMPLE #	AU (OZ/TON)			Ag (OZ/TON)			Zn (ppm)	Pb (ppm)	As (ppm)	Sb (ppm)
	+150	-150	AVG.	+150	-150	AVG.				
3586-L80	·080	·036	·036	2·26	·32	·33	1,250	1,760	>2,000	—
3586-L92	·008	·009	·008	·58	·21	·22	580	302	>2,000	—
3586-1	·004	·006	·006	·05	·18	·17	1,400	1,760	1,800	—
3586-2	·013	·013	·013	·46	·35	·35	210	844	824	—
3586-3	·041	·025	·026	·35	·35	·32	960	845	>2,000	—
3586-4	·005	·004	·004	·07	·11	·11	240	93	166	—
3586-5	·001	·001	·001	·01	·01	·01	112	39	47	—
3586-L75	·372	·036	·052	·78	3·72	3·58	9,600	4,740	>2,000	90

PLAN VIEW – TRENCH 3S



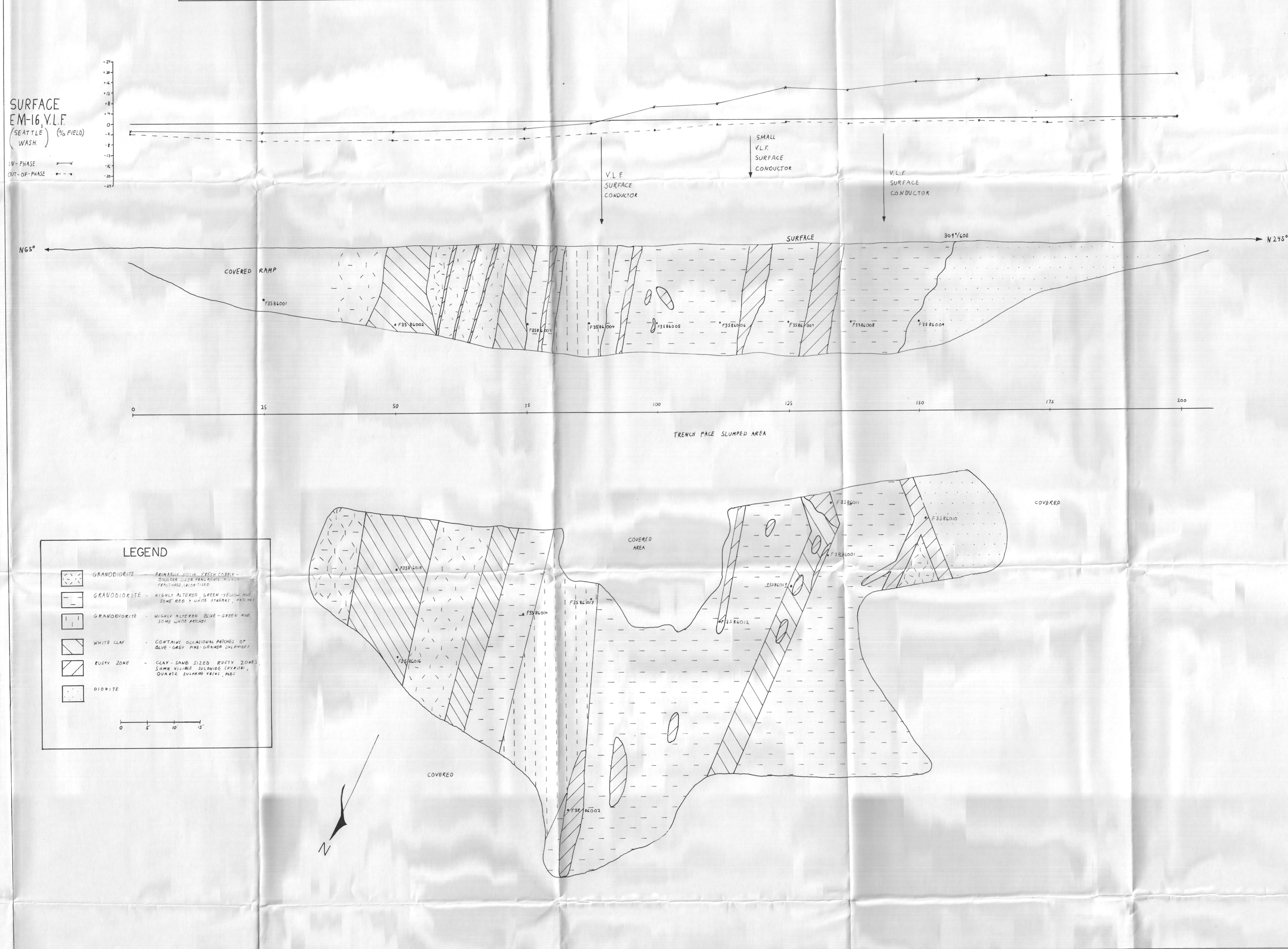
SCALE
1 inch = 8 Ft.
0 8 16
FEET

N

AURCHEM RESOURCES LTD.	
TRENCH 3 SOUTH	
NTS: 115 I/3	DATE: SEPT. 18, 1986
PROVINCE: YUKON	AREA: DISCOVERY CREEK
DATA BY: M. ANDERSON	DRAWN BY: M. LANGDON

TRENCH 3 South.

CROSS SECTION & PLAN VIEW OF TRENCH F3 (MACK CLAIM)(2 SOUTH) - VLF PROFILE, SAMPLE LOCATIONS AND GEOLOGY OF THE DISCOVERY CREEK PROJECT - YUKON TERRITORIES



TRENCH 2 South.

SCHEMATIC CROSS-SECTION ON L1+96 SOUTH (N60°)

GEOLOGY

OLDEST
- 2 INT. & BASALTIC FLOWS, MINOR INTERMIXED SEDIMENTS.
- 1 DIORITE (INTRUSIVE OF ABOVE)

YOUNGEST
- 3 GRANODIORITE
- 4 RHYOLITE (EXTRUSIVE OF ABOVE)

VEIN GEOLOGY

- BLUE MUD – HANGING WALL CLAY ALTERATION ZONES – HIGHLY VARIABLE MINERALIZATION.
- LOW GRADE MINERALIZED VEIN MATERIAL, qtz/calcite VEINS.
- HIGH GRADE MINERALIZED VEINS.
  BOILING ZONE → MASSIVE SULPHIDES.

EM-16 GEOPHYSICS

- → WEAK
- → VERY STRONG

SCALE

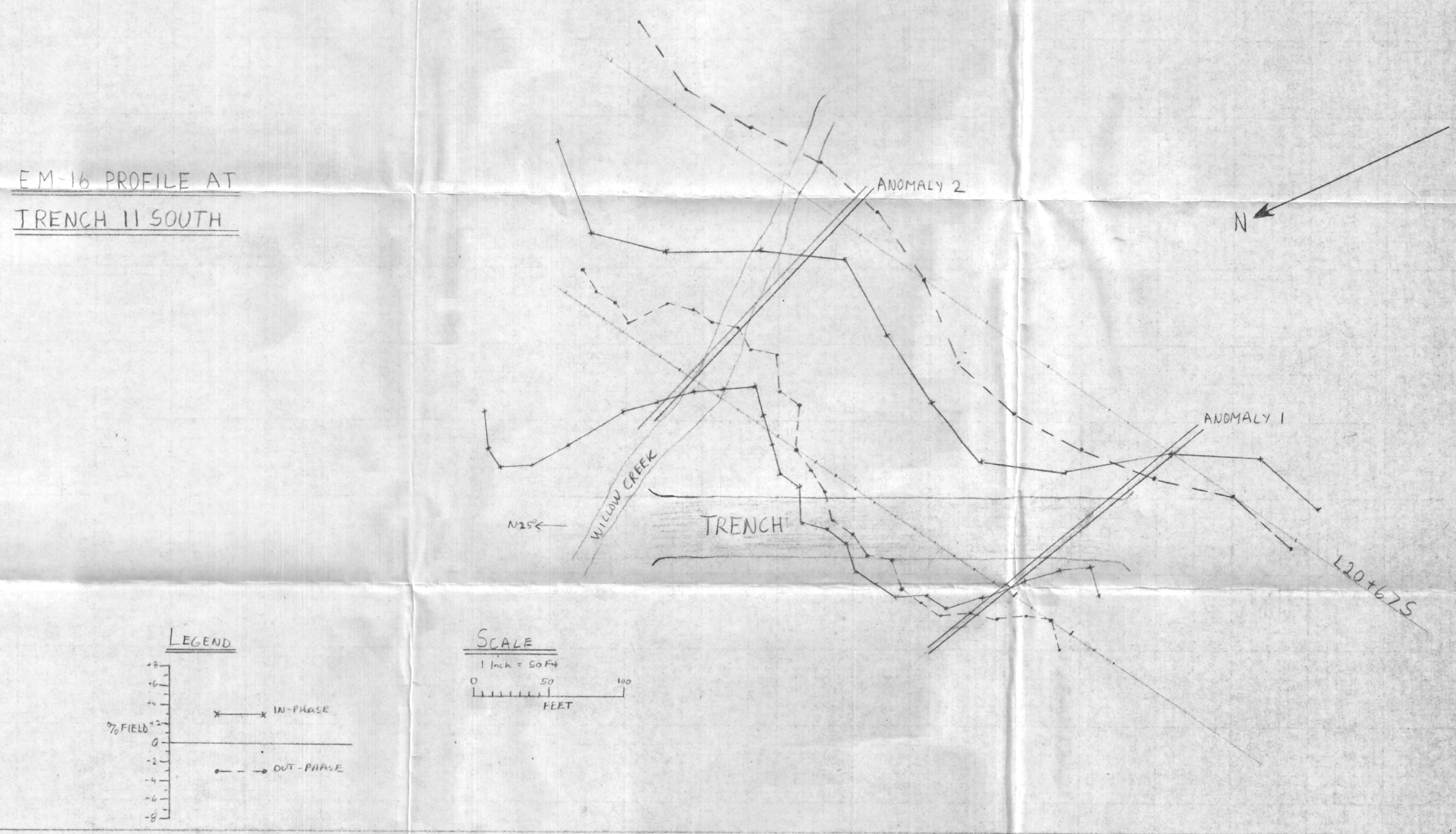
1 inch = 200 Ft.

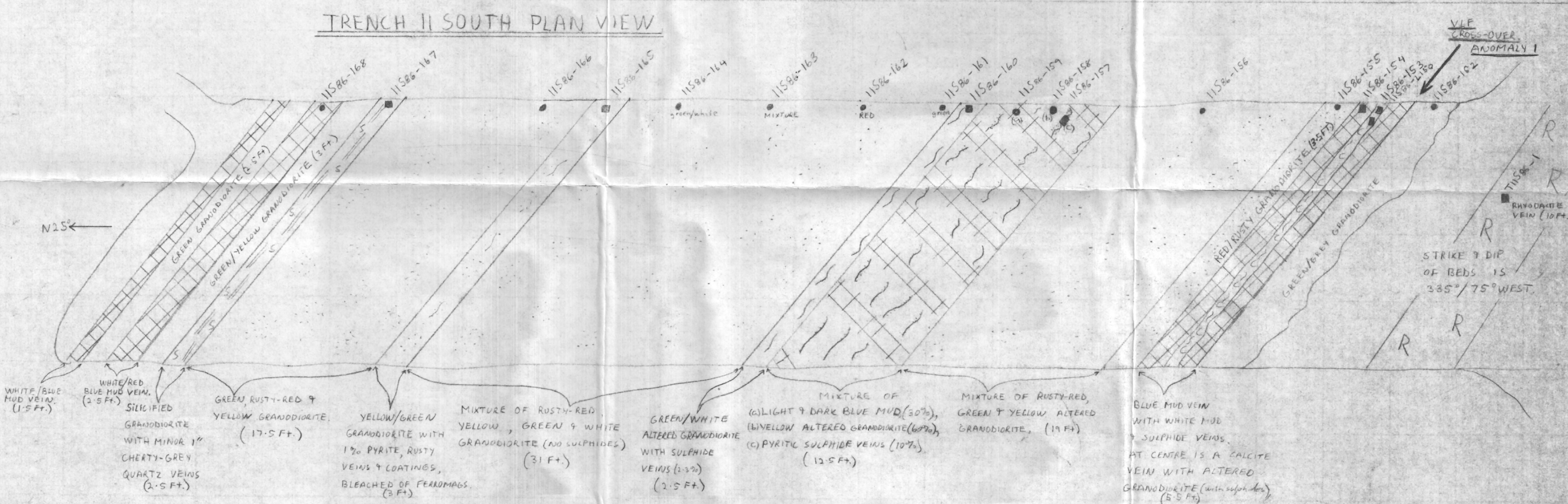
0 200 400

FEET.

AURCHEM RES. LTD.	
[redacted]	
DATE OCT 3, 1986.	NTS. 115 I/3
LOCATION: DISCOVERY CR. YUKON	DRAWN BY. M. LANGDON
ON LINE L1+96 S.	VIEW CUT AT N60°

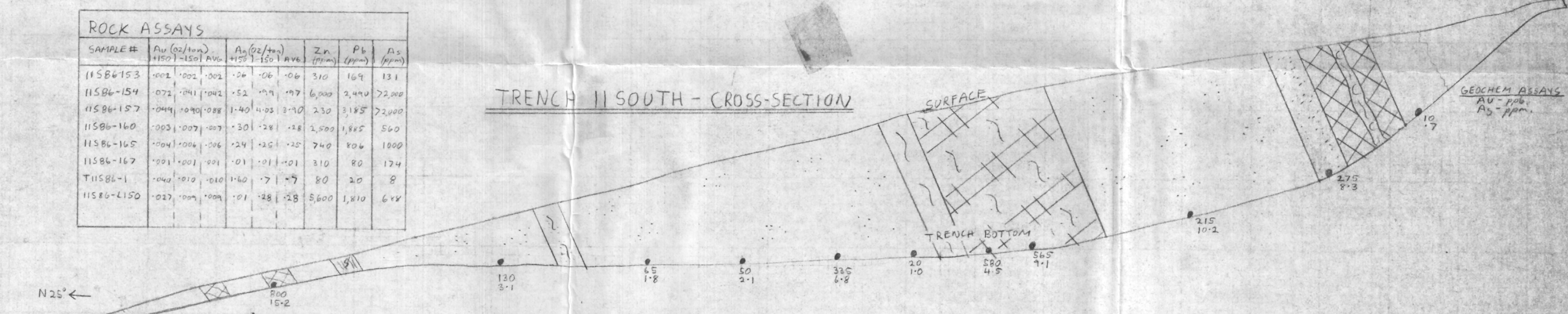
DETAILED VLF
OVER TRENCHES
3S, W8, 11 S

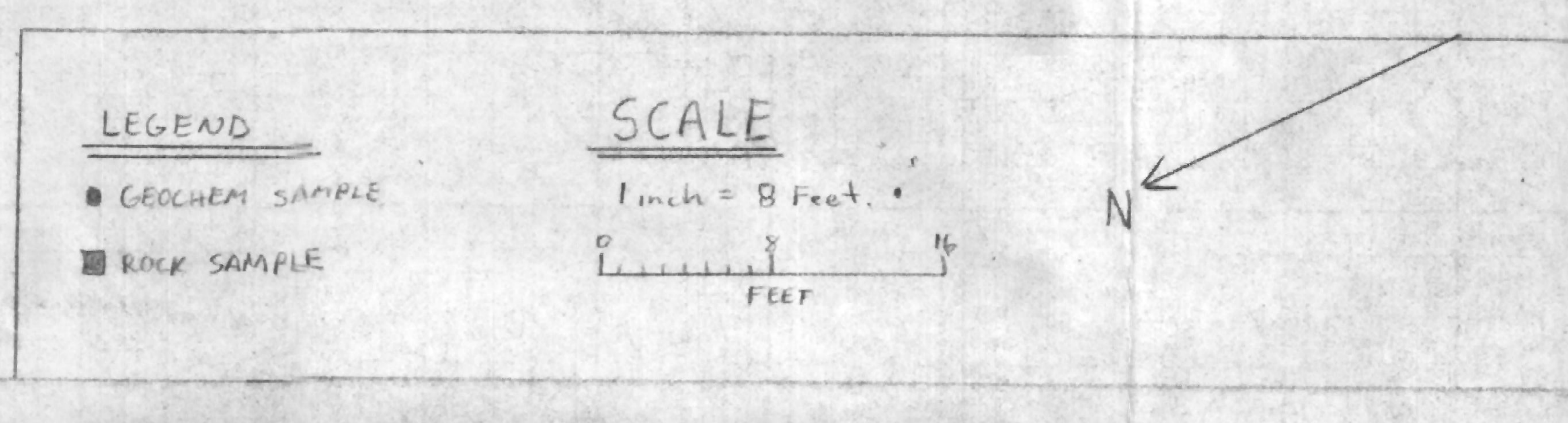




ROCK ASSAYS

SAMPLE #	Au (oz/ton) +150	Au (oz/ton) -150	Au (oz/ton) AVG	Ag (oz/ton) +150	Ag (oz/ton) -150	Ag (oz/ton) AVG	Zn (ppm)	Pb (ppm)	As (ppm)
11S86153	·002	·002	·002	·06	·06	·06	310	169	131
11S86-154	·072	·041	·042	·52	·99	·97	6,000	2,490	>2,000
11S86-157	·049	·090	·088	1·40	4·03	3·90	230	3,185	>2,000
11S86-160	·003	·007	·007	·30	·28	·28	2,500	1,885	560
11S86-165	·004	·006	·006	·24	·25	·25	740	806	1000
11S86-167	·001	·001	·001	·01	·01	·01	310	80	174
T11S86-1	·040	·010	·010	1·60	·7	·7	80	20	8
11S86-L150	·027	·009	·009	·01	·28	·28	5,600	1,810	648





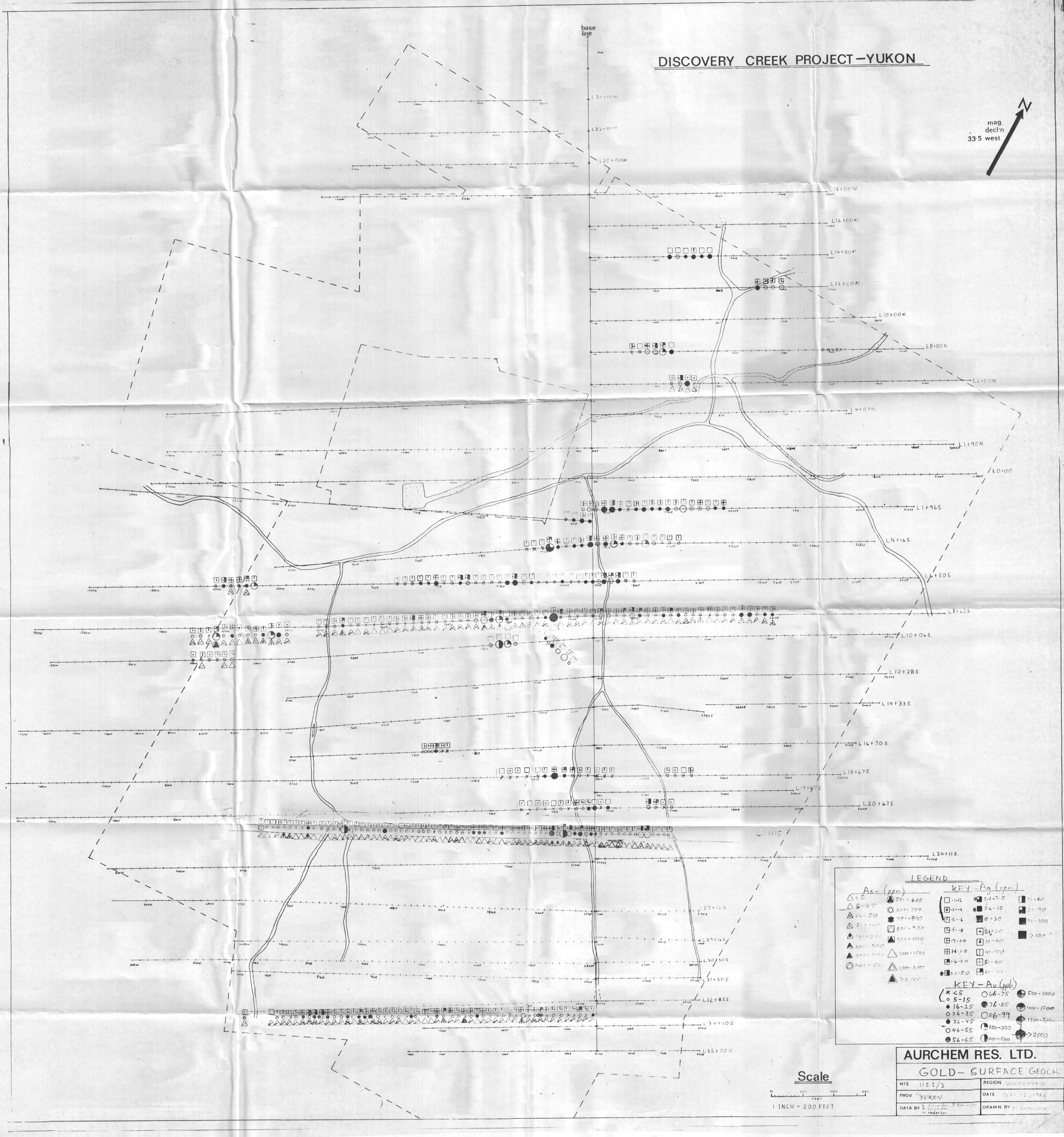
AURCHEM RESOURCES LTD.	
TRENCH 11 SOUTH	
DATE, SEPT. 17, 1986	NTS. 115I/3
PROJECT, DISCOVERY CK.	PROV., YUKON
DATA BY; M. LANGDON	DRAWN BY; M. LANGDON

SOIL
GEOCHEM.

EM-16 V.L.F. PROFILE FROM TRENCH WEDGE 8 TO TRENCH 11 SOUTH — DISCOVERY CREEK PROJECT, YUKON TERRITORIES
SCALE
1 INCH = 20FT.
VLF SCALE
SEATTLE WASH. DATA
MAG. DECL'N 33° WEST
N
TRENCH 11S
ANOMALY
LINE 725W
LINE 615S
LINE 550S
LINE 465S
LINE 380S
LINE 315S
LINE 120S
LINE 80S
LINE 40S
LINE 80S
TRENCH WEDGE 8
VEIN IN TRENCH
MAIN CREEK
TRIB CREEKS
CREEKS
DATE; JUNE 15/86
DRAWN BY; M. LANGDON
DATA BY; M. LANGDON M. ANDERSON
AUROCHEM
AURCHEM

TRENCH

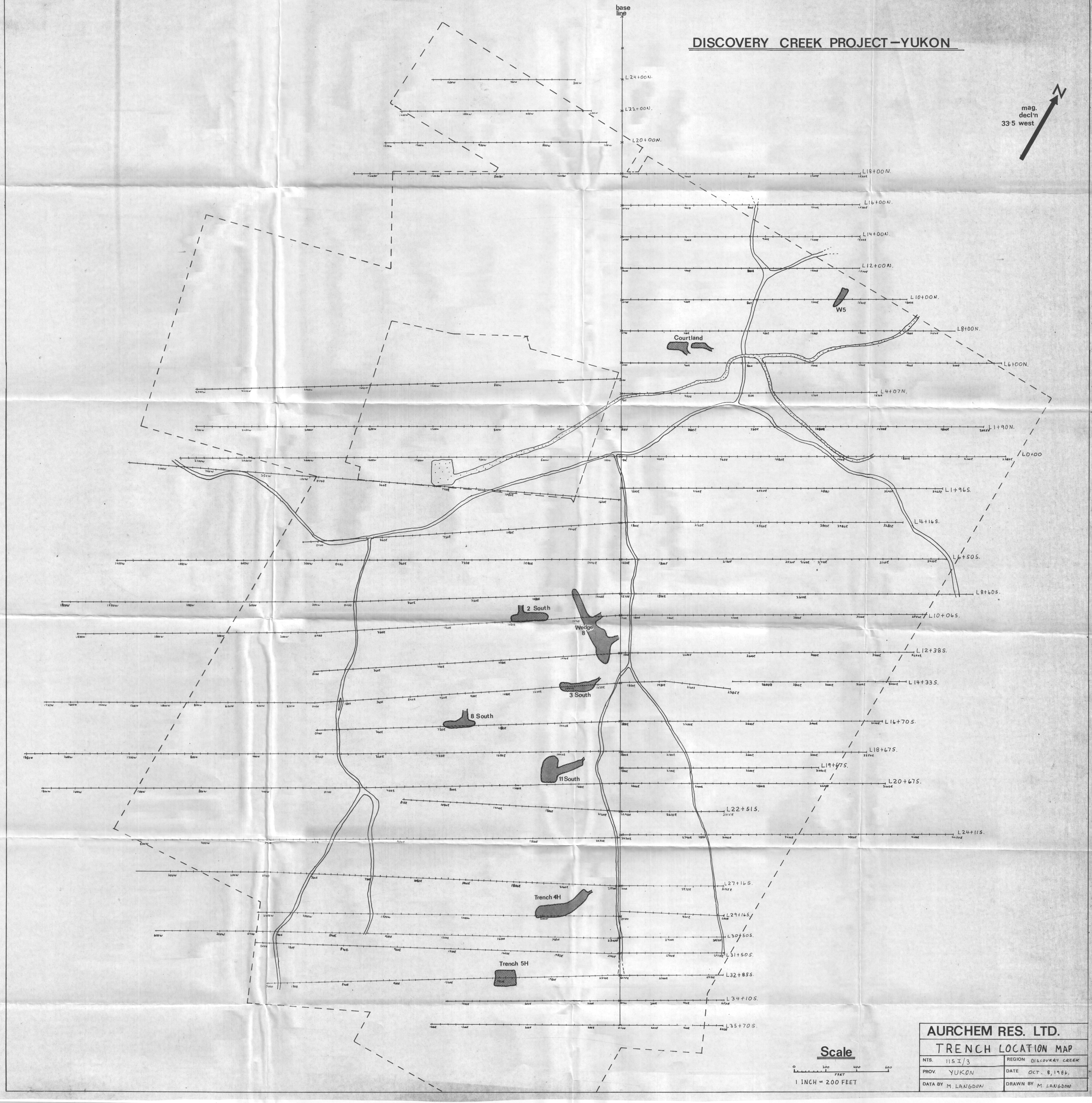
LOCATION
MAP



DISCOVERY CREEK PROJECT—YUKON
base line
N
mag. decl'n 33·5 west
L22+00N
L20+00N
L18+00N
L16+00N
L14+00N
L12+00N
L10+00N
L8+00N
L4+07N
L1+90N
L0+00
L1+96S
L4+16S
L6+50S
L8+60S
L10+06S
L12+38S
L14+33S
L16+70S
L18+67S
L20+67S
L24+11S
L27+16S
L30+50S
L31+50S
L32+85S
L34+10S
L35+70S
LEGEND
As- (ppm)
KEY -Ag (ppm)
KEY - Au (ppb)
Scale
0 200 400 600
FEET
1 INCH = 200 FEET
AURCHEM RES. LTD.
GOLD- SURFACE GEOCH.
NTS 115 I/3
REGION WHITEHORSE
PROV YUKON
DATE SEPT. 12, 1986
DATA BY L. Schneider, J. Schneider, R. Schneider, M. Anderson
DRAWN BY M. LANGDON

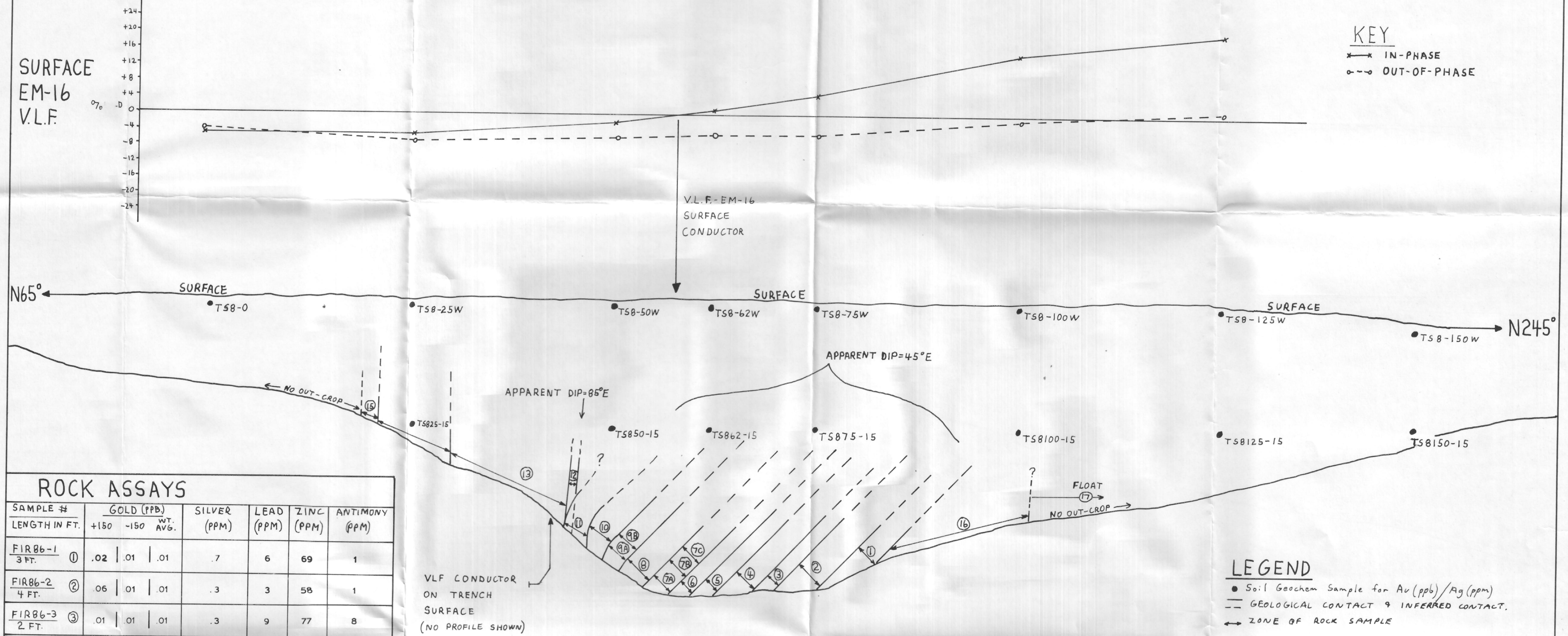
TRENCH

85


DISCOVERY CREEK PROJECT–YUKON
base line
mag. decl'n 33·5 west
N
L24+00N.
L22+00N.
L20+00N.
L18+00N.
L16+00N.
L14+00N.
L12+00N.
L10+00N.
L8+00N.
L6+00N.
L4+07N.
L1+90N.
L0+00
L1+96S.
L4+16S.
L6+50S.
L8+60S.
L10+06S.
L12+38S.
L14+33S.
L16+70S.
L18+67S.
L19+67S.
L20+67S.
L22+51S.
L24+11S.
L27+16S.
L29+16S.
L30+50S.
L31+50S.
L32+85S.
L34+10S.
L35+70S.
W5
Courtland
2 South
Wedge 8
3 South
8 South
11 South
Trench 4H
Trench 5H
Scale
0 200 400 600
FEET
1 INCH = 200 FEET
AURCHEM RES. LTD.
TRENCH LOCATION MAP
NTS. 115 I/3
REGION DISCOVERY CREEK
PROV. YUKON
DATE OCT. 8, 1986.
DATA BY M. LANGDON
DRAWN BY M. LANGDON

CROSS-SECTION PROFILE OF TRENCH 8S.—V.L.F., SOILS & GEOLOGY

DISCOVERY CREEK PROJECT—YUKON TERRITORY



ROCK ASSAYS

SAMPLE # / LENGTH IN FT.		GOLD (PPB.) +150	GOLD (PPB.) -150	GOLD (PPB.) WT. AVG.	SILVER (PPM)	LEAD (PPM)	ZINC (PPM)	ANTIMONY (PPM)
FIR86-1 / 3 FT.	1	.02	.01	.01	.7	6	69	1
FIR86-2 / 4 FT.	2	.06	.01	.01	.3	3	58	1
FIR86-3 / 2 FT.	3	.01	.01	.01	.3	9	77	8
FIR86-4 / 3 FT.	4	.09	.03	.03	.3	15	109	7
FIR86-5 / 2 FT.	5	.02	.11	.11	1.4	13	115	3
FIR86-6 / 2 FT.	6	.05	.03	03	.7	39	374	16
FIR86-7A / 3 FT.	7A	4.74	5.02	5.01	40.0	11,760	6980	173
FIR86-7B / 3 FT.	7B	1·09	·87	·88	15·3	3,180	1,648	56
FIR86-7C / 3 FT.	7C	94·4	2·45	2·74	19·2	3,500	1,860	65
FIR86-8 / 4 FT.	8	·37	·49	·49	3·9	955	2,018	25
FIR86-9A / 3 FT.	9A	9·67	5·14	5·26	12·0	1,665	2,850	133
FIR86-9B / 3 FT.	9B	·31	·68	·66	11·5	683	2,066	56
FIR86-10 / 3 FT.	10	·11	·40	·40	12·0	2,840	2,040	28
FIR86-11 / 3 FT.	11	·05	·08	·08	5·5	606	1,568	21
FIR86-12 / 1 FT.	12	9·95	1·63	1·86	30·0	6,720	1,220	51
FIR86-13 / 16 FT.	13	·02	·02	·02	3·1	39	291	1
FIR86-14 / 10 FT.	14	·06	·07	·07	3·4	72	579	26
FIR86-15 / 2 FT.	15	·01	·01	·01	1·7	6	323	10
FIR86-16 / 10 FT.	16	·01	·03	·02	1·6	15	114	1

LEGEND

- Soil Geochem Sample for Au (ppb)/Ag (ppm)
- GEOLOGICAL CONTACT & INFERRED CONTACT.
- ZONE OF ROCK SAMPLE

SCALE

1" = 8 FT.

0 8 16 24 FEET.

GEOLOGY-GENERAL

17 UNALTERED DIORITE; <3% magnetite, no sulphides visible.

16 FRACTURED DIORITE; <10% magnetite, no sulphides visible.

1 METADIORITE; highly shattered, minor qtz/carb. coatings on fractures, 15-20% magnetite, 3-5% pyrite, minor rusty qtz. patches.

2 RUSTY METADIORITE; same as 1 but very oxidized.

3 BLEACHED DIORITE?; altered to clays, creamy-red/yellow colour, minor pyrite & mag. (<3%), 1cm wide blood-red cross-cutting veins.

4 BLEACHED DIORITE & VEIN; same as 3 but the middle 1ft. is a blood-red coloured rusty-weathered vein (calcite + pyrite + mag.).

5 BLEACHED DIORITE & VEIN; same as 4 but has 3 or 4 rusty veins, highly altered, slightly sheared in places.

6 RHYOLITE; medium to coarse grained, greyish-blue colour, calcite and qtz, ≃5% sulphides, very homogenious.

7 VEIN MATERIAL; highly irregular qtz/carb vein with sulphides (15%), massive sulphide pods & veinlets, very rusty altered material, the width of the vein down strike highly variable as well as the component content, overall about 30% sulphides.

8 METADIORITE WITH VEINLETS; fine grained metadiorite with minor sulphides that have veinlets and pods of qtz/carb/sulphide material as in 7.

9 BLACK METADIORITE WITH VEINLETS; same as 8 but diorite is black coloured from mag./ilmenite/manganese coatings, colour rusty black-red.

10 MIXTURE; light green fine grained rhyolite/fine grained diorite/thin qtz-carb-sulphide veinlets → all shattered and intermixed, <5% sulphides.

11 BLACK METADIORITE WITH VEINLETS; same as 10 but no rhyolite fragments.

12 RUSTY WEATHERED VEIN; highly oxidized rusty coloured vein of unknown lithology, possibly vein material (7) + diorite.

13 DIORITE; relatively fresh unaltered diorite, blocky fracture, <1% disseminated pyrite & pyrrhotite, minor qtz/pyrite fracture planes.

14 RHYOLITE; light grey-green fine grained rhyolite, siliceous, minor carb. veins, fracture veinlets with sulphides plus finely diss. sulphides (<3%)

15 PORPHYRITIC RHYOLITE; reddish-green rhyolite with blue-green subeuhedral porphyritic plagioclases, 1-2% very fine black sulphides.

NOTE; LITHOLOGIES 1 THROUGH 12 REPRESENT A GENERALLY HIGHLY RUSTY-COLOURED OXIDIZED ZONE.

PROCHEM | AURCHEM

DATE; JUNE 11, 1986.
DRAWN BY; M. LANGDON
DATA; M. LANGDON, L. SCHNEIDER, M. ANDERSON.

N.T.S.; 115-I/3
REGION; MOUNT NANSEN

TRENCH

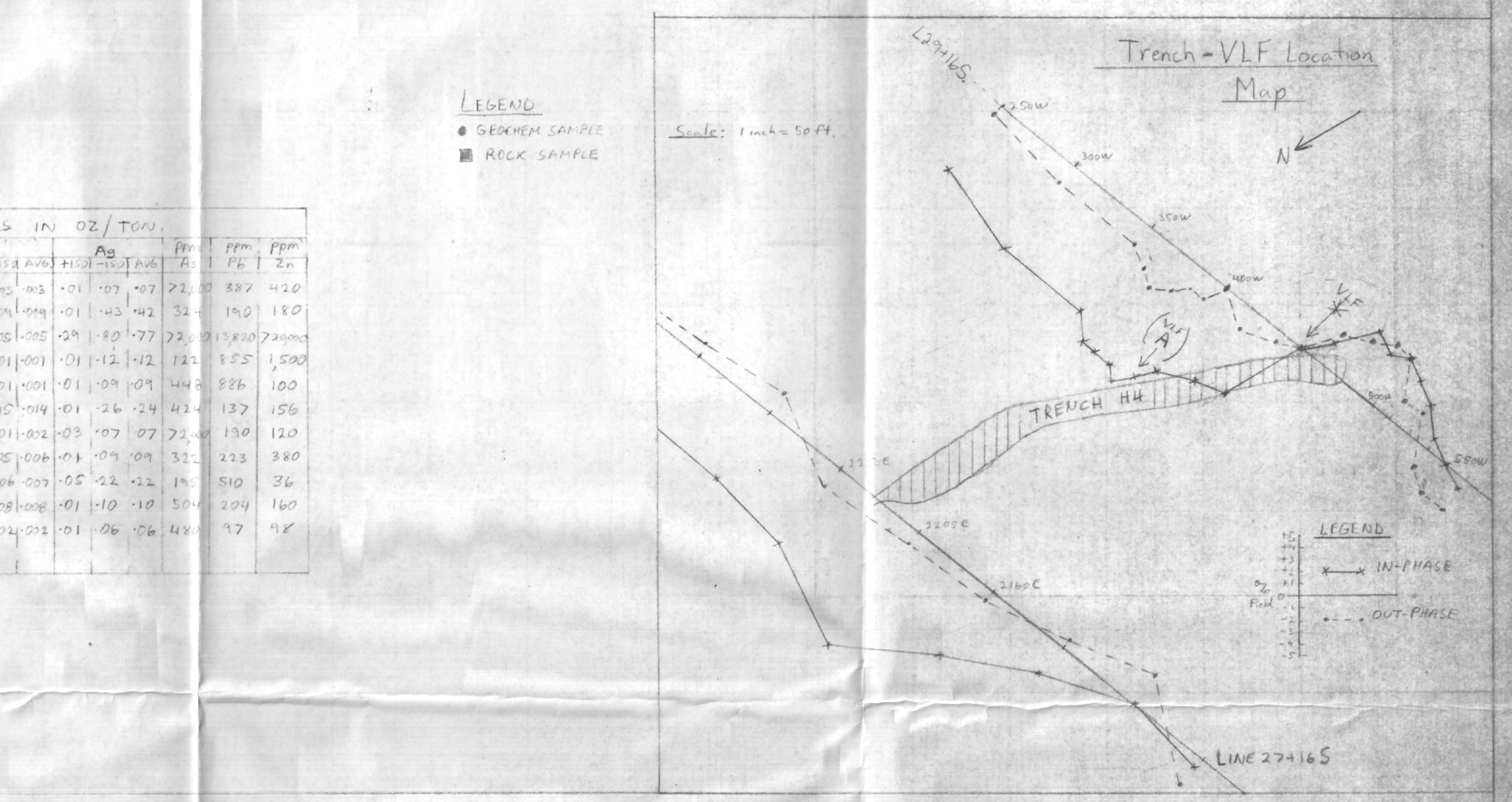
W 8

PLAN VIEW OF TRENCH H4 — GEOLOGY & ASSAYS

LEGEND
- GEOCHEM SAMPLE
- ROCK SAMPLE

ROCK ASSAYS IN OZ/TON.									
SAMPLE #	Au			Ag			ppm	ppm	ppm
	+150	-150	AVG	+150	-150	AVG	As	Pb	Zn
H486-11	·009	·003	·003	·01	·07	·07	72,000	387	420
H486-27	·001	·009	·009	·01	·43	·42	324	190	180
H486-24	·005	·005	·005	·29	·80	·77	72,000	13,820	720,000
H486-25	·001	·001	·001	·01	·12	·12	122	855	1,500
H486-26	·001	·001	·001	·01	·09	·09	448	886	100
H486-28	·001	·015	·014	·01	·26	·24	424	137	156
H486-36	·014	·001	·002	·03	·07	·07	72,000	190	120
H486-37	·015	·005	·006	·01	·09	·09	322	223	380
H486-38	·036	·006	·007	·05	·22	·22	195	510	36
H486-39	·002	·008	·008	·01	·10	·10	504	204	160
H486L-151	·001	·002	·002	·01	·06	·06	480	97	98

Trench - VLF Location Map



CROSS-SECTIONAL VIEW OF TRENCH H4

AURCHEM RESOURCES LTD.	
TRENCH H4	
DATE Sept. 16, 1986	NTS 115I/3
LOCATION Discovery Creek Project	PROVINCE YUKON
DRAWN BY M. LANGDON	DATA BY M. LANGDON, M. ANDERSON

TRENCH 11 SOUTH.

W8 TRENCH – PLAN VIEW
GEOLOGY, SAMPLE LOCATION & ASSAYS

GRAB SAMPLE ASSAYS

SAMPLE #	AU (PPM)	Ag (PPM)	Pb (PPM)	Zn (PPM)	As (PPM)
W886-L57					
W886-L60					
W886-L61					
W886-L62					
W886-L63					
W886-L64					
W886-L65					
W886-L66					
W8PC-L67 (PAN CONC)					
W886-L68					
W886-L69					
W886-L70					
W886-L71					
W886-L72					
W886-L73					
W886-L74					
W886-L76					
W886-L77					
W886-L78					
W886-L79					
W886-L81					
W886-L					

SCALE

1 INCH = 4 FT.

0 4 8
FEET.

115° ⟷ 295°

MAG DECL'N
33.5° WEST

N

LEGEND

- RED-RUSTY VEIN USUALLY < 3" WIDE WITH SULPHIDES.
- GRANODIORITE
- MINERALIZED GRANODIORITE
- WHITE QTZ/CALCITE VEIN MATERIAL WITH SULPHIDE PATCHES (NON-CONSOLIDATED)
- LIGHT BLUE MUD (VEIN)
- DARK BLUE MUD (VEIN)
- LIGHT BLUE QTZ VEIN WITH CALCITE
- LIGHT BLUE VOLCANIC WITH QTZ EYES.
- PINK TO MAUVE FINE GRAINED GRANODIORITE WITH QTZ EYES.
- SILICIFIED GRANODIORITE WITH SULPHIDES. (IF BRECC. Δ)
- GREY QTZ. VEIN FROM WITHIN DARK BLUE MUD. (MINOR CALCITE)
- GALENA/QTZ / SULPHIDE VEIN
- STRIKE & DIP