Moose Mountain Technical Services

Canarc Resource Corp.
800-850 West Hastings Street
Vancouver, BC   V6C 1E1

October 4, 2007

Dear Sirs;

Please find attached the report entitled "New Polaris Project - Preliminary Assessment" This report is based on the previous NI 43-101 report entitled "Resource Potential – New Polaris Project" from March 5, 2007. The study now includes additional Preliminary Assessment work defining underground mining limits based on Scoping level studies of the metallurgy, processing, mining, and infrastructure requirements of a potential operation.

Should you have any questions, do not hesitate to contact us.

Sincerely,

J.H. Gray PEng.

NEW POLARIS PROJECT
PRELIMINARY ASSESSMENT

North Western British Columbia

NTS: 104 K 12

Latitude: 58º42'N **Longitude: 133º37W**

Atlin Mining Division

Submitted to:
Canarc Resource Corp.

800-850 West Hastings Street
Vancouver, BC V6C 1E1

October 4th 2007

Prepared by:

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3.0 SUMMARY

New Polaris (formerly Polaris-Taku) is an early Tertiary mesothermal gold mineralized body located in northwestern British Columbia about 100 kilometres south of Atlin, BC and 60 kilometres northeast of Juneau, Alaska. The nearest roads in the area terminate twenty kilometres due south of Atlin and 10 kilometres southeast of Juneau. Access at the present time is by aircraft. A short airstrip for light aircraft exists on the property.

The deposit was mined by underground methods from 1938 to 1942, and from 1946 to early 1951, producing a total of 740,000 tonnes of ore at an average grade of 10.3g/t gold.

The property consists of 61 contiguous Crown-granted mineral claims and one modified grid claim covering 2,100 acres. All claims are 100% owned and held by New Polaris Gold Mines Ltd., a wholly owned subsidiary of Canarc Resource Corp. subject to a 15% net profit interest held by Rembrandt Gold Mines Ltd. Canarc can reduce this net profit interest to a 10% net profit.

The deposit is composed of three sets of veins (quartz-carbonate stringers in altered rock), the “A-B” veins are northwest striking and southwest dipping, the “Y” veins are north striking and dipping steeply east and finally the “C” veins are east-west striking and dipping to the south to southeast at 65º to vertical. The “C” veins appear to hook around to the north and south into the other two sets of veins so that their junctions form an arc. The gold is refractory and occurs dominantly in finely disseminated arsenopyrite grains that mineralize the altered wallrock and stockwork veins. The next most abundant mineral is pyrite, followed by minor stibnite and a trace of sphalerite. The zones of mineralization range from 15 to 250 metres in length and 0.3 to 14 metres in width.

Canarc explored the “C” vein system between 1988 and 1997, and carried out infill drilling in 2003 through 2006, to better define the continuity and grade of the vein systems.

An updated resource estimate was prepared by Giroux Consultants Ltd. using ordinary kriging of 192 recent drillholes and 1,432 gold assay intervals constrained within four main vein segments as modeled in 3D by Canarc geologists. The total New Polaris database consists of 1,056 diamond drillholes with a total of 31,514 sample intervals.

The geologic continuity of the C vein has been well established through historic mining and diamond drilling. Grade continuity was quantified using a geostatistical semivariograms, which measure distances (ranges) and directions of maximum continuity. The four principle veins in the semivariogram model produced ranges between 50 and 90 metres, both along strike and down plunge.

For this study, the classification for each resource block was a function of the semivariogram range. In general, blocks estimated using ¼ of the semivariogram range were classed as measured, blocks estimated using ½ of the semivariogram range were classed as indicated, and all other blocks estimated were classed as inferred.

The following tables list the undiluted resource estimate, including the “C” vein west (CWM) from the –90m elevation down, and the “C” vein east (CLOE and CHIE) from the –135m elevation down (the elevations, –90m in the west, and –135m in the east, represent the lower elevations of previous mine development and production. The resource potential above these elevations has been discounted in this study, but are listed in the History item, Section 8).

Measured, undiluted resource

Cutoff grade, g/t Au	Tonnes > Cutoff (tonnes)	Grade > Cutoff Au (g/t)	Contained Metal (oz)
2.00	390,000	9.48	119,000
4.00	330,000	10.62	113,000
6.00	271,000	11.89	104,000
8.00	203,000	13.54	88,000

Indicated, undiluted resource

Cutoff grade, g/t Au	Tonnes > Cutoff (tonnes)	Grade > Cutoff Au (g/t)	Contained Metal (oz)
2.00	1,280,000	10.97	451,000
4.00	1,180,000	11.65	442,000
6.00	1,017,000	12.71	416,000
8.00	806,000	14.22	368,000

Measured + Indicated, undiluted resource

Cutoff grade, g/t Au	Tonnes > Cutoff (tonnes)	Grade > Cutoff Au (g/t)	Contained Metal (oz)
2.00	1,670,000	10.62	570,000
4.00	1,510,000	11.42	555,000
6.00	1,288,000	12.54	519,000
8.00	1,009,000	14.08	457,000

Inferred, undiluted resource

Cutoff grade, g/t Au	Tonnes > Cutoff (tonnes)	Grade > Cutoff Au (g/t)	Contained Metal (oz)
2.00	2,060,000	10.52	697,000
4.00	1,925,000	11.03	683,000
6.00	1,628,000	12.15	636,000
8.00	1,340,000	13.27	571,000

The deposit represents an important gold resource and follow-up work should include test mining and infill drilling.

The above resource estimate has been documented in the Technical Report "Resource Potential – New Polaris Project" March 5, 2007. This report was posted on Sedar on March 15, 2007.

For continuity this current report includes the previous Technical Report verbatim plus addition work defining an economic mine plan and operation at a Preliminary Assessment level. The preliminary assessment is based on resources, not reserves, and a portion of the modeled resources to be mined are in the inferred resource category. Resources are considered too speculative geologically to have economic considerations applied to them so the project does not yet have proven economic viability.

The mine plan uses a combination of shrinkage, Alimak, cut and fill and longhole stoping. Development will include a decline from surface and existing working and sublevels. An onsite mill will produce a bulk concentrate for off site autoclaving or bio-leaching and refining. Capital and operating cost estimates include the onsite infrastructure, equipment, supplies, and personnel to support the operation. The project will be a fly-in fly-out project with an onsite camp and airstrip. Major supplies and concentrate shipping will be barged in and out on a seasonal basis.

Development by others in the local area will affect the New Polaris infrastructure and access. Redcorp Venture's Tulsequah project and the nearby Big Bull deposit are examples. In July 2007 a barge load of construction equipment was delivered to site and more loads were expected in the year. The Tulsequah project is less than five kilometers north of the New Polaris project, and could have a significant contributing impact on the New Polaris project with respect to access, infrastructure and other local supply issues.

The results of the Preliminary assessment planning are:

Scheduled Resources	806,000 tonnes measured and indicated grading 13.2 gpt Au (after dilution) and 944,000 tonnes inferred grading 11.9 gpt Au (after dilution) and a 9 gpt cutoff
Production Rate	600 tonnes per day
Grade	12.5 grams per tonne (diluted 20%)
Recoveries	91% gold into concentrate
Output	80,000 oz gold per year
Mine life	8 years

The base case financial parameters are:

Gold Price	US$ 650 per oz	
Exchange Rate	US$ 0.90 = CA$ 1.00	
Capital Cost	CA$90.5 million	
Cash Cost	US$ 327 per oz (excluding offsites)	
	Pre-Tax	After Tax
Net Present Value (NPV) (0%)	CA$60.4 million	CA$40.9 million
NPV (5%)	CA$32.6 million	CA$18.4 million
NPV (8%)	CA$20.3 million	CA$ 8.3 million
NPV (10%)	CA$13.4 million	CA$ 2.7 million
	Pre-Tax	After Tax
Internal Rate of Return	14.9%	11.1%
Payback Period	3.8 years	4.7 years

The preliminary assessment indicates that the New Polaris base case has potential for positive results and therefore further work is recommended to optimize the project and complete a feasibility study.

Additional flotation test work is underway to try and improve gold recoveries and concentrate grades. Autoclave and bio-leach test results will also be completed 3rd quarter 2007, which will be used in marketing the New Polaris concentrates.

Canarc is contemplating a future work program to include driving a decline from surface down to the 1050 mine level (1000 feet below surface), developing one or more drifts and raises within the C vein, “trial” mining to extract a bulk sample, shipping and processing of a representative portion of the bulk sample for final metallurgical testing, finalizing the process flow sheet and completing a feasibility study at an estimated cost of CA$18.7 million. The results of these test will be used as the basis for future Feasibility Study work.

4.0 INTRODUCTION

Canarc Resources Corp. (Canarc) is engaged in the exploration and advancement of the New Polaris Gold Project in British Columbia, Canada.

Moose Mountain Technical Services (MMTS) and Giroux Consultants Ltd. (GCL) were retained by Canarc to assist with the mineral resource modeling and resource estimate of the New Polaris property, and to prepare a Technical Report compliant with NI 43-101 (the Instrument) and Form 43-101F1.

Canarc has consolidated the exploration information for the property from previous owners and participants including the Polaris-Taku Mining Company and Suntac Minerals Corp. Canarc completed their first drill program on the property in 1992.

Mr. Robert J. Morris of MMTS conducted a site visit and detailed examination of the property August 22 through to August 23 2006. During the site visit, sufficient opportunity was available to examine core logging procedures, drill core from the 2006 program as well as conduct a general overview of the property, including selected drill sites, historic core, an underground tour, and the condition of existing project infrastructure. Based on his experience, qualifications and review of the site and resulting data, the author, Mr. Morris, is of the opinion that the programs have been conducted in a professional manner and the quality of data and information produced from the efforts meet or exceed acceptable industry standards. It is also believed that for the most part, the work has been directed or supervised by individuals who would fit the definition of a Qualified Person in their particular areas of responsibility as set out by the Instrument.

Mr. Gary Giroux of Giroux Consultants Ltd. completed the resource estimate. While actively involved in the preparation of the resource estimate, MMTS and GCL had no direct involvement or responsibility in the collection of the data and information or any role in the execution or direction of the work programs conducted for the project on the property or elsewhere. The resource estimate is based on the most recent interpretations by project staff coupled with other data and reports provided by Canarc. Much of the data, including the drill hole assay and geological database upon which the estimate is based, has undergone thorough scrutiny by project staff as well as certain data verification procedures by MMTS.

Mr. Jim Gray of MMTS has compiled the above work along with additional technical planning and evaluation for the mine plan, processing and metallurgy, capital and operating costing, and financial analysis to produce this Technical Report. The previous Technical Report entitled "Resource Potential – New Polaris Project" March 5, 2007 has been used almost verbatim for completeness. The added detail in this current work advances the New Polaris project evaluation to a Preliminary Assessment level. These plans and evaluation were undertaken by MMTS personnel plus the senior level technical and professional persons listed in Item 5.

Sources of information are listed in the references, Item 23.

5.0 RELIANCE ON OTHER EXPERTS

Moose Mountain Technical Services (MMTS) has reviewed the work and compiled this report for Canarc Resources Corp. (Canarc) based on work by MMTS personnel and on the work of others. Canarc supplied the drill hole and assay data base information and Geology interpretations. The responsible experts and their scope of work are namely:

R.J. Morris P. Geo. - MMTS Geology review and drill hole/assay data base QA/QC;

G. H. Giroux P. Eng.. - Giroux Consultants Ltd. (GCL) .- Geostatistics, grade interpolations, and geological resource estimate;

P. A. Stokes P. Eng. – Beaconhill Consultants (1988) Ltd – Mine planning concepts and Financial modeling;

Jasmin Yee P.Eng. - Jasmin Yee and Associates Ltd. – Metallurgical testing and process design processing capital and operating costs;

K Chamberlain - Site Infrastructure and capital and operating costs;

K.E. Robinson P. Eng. – EBA Consultants Ltd. – Tailings Dam Design and Costs

J. H. Gray P. Eng. – MMTS - Study Manager

The quality of information, conclusions and estimates contained herein are based on industry standards for engineering and evaluation of a mineral project. The report is based on: i) information available at the time of preparation, ii) data supplied by outside sources, iii) engineering, evaluation, and costing by other technical specialists and iv) the assumptions, conditions and qualifications set forth in this report. These assumptions include future gold prices, smelter terms, shipping terms, supply costs, and labour rates, all of which are forward looking estimates.

This report is intended to be used by Canarc, subject to the terms and conditions of its contract with MMTS. MMTS disclaims any liability to any third party in respect of any reliance upon this document without MMTS's written consent.

MMTS has not verified the Legal aspects of the ownership of the mineral claims nor the rights granted by the Government of British Columbia. MMTS has not verified Environmental, Socio-Economic, Permitting, or Political issues. The basis of the assumptions in this report are deemed to be reasonable for this location at a Preliminary Assessment level.

6.0 PROPERTY DESCRIPTION AND LOCATION

Adapted from McClintock, 2006

"The New Polaris (formerly the Polaris-Taku mine) property consisting of a group of 61 contiguous crown grants and one modified grid claim totaling 1,196 ha (2,956 acres) located 96 km (60 miles) south of Atlin, BC and 64 km (40 miles) northeast of Juneau, Alaska. Located at approximately 133º37'W Longitude and 58º42'N Latitude, the deposit lies in close proximity to the "Tulsequah Chief" property of Redcorp on the eastern flank of the Tulsequah River Valley (Figure 6-1).

The claims are 100% owned and held by New Polaris Gold Mines Ltd., a wholly owned subsidiary of Canarc Resource Corp. subject to a 15% net profit interest held by Rembrandt Gold Mines Ltd. which Canarc has the right to reduce to 10%. The claims locations are shown on Figure 6-2 while Table 6-1 summarizes the claims shown on Figure 6-2. With the exception of the W.W.1 claim, the claims are crown granted and are kept in good standing through annual tax payments. The W.W.1 is a modified grid claim. The claim has sufficient work filed on it to keep it in good standing until February 4, 2015. The crown granted claims were legally surveyed in 1937. The mineralized areas are shown on Figure 6-3 and 9-2, which shows the geology of the property on the mineral showings. The Polaris No. 1, Silver King No. 1, Silver King No. 5, Black Diamond, Lloyd and Ant Fraction crown grants include the surface rights. Surface rights for the remainder of the property lie with the Crown.

The location of the known mineralization relative to the outside boundary of the property is shown on Figure 6-3**.**

Mining of the AB Vein system and to a lesser extent the Y and C veins was carried out during the 1930s to early 1950s. Much of the former infrastructure has been reclaimed. A $249,000 reclamation bond is in place and it is the writer's opinion that this adequately covers the cost of reclaiming the original mill site and infrastructure. At this time there is no legal or regulatory requirement to remove or treat the tailings on the property. It is recommended that sampling of the tailings and water be carried out to determine if there acid water or contaminants draining from the tailings and mine workings. As well, sampling of water down stream from the site to determine if drainage form the tailings and waste rock is affecting the water quality of Whitewater Creek or the Tulsequah River. If there is contamination of the waters down stream from the waste dumps and tailings a mitigation plan will be required. The cost of the mitigation will depend upon the level of contamination of the water down stream.

Prior to commencing exploration on the property a Notice of Work is required to be submitted to the Mining and Minerals Department of the BC Ministry of Energy and Mines. Work can only commence once approval has been received."

Exploration work carried out in 2006 was covered by:

Mines Act Permit MX-1-208
Approval #06-0100054-0808

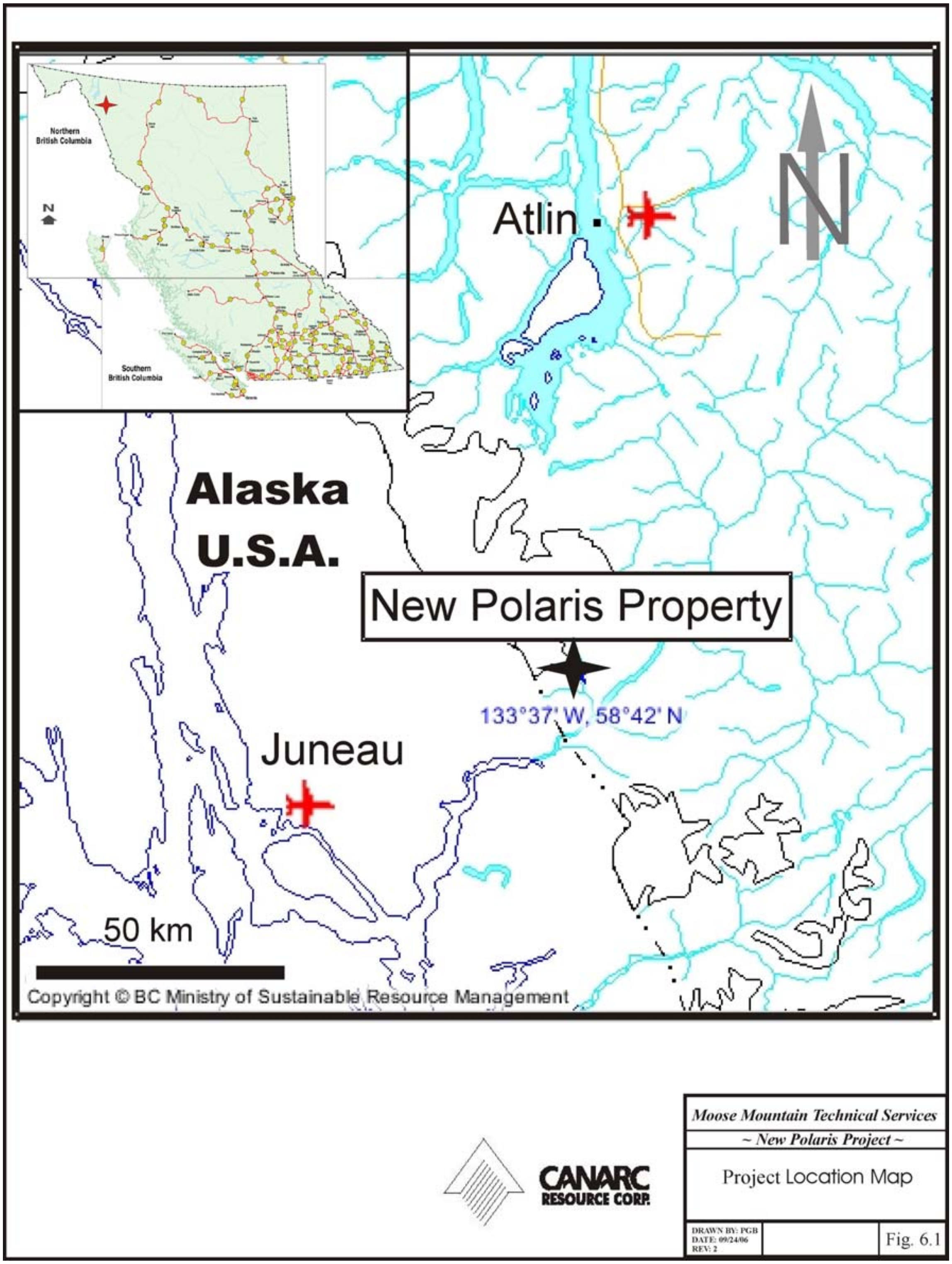


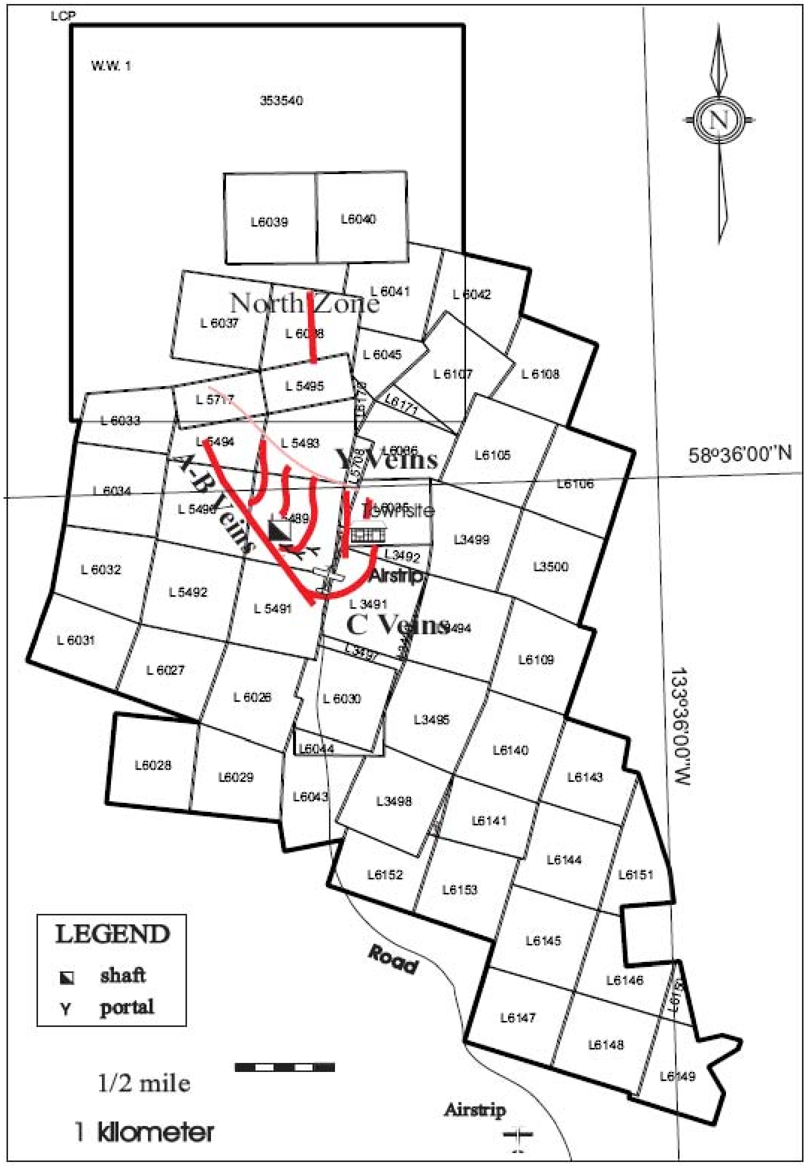
Figure 6-1 Location Map

Table 6-1 List of Claims

Claim Name	Lot No.	Folio No.	Claim Name	Lot No.	Folio No.
Polaris No. 1	6109	4472	Snow	3497	4545
Polaris No. 2	6140	5223	Snow No. 2	3495	5088
Polaris No. 3	6141	5223	Snow No. 3	3494	5495
Polaris No. 4	3498	4545	Snow No. 4	3499	5495
Polaris No. 5	6143	5223	Snow No. 5	6105	4472
Polaris No. 6	6144	5223	Snow No. 8	6107	4472
Polaris No. 7	6145	5223	Snow No. 7	3500	4472
Polaris No. 8	6146	5223	Snow No. 6	6106	4472
Polaris No. 9	6147	5223	Snow No. 9	6108	4472
Polaris No. 10	6148	5290	Black Diamond	3491	4472
Polaris No. 11	6149	5290	Black Diamond No. 3	6030	4944
Polaris No. 12 Fr	6150	5290	Blue Bird No. 1	5708	4545
Polaris No. 13 Fr	6151	5290	Blue Bird No. 2	5707	4545
Polaris No. 14	6152	5290	Lloyd	6035	5010
Polaris No. 15	6153	5290	Lloyd No. 2	6036	5010
Silver King No. 1	5489	4804	Rand No. 1	6039	5010
Silver King No. 2	5490	4804	Rand No. 2	6040	5010
Silver King No. 3	5493	4804	Minto No. 2	6033	4944
Silver King No. 4	5494	4804	Minto No. 3	6034	4944
Silver King No. 5	5491	4804	Jumbo No. 5	6031	4944
Silver King No. 6	5492	4804	Ready Bullion	6032	4944
Silver King No. 7	5495	4804	Roy	6042	5088
Silver King No. 8	5717	4545	Frances	6041	5010
Silver Queen No. 1	6026	4545	Eve Fraction	6170	5495
Silver Queen No. 2	6027	4545	Eve No. 1 Fraction	6171	5495
Silver Queen No. 3	6028	4944	P.T. Fraction	3493	5495
Silver Queen No. 4	6029	4944	Ant Fraction	3492	5088
Silver Strand No. 1	6037	5010	Atlin Fraction	3496	5088
Silver Strand No. 2	6038	5010	Powder Fraction	6043	5088
F.M. Fraction	6044	5088	Jay Fraction	6045	5088
Par Fraction	6154	5290			

W.W.1 Tenure No. 353540 Issue date February 4, 1997. Expiry date: February 4, 2015.

Figure 6-2 Claim Location Map



New Polaris Property
Claim Map with Principle Vein Locations

Figure 6-3 Claim Map with Principle Vein Locations

7.0 ACCESSIBILITY, CLIMATE, LOCATE RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Adapted from McClintock, 2006

"The New Polaris project area lies on the eastern flank of the steep, rugged, Coast Range Mountains. Relief is extreme with elevations ranging from the sea level to 2,600 metres.

Extensive recent glaciation was the dominant factor in topographic development. The Taku and Tulsequah Rivers are the most prominent topographic feature of broad valley bounded by steep mountains. Numerous tributary streams flow from valleys filled with glaciers. The majority of the glaciers are fingers branching from the extensive Muir ice cap, lying to the northwest of the Taku River. The Tulsequah glacier, which terminates in the Tulsequah valley about 16 kilometres north of the New Polaris mine site, is one of the largest glaciers in the immediate area. It forms a dam causing a large lake in a tributary valley that breaks through the ice barrier (Jakülhlaup) during the spring thaw every year, flooding the Tulsequah and Taku valleys below for three to five days.

Small aircraft provides access from Atlin or Juneau. Ocean-going barges have been used in the past to access the site when heavier equipment is required. Redcorp Ventures Ltd. (Redcorp) has applied to complete a road to their project site, across the river and to the north, which could change the infrastructure to the site. The property can be operated year round, however access would be difficult during break up and freeze up.

The climate is one of heavy rainfalls during the late summer and fall months, and comparatively heavy snowfall, interspersed with rain during the winter. The annual precipitation is approximately 1.5 metres of which 0.7 metres occurs as rainfall. The snow seldom accumulates to a depth greater than 1.5 metres on the level. Winter temperatures are not severe and rarely fall below –15ºC. Summer temperatures, in July, average 10ºC with daytime temperatures reaching the high 20's on occasion. The vegetation is typical of northern temperature rain forest, consisting primarily of fir, hemlock, spruce and cedar forest on the hillsides and aspen and alder groves in the river valley."

8.0 HISTORY

Adapted from McClintock, 2006

"From 1923 to 1925 the Big Bull and Tulsequah Chief properties were discovered along the east side of the Tulsequah River and opened up the Taku River district. In 1930, Noah A. Timmins Corporation optioned some of the claims that make up the New Polaris property and conducted trenching and diamond drilling in 1931. The trenching exposed a number of veins of which 10 showed promising grades. A short exploration adit about 9 m long (30 feet) was also driven into the side of the hill and Timmins drilled 19 holes for a total of 1,615 m (5297 feet) but was unable to correlate the intersections and elected to drop the option in September 1932.

The Alaska Juneau Gold Mining Company then optioned the property and conducted underground exploration from the "AJ" (Alaska Juneau) adit. Alaska Juneau drove a total of 190 m of drifting (625 feet) and, although they intersected "ore grade" mineralization, they too had problems with correlation and dropped the property in the fall of 1934.

H. Townsend and M.H. Gidel of the Anaconda Corporation examined the property in 1934 carefully mapping the showings. They came to the conclusion that commercial ore bodies existed even though these showed irregularity due to faulting. Samples were sent to Geo G Griswold in Butte, Montana, who obtained gold recoveries from flotation tests in the order of 88%.

D.C. Sharpstone then secured an option on the property on behalf of Edward C. Congdon and Associates of Duluth, Minnesota. Congdon conducted 236m (775 feet) of underground exploration in the "AJ" tunnel and collared 26 m (85 feet) into the Canyon adit. The Polaris-Taku Mining Company was then incorporated in 1936 to take over the property from Congdon. Polaris-Taku erected a 150-ton per day flotation mill in 1937 and mined underground continuously until it was closed down in April 1942 due to labor restrictions brought on the Second World War. Mining operations resumed in April 1946 and continued until 1951 when the mine was closed due to high operating costs, a fixed gold price and the sinking of a concentrate barge shipment during a storm in March 1951. Up to this date, 231,604 oz of gold was produced at a recovered grade of 0.3opt.

An Edwards Roaster and a cyanide plant to produce bullion were installed and tested in 1949 in order to improve recovery and reduce shipping cost of concentrates to the Tacoma smelter. The addition of the roasted helped improve milling economics, but its capacity was somewhat limited as it could treat only about 45% of the concentrates produced from the flotation plant. After closure, the mill was leased to Tulsequah Mines Ltd. (owned by Cominco) who modified it to process 600 TPD of massive sulphide polymetallic ore (containing gold, silver, copper, lead and zinc) from the Tulsequah Chief and Big Bull Mines. Tulsequah Mines Ltd. used the mill from 1953 to 1957.

Numalake Mines acquired the property in 1953, changed their name to New Taku Mines Ltd and undertook rehabilitation work of the mine's plant. A negative feasibility study in 1973 halted this work. New Taku changed its name to Rembrandt Gold Mines Ltd. in 1974. The property lay idle until Suntac Minerals Corp. optioned the property in 1988 and started surface exploration. Canarc merged with Suntac in 1992 and acquired a

100% interest from Rembrandt in 1994 subject to a 15% net profit interest, which Canarc can reduce to 10%. Canarc's subsidiary, New Polaris Gold Mines Ltd. (formerly Golden Angus Mines Ltd.), currently operates the property.

Exploration restarted on the Polaris Taku property in 1988. During the period 1988 to the end of 2005, a total of 49,427 m (162,163 feet) in 220 holes were drilled on the AB, C and Y vein systems. Individual annual footages are provided in Table 8-1."

Table 8-1 Summary of Exploration Drilling to 2006

Year	Zone	No. of Holes	Metres
1988	Y vein	8	1028
1989	Y vein	19	4078
1990	C vein	10	2862
1991	Y & C veins	11	3333
1992	Y& C veins	23	6378
1993	C vein	8	1301
1994	C & Y veins, North Zone	30	5235
1995	North Zone	20	7600
1996	Underground	24	3205
1997	Underground	49	8869
1998	No drilling	0	0
1999	No drilling	0	0
2000	No Drilling	0	0
2001	No drilling	0	0
2002	No drilling	0	0
2003	C & AB veins	3	1530
2004	C vein	7	1651
2005	C vein	8	2357
2006	C vein	72	24801
Total		**220**	**73,821 m**

"A general distribution of this drilling can be seen in Figure 11-1. Initial efforts were confined to the lower elevations of the property due to limited availability of road building equipment and were designed to test the "Y" Vein system either down dip or along strike from old workings. Discovery of the "C" Vein system in 1989 resulted in a refocusing of efforts towards defining this Zone. Drilling during 1994 and 1995 has been designed to test the North Zone and the downward continuity of the "C" Zone. Drilling on the North Zone cut low-grade gold mineralization in a gently dipping shear zone. Drilling at 60 m (200 foot) centres showed the mineralization to be of limited extent and bounded down dip by a post mineralization fault. No additional drilling of the North Zone is warranted.

Diamond drilling from underground workings in 1996 was focused from the AJ level and targeted both the AB and Y vein systems. This work showed that the AB system did not continue to depth and appears at its south east end to bend from a south east strike to an easterly strike direction and become part of the C vein system. As there appears to be little potential for significant additional mineralization on the AB vein system, little exploration of the AB vein has been carried out since 1997.

Diamond drilling from underground workings in 1997, was focused from the AJ, Polaris and 150 levels and targeted the AB, Y, and C vein systems. Due to the location of the workings relative to the orientation of the veins, many of the holes were drilled sub parallel to the dip and strike of the veins. For this reason, since 1997 drilling has been carried out from surface to allow holes to test the veins obliquely to strike and dip.

Drilling to the end of 1997 identified the C vein system as having the most potential for extensive gold mineralization with gold grades and thicknesses comparable to that mined in the 1930s to early 1950s. Mineralization was encountered in drill holes over a 250 metre by 300 metre area, which remained open to depth. Although the mineralization appears to be continuous between drill holes, the spacing between vein pierce points was too great to give the confidence to calculate a resource. Drilling from 2003 to 2005 focused on closing the drill hole spacing in order to determine the continuity of the grade and thickness of the C vein system.

Drilling to the end of 1997 on the Y vein indicates they are relatively narrow and less continuous along strike than the C veins. Gold grades are comparable to the C vein and these veins have remaining potential for the discovery of additional gold mineralization at depth. Further drilling is required to prove the continuity, gold grades and thicknesses of the veins. The smaller size potential of the Y vein system makes it a second order priority for future drilling.

Since the closure of the Taku Polaris Mine in 1951, several resource estimations have been made with the goal of identifying the probable order of magnitude of "reserves" that may be defined over time. None of these estimations meet the definition requirements of NI 43 – 101 for a resource. The Authors have not done sufficient work to classify them as current reserves or resources and is not treating them as current. These estimates therefore should not be relied upon.

An estimate of Polaris-Taku reserves was made prior to closure in 1951 based on stringent precepts. "Reasonably Assured" ore was projected 7.6 m (25 feet) in the plane of the vein above and below sampled drift sections of mineable grade while "possible" ore was projected an additional 7.6 m beyond these confines (Parliament 1949). These reserves were apparently based solely on underground sampling without using underground diamond drill intercepts (WGM 1992). The "remaining reserves" at the time of closure was 105,000 tons grading 0.42 oz/ton including 17% dilution. None of these estimations meet the definition requirements of NI 43 – 101 for a resource. The Authors have not done sufficient work to classify them as current reserves or resources and is not treating them as current. These estimates therefore should not be relied upon.

Adtec Mining Consultants (1972) re-estimated these "reserves" in contemplation of reopening the mine. These were recalculated to be 148,000 tons at 0.29 oz/ton. Based on similar definitions and existing mine drawings and assay plans, Adtec Consultants (1983) re estimated the remaining "reserves" within the mine workings. These were defined to be in the order of 223,000 tons at 0.32 opt gold (diluted) based on a 0.15 oz/t cut-off and a minimum mining width of 4 feet. These reserves were subdivided into 51,000 tons of "assured" and 72,000 tons of "reasonably assured" reserves. This estimate does not meet the definition requirements of NI 43 – 101 for a resource. The Authors have not done sufficient work to classify them as current reserves or resources and is not treating them as current. This estimate therefore should not be relied upon. Significant work has been done since this estimation and the Author does not believe this estimate is relevant.

Beacon Hill re estimated these reserves in 1988 for Suntac Minerals Corporation using a minimum mining width of 1.5 m (5 feet (instead of 4 feet)) with similar results. Their reserve estimate was "limited to those areas where continuous sampling data was available along drifts, raises and stope backs, etc. and where it appears that minimal development work would be required to access the reserves". Beacon Hill estimated a total probable and possible reserve of 244,420 tons at 0.33 oz. opt gold with 132,210 tons at 0.33 opt gold classed as probable and 112,210 tons at 0.32 opt gold classed as possible. This estimate does not meet the definition requirements of NI 43 – 101 for a resource. The Authors have not done sufficient work to classify them as current reserves or resources and is not treating them as current. This estimate therefore should not be relied upon.

In 1989, Beacon Hill added further probable and possible mining reserves from 27 new drill holes completed by Suntac. They estimated that the new drilling had increased the reserves by 380,000 tons at 0.39 oz. Au/SDT (probable) and 820,000 tons at 0.39 opt gold (possible) which, added to their previously calculated reserves, brought the overall reserve potential up to 1,450,000 tons at 0.38 opt gold (diluted) above the lowest worked level of the mine (600 level at elev. –462 feet Below Sea level 'BSL'). This estimate does not meet the definition requirements of NI 43 – 101 for a resource. The Authors have not done sufficient work to classify them as current reserves or resources and is not treating them as current. This estimate therefore should not be relied upon.

Montgomery Consultants were commissioned to conduct a Geostatistical Study of the Geological Resource for the Polaris-Taku Deposit in 1991. G.H. Giroux carried out this review and calculated a total resource of 2,225,000 tons grading 0.433 opt gold based on a geostatistical approach using a cut-off grade of 0.25 opt gold. These reserves were divided into 333,000 tons at 0.437 opt gold (probable) and 1,892,000 tons at 0.432 opt gold (possible). The estimate discounted much of the reserves around the old workings and did not include dilution and minimum mining width provisions. These estimates were based on both old and new drilling and extended the resource base down to roughly 1200 feet BSL. This estimate does not meet the definition requirements of NI 43 – 101 for a resource. The Authors have not done sufficient work to classify them as current reserves or resources and is not treating them as current. This estimate therefore should not be relied upon.

Watts, Griffis, and McQuat were contracted to review the previous reserves in August 1992. Their review incorporated the residual reserves within the mine workings, as estimated by Beacon Hill in 1989, into their overall estimate of a total (diluted) mineral resource of 1,600,000 tons at 0.46 opt gold. Their estimations were based upon a minimum mining width of 5 feet or 15 % dilution and a cut-off grade of 0.25 opt gold. The improvement in grade stems from the inclusion of new deeper holes that extend the known mineralization to a depth of 1200 feet BSL and exclusion of lower grade material previously included in the Montgomery estimate. This estimate does not meet the definition requirements of NI 43 – 101 for a resource. The Authors have not done sufficient work to classify them as current reserves or resources and is not treating them as current. This estimate therefore should not be relied upon.

Giroux was further contracted to provide resource updates throughout 1992 and in February 1995 he re estimate the resources for the newly drilled portions of the "C" Zone. Recent drilling has also confirmed the existence of a new "North" Zone, which, although it appears to be low grade (0. 18 opt gold) has exhibited possible significant widths in the order of 22 feet. Giroux has included estimations for this zone, which for purposes of this review have been excluded due to grade. The results of his re estimate show that the "C" Vein discovered just prior to mine closure represents a significant new addition to the resource base. He has estimated a total of 85,700 tons grading 0.426 opt gold (probable) and 595,000 tons grading 0.425 opt gold (possible) for this zone below the 450 Level (elev. 313 ft BSL) and 1000 feet BSL.

Most of this resource lies above 800 feet BSL and within 200 feet of the existing shaft bottom. The total resources estimated by Giroux to date are summarized on Table 4.2. His estimates were in situ based on a 0.25 opt gold cut-off and did not include dilution provisions as shown below and considered to be relevant as they are based on a significant amount of data and were independently calculated.

In order to summarize the variety of estimations identified above; Godfrey Walton did the following: Beacon Hill estimation of residual reserves within and around the workings was totaled. To this total, the geostatistical resource estimation of Giroux were added after applying a general dilution factor of 25% at zero grade to Giroux's figures for the " Y " Zone and 15% at zero grade for the "AB" and "C" Zones. The in-situ resource base is presently estimated as 582,910 tons at 0.359 opt gold (Probable), and 2,614,210 tons at 0.363 opt. gold (Possible) including appropriate dilution factors. The dilution factors were estimated based on vein characteristics. The "Y" Veins are described as being high grade, but narrow which makes them prone to high dilution from over-break during mining as well as over mining. The "AB" veins in-situ grade, as estimated by Giroux, already contains internal dilution from a parallel dike. To this total, Walton added overall additional dilution of 15 %, which, he felt, was appropriate, as the "C" vein would not experience much dilution since it is generally thought to be fairly thick. This estimate does not meet the definition requirements of NI 43 – 101 for a resource. The Author has not done sufficient work to classify them as current reserves or resources and is not treating them as current. This estimate therefore should not be relied upon.

In the Author’s opinion, the residual reserves in and around the workings included in the Beacon Hill estimation are unlikely to contribute significantly to any new mining operation. For the most part it is in remnants scattered amongst the old stopes and will be difficult to access and develop.”

Table 8-2 Historic Resource Estimates

Polaris Takus Geostatistical Resources								
	PROBABLE RESOURCES				POSSIBLE RESOURCES			
	In-Situ		Diluted		In-Situ		Diluted	
	Tons	Grade	Tons	Grade	Tons	Grade	Tons	Grade
Zone	(SDI)	(oz/SDT)	(SDI)	(oz/SDI)	(SDI)	(oz/SDT)	(SDI)	(oz/SDI)
GIROUX (1995)								
Y Zone	210,000	0.461	262,500	0.369	987,000	0.469	1,234,000	0.375
AB Zone	78,000	0.403	89,700	0.35	508,000	0.387	584,000	0.337
C Zone	85,700	0.426	98,500	0.37	595,000	0.425	684,000	0.37
Sub-total	373,000	0.441	450,700	0.365	2,090,000	0.437	2,502,000	0.365
BEACON HILL (1988)								
Upper Levels	53,440	0.37	67,800	0.29	41,560	0.35	53,450	0.27
Lower Levels	50,170	0.5	64,410	0.39	45,000	0.48	58,760	0.37
Sub-total	103,610	0.43	132,210	0.33	85,560	0.42	112,210	0.32
TOTAL	**476,610**	**0.439**	**582,910**	**0.359**	**2,175,560**	**0.436**	**2,614,210**	**0.363**

Note: With NI 43-101 guidelines, the terms Probable Resources and Possible Resources have been replaced with Measured Resources and Indicated Resources.

Subsequent to the above Historical estimates additional exploration has been done, particularly 90 hole drilled from 2003 to 2006 by Canarc. A revised Resource estimate was made an included in the NI 43101 Technical Report entitled “Resource Potential, New Polaris Project” March 5, 2007. This current report is the basis of the March 5, 2007 Technical Report and for completeness the Resource estimate is included in this report in its entirety.

9.0 GEOLOGICAL SETTING

Adapted from McClintock, 2006

"The geology has been taken from regional reports and a number of company reports listed in the references."

9.1 Regional Geology

Adapted from McClintock, 2006

"The New Polaris Mine lies on the western edge of a large body of Upper Triassic Stuhini Group volcanic rocks, which has been intruded by a Jurassic-Cretaceous granodiorite body north of the mine. Older Triassic volcanic rocks and earlier sediments underlie the Stuhini volcanic rocks. The granodiorite is part of the Coast Plutonic Complex (Figure 9-1).

The structural trend in the area is northwest-southeast, paralleling major faults and folds to the east and intrusive alignment to the west. The Triassic volcanic rocks and older sedimentary rocks have been folded and sheared with the Stuhini Group rocks being deformed into broad to isoclinal, doubly plunging symmetrical folds with large amplitudes."

9.2 Property Geology

Adapted from McClintock, 2006

"Canarc has carried out extensive mapping of the Polaris-Taku property since the early 1990's. The work has been done by a number of employees and contractors and is shown in Figure 9-2. The gold deposit is hosted within an assemblage of mafic (basalt and andesite units) volcanic rocks altered to greenschist metamorphic facies. The orientation of these units is inconclusive because there are no marker beds in the sequence. It is thought that the units are steeply dipping (70º to 80º) to the north based on the orientation of the limestone/basalt interface at the southern portion of the property.

A serpentinite unit is located to the northeast, which was identified in recent (1996/97) drilling and underground mapping. This unit appears to form the eastern extent of the mineralization. The age relationship is unclear, but it is assumed that the serpentinite is a later stage feature possibly associated with tectonism in the area.

The 'vein' zones are structurally controlled shear zones and are typified by silicification and carbonatization cross cutting actual quartz-carbonate veins. These zones have sharp contacts with the wall rock and form anastamosing ribbons and dilations. These zones have been deformed several times, which makes original textures difficult to determine. The zones are generally tabular in geometry forming en-echelon sheets within the more competent host lithologies.

All of the strata within the property have been subjected to compression, rotation and subsequent extension. The plunge of folds appears to be variable though generally shallow. Small-scale isoclinal folds strike north-northwesterly and plunge moderately to the north. Numerous faults are found on the property, the more significant of which are discussed later.

The possible extension of the Llewellyn fault, termed the South Llewellyn fault, continues south from the Chief Cross fault along mine grid coordinate 4400 East. Slightly north of Whitewater Creek it is offset to the west by an east-west fault, the 101 fault, to continue in a more southeast orientation of the opposite side of Whitewater Creek. This northwest-southeast orientation structure was named the Limestone Fault due to its bedding parallel attitude within a discontinuous limestone/marble horizon. It marks the southwest boundary of the “mine wedge”: the wedge shaped package of rock within which all past production took place. The northern boundary of the “mine wedge” is further defined as mentioned above by the Whitewater Creek Schist Zone, a zone of schistose chlorite-amphibolite-serpentinite less than 300 feet thick. A complex network of brittle faults is also found within this zone.

Three major faults, Numbers 1 and 5, and an unnamed fault, lie within the mine wedge. The No.1 and No.5 faults strike northwest-southeast, dipping approximately 45º to the northeast, and are sub-parallel to the unnamed fault, which dips steeply to the southwest. The No.1 fault has reverse displacement of up to 100 feet while the displacement of the No.5 fault is poorly defined. The southwest dipping, unnamed fault showed no displacement, as it apparently parallels the A-B vein system. The mined out areas indicate the wedge shape, the predominant orientations and continuity of the zones, and the overall plunge of the system to the southeast. An early interpretation of the structure showed that various veins appear to meet and form “junction arcs” where both thickness and grade improve.”

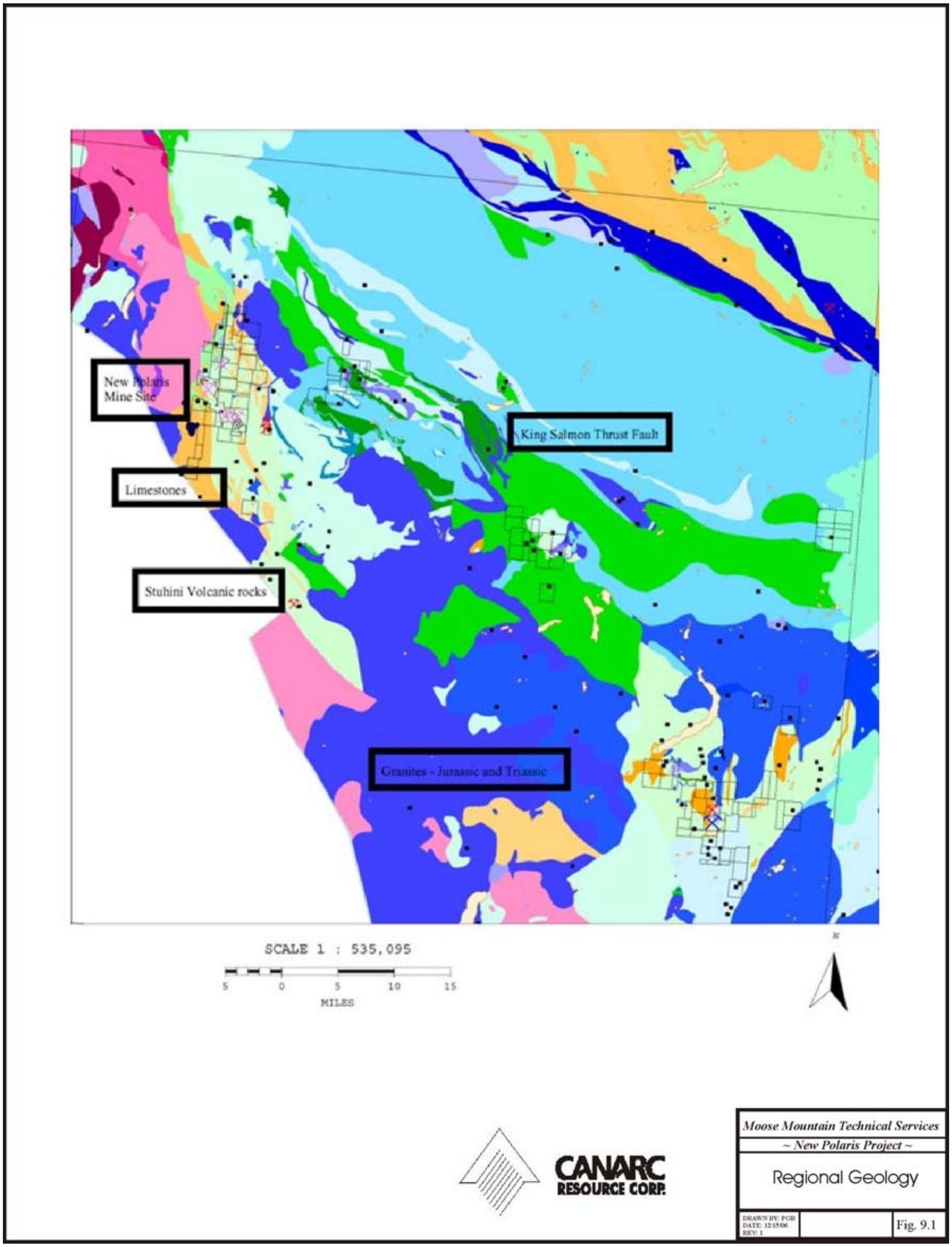


Figure 9-1 Regional Geology

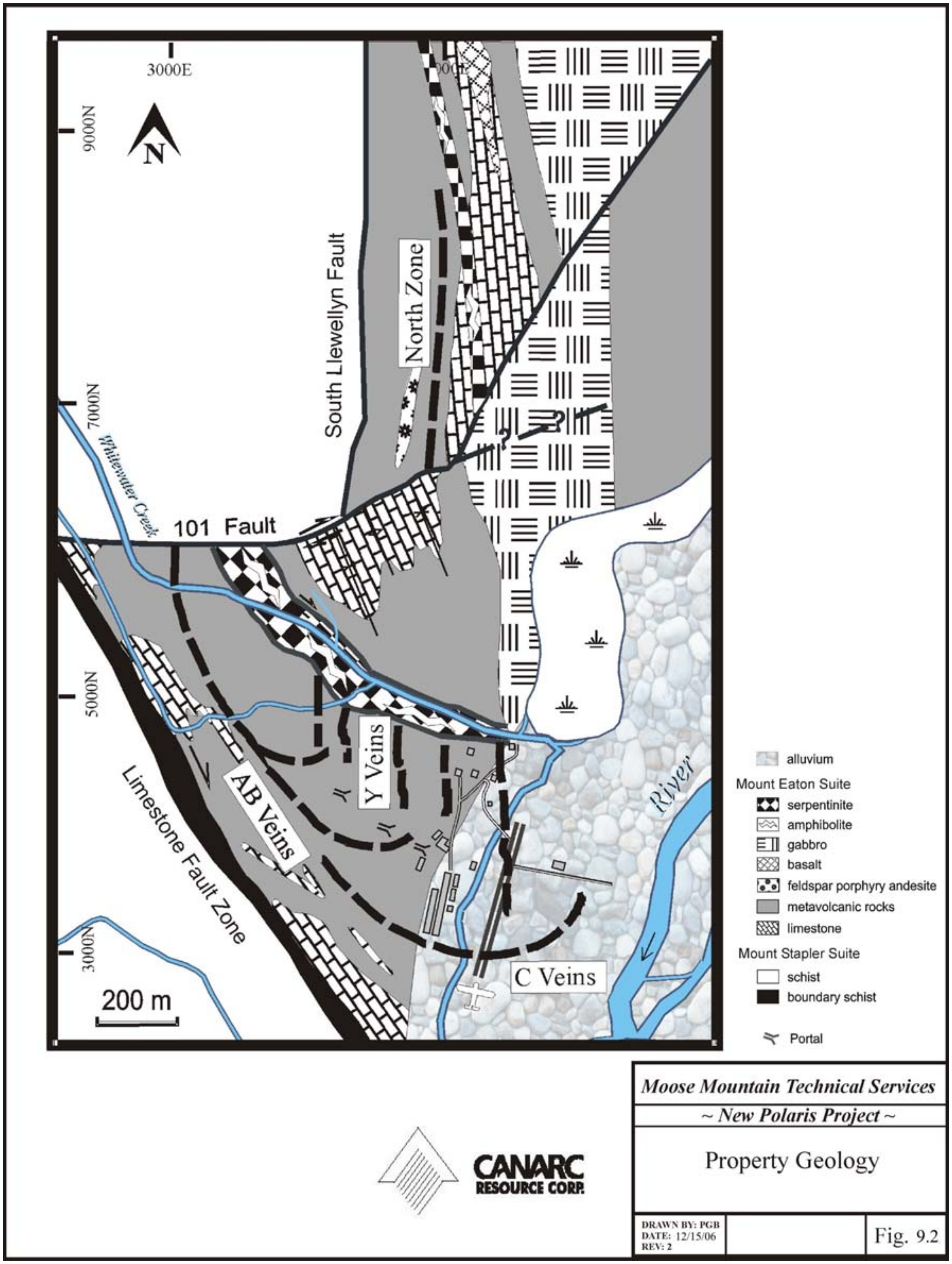


Figure 9-2 Property Geology

10.0 DEPOSIT TYPES

The New Polaris deposit is classified as a mesothermal lode-gold deposit (Hodgson, 1993).

> "In general, it is quartz-vein-related, with associated carbonatized wall rocks. The deposits are characterized by a high gold/silver ratio, great vertical continuity with little vertical zonation, and a broadly syn-tectonic time of emplacement. They are commonly associated with pyrite, arsenopyrite, tourmaline and molybdenite. Mineralization may occur in any rock type and ranges in form from veins, to veinlet systems, to disseminated replacement zones. Most mineralized zones are hosted by and always related to steeply dipping reverse- or oblique-slip brittle-fracture to ductile-shear zones."

Adapted from McClintock, 2006

> "The exploration target on the New Polaris project is orogenic lode gold deposits also known as Mesothermal vein deposits. Numerous examples of this type of deposit are known through out the work including the Cambell Red Lake deposits in Ontario and the Bralorne deposit in British Columbia. Past exploration studies have demonstrated that the New Polaris vein systems have all of the attributes of the orogenic vein gold deposit including, but not limited to association with major structural break, quartz-carbonate vein association, low-sulphide assemblage of pyrite and arsenopyrite, chloritic and sericitically altered wall rocks and persistent gold mineralization over a vertical distance of nearly 1 km."

The deposit type and model is considered appropriate for a Mesothermal lode-gold deposit.

11.0 MINERALIZATION

Adapted from McClintock, 2006

> "Mineralization of the New Polaris deposit bears strong similarities to many Archean lode gold deposits such as the arsenical gold camp of Red Lake, Ontario where the gold-bearing arsenopyrite is disseminated in the altered rock and in quartz-carbonate stringers.
>
> The vein mineralization consists of arsenopyrite, pyrite, stibnite and gold in a gangue of quartz and carbonates. The sulphide content is up to 10% with arsenopyrite the most abundant and pyrite the next important. Stibnite is fairly abundant in some specimens but overall comprises less then one-tenth of 1% of the vein matter. Alteration minerals include fuchsite, silica, pyrite, sericite, carbonate and albite.
>
> In general, the zones of mineralization ranging from 15 to 250 metres in length with widths up to 14 metres appear to have been deposited only on the larger and stronger shears. Their walls pinch and swell showing considerable irregularity both vertically and horizontally. Gold values in the veins have remarkable continuity and uniformity, and are usually directly associated with the amount of arsenopyrite present. The prominent strike directions are north-south and northwest-southeast, which is interpreted to be within a major shear zone. Up to 80% of the mine production was from "structural knots" or what is now known as "C" zones. In detail the "C" zones are arcuate structures. Figure 11-1 shows a 3D view of the "C" vein system.
>
> The vein mineralization has well marked contacts with the wall rock. The transition from mineralized to nonmineralized rock occurs over a few centimeters. The mineralization consists of at least 3 stages of quartz veining. The initial stage of quartz-ankerite introduced into the structure was accompanied by a pervasive hydrothermal alteration of the immediately surrounding wall rock. Arsenopyrite, pyrite and lesser stibnite were deposited with the alteration. Later stages of quartz-ankerite veining are barren and have the effect of diluting the gold grades in the structure. The sulphide minerals are very fine-grained and disseminated in both the wall rock and early quartz and ankerite veins. Free gold is extremely rare and to the end of 2005 had not been recognized in core samples. The majority of the gold occurs in arsenopyrite and to a lesser extent in pyrite and stibnite. Because there is no visible gold and the host sulphides are very fine-grained and disseminated there is little nugget effect and gold values even over short intervals rarely exceed 1 opt."

Mineralization was observed by Morris during the site visit both in drill core and underground. The description of the mineralization appears applicable to the New Polaris project.

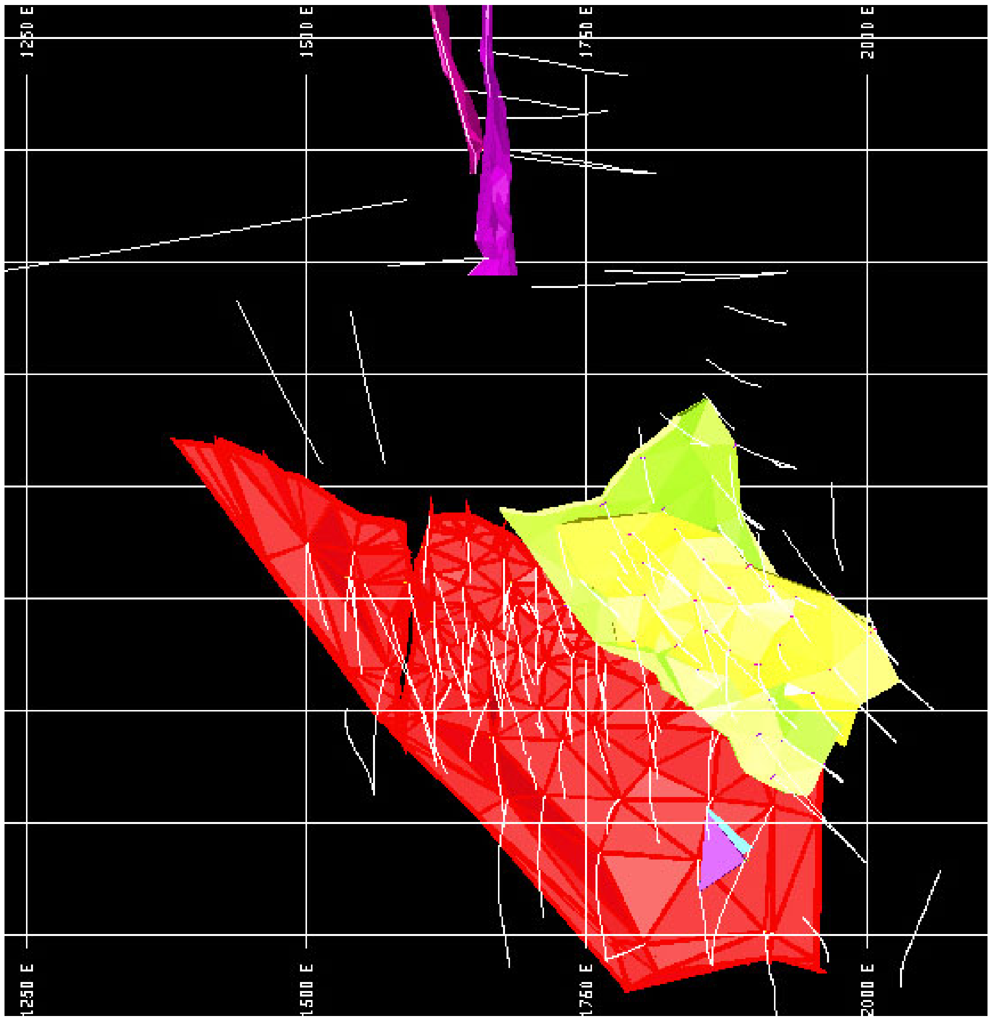


Figure 11-1 3D Model C Vein, plan view
(C vein segments are red, yellow, green, as well as the two small purple and blue; the two Y veins are in the north, shown as purple; the grid is 100m north/south and 250m east/west; shows only drillholes since 1992)

12.0 EXPLORATION

The New Polaris property represents an advanced exploration project on a former gold producer. The early exploration work in the area located gold mineralization on surface and subsequent exploration led to mining of approximately 689,500 tonnes of material grading 10.29 g/t gold. Recent exploration work, since 1988, has been directed at gaining knowledge about the geology of the area and expanding the resource base of the mineralized zones.

Geological mapping, geochemical surveys, geophysical techniques, and drilling have added considerable value to the project. Table 12.1 lists the relevant exploration work on the property along with contractor name and supervisor.

Surface mapping, geochemistry and geophysics over the "mine wedge" were completed by Orequest in 1988. Surface mapping and geochemistry, on the "north grid", were completed in 1993.

Underground exploration included the rehabilitation of the AJ Level in 1988 and the rehabilitation of all of the other levels, including the Polaris Portal, in 1996 and 1997. The underground rehabilitation also included a re-survey of the old workings so that the more recent surface work could be aligned with the old underground workings.

Only drill hole data has been used in this analysis. In total the database includes 1,056 drillholes with a total of 31,514 samples, of which 1,432 are within the mineralized zones (see section 19 for more details).

The procedures followed in the field and through the interpretation stage of exploration have been professional. Various crews under the supervision of professional geologists carried out the exploration work. It is considered that the reliability of the data obtained with exploration is very high.

Table 12-1 Exploration Employees / Contractors

Year	Supervisor	Drilling Contractors	Laboratory	Underground
1988	Cloutier		TSL	
1989	Cloutier		TSL	
1990	Cloutier		TSL	
1991	Marriott	Arctic	Min En/Chemex	
1992	Marriott	Arctic	Chemex	
1993	Marriott/Moors	Arctic	Chemex	
1994	Moors	Arctic/Falcon	Chemex	
1995	Moors	Arctic/Falcon	Chemex	
1996	Karelse/Watkinson	Advanced	Northern	Main Street
1997	Karelse/Watkinson	Advanced	Northern	Main Street
1998	-			
1999	-			
2000	-			
2001	-			
2002	Moors			
2003	Moors	Hy-Tech	ALS Chemex	
2004	Moors/Aspinall	Hy-Tech	ALS Chemex	
2005	Moors/Aspinall	Hy-Tech	ALS Chemex	
2006	Moors/Cote	Hy-Tech	ALS Chemex	

13.0 DRILLING

Adapted from McClintock, 2006

“Diamond drill programs have been carried out on the New Polaris Project since the project was reactivated in 1988 (Table 13-1). Initially, the drilling focused on the down dip and along strike extensions of the Y veins. This work showed that the Y veins, while good grade were narrow and less continuous than the AB vein system. It also showed that the Y vein system is comprised of about 12 separate veins all of which are narrow and of short strike length.

In 1990, drilling shifted to the area beneath the lowest most C vein stopes. This drilling found that the vein system continued to depth and that gold grades in the 0.3 to 0.45 opt range over an average true thickness of 3 m (10 feet) were present. From 1991 to 1993 most of the drillholes tested the C veins with fewer drilled on the Y vein system.

In 1994, the North Zone was discovered and was tested with a total of 30 drill holes during the 1994 and 1995 period. Although thicknesses of the North Zone are up to 6.7 m (22 feet), the grades are overall low (less than 0.2 opt). This combined with the limited extent due to structural termination of the zone by a fault resulted in a decision to terminate exploration of the North Zone.

Encouraging drill results from the C veins and to a lesser extent from the Y vein system led to further drilling on these two vein systems. Drilling on the C vein showed the veins to be open to depth and to have gold grades that ranged from 0.2 to 0.6 opt over true thicknesses of 3 m (10 feet). The increased interest in the C vein system was due to its greater continuity and thickness compared to the Y vein. The narrow width and lesser continuity of the Y vein system made it a secondary exploration target.

In 1996 and 1997 the Y, C and AB veins were explored from underground. The plan was to closely test the upper portions of the Y, C and AB veins in order to allow calculation of a resource that might form the basis for resumption of mining. The results of the underground drilling program were mixed. The underground workings for the most part were driven along the vein structures with few crosscuts from which holes could be drilled to cut the down dip and along strike extension of the veins. As a result, except for those holes that tested the area immediately below the workings, most cut the veins at shallow angles. The very shallow angles that in places approach parallel to the vein make the use of these intersections inappropriate for a resource calculation (An example is hole 97-44 that cut 34.1 m grading 0.42 opt). Despite the number of holes drilled during 1996 and 1997, the work did little to expand the extent of the mineralization in the AB, C or Y vein systems. The work did confirm that the mineralized shoots in the lower most stopes on the Y and C veins were open to depth. The tables below summarize the locations of the 1988 to 2005 drilling. Composites of assay results are listed for the C vein system holes that intersected significant mineralization.

Poor market conditions after 1997 made financing of the New Polaris Project difficult. Drilling restarted on the property in 2003 with the objective of testing the extent of the C vein mineralization

Godfrey Walton, P. Geo., at the request of Bradford J. Cooke undertook a review of the Polaris Project and recommended additional drilling in order to test the continuity of the “C” vein zone mineralization at depth below the lower most mine workings. To this end, limited drill programs were carried in 2003 to 2005. If successful, these programs would lead to the expanded drill program, to allow a higher confidence level for the mineral resource estimation, as recommended as a second phase by Walton.

The 2003 to 2005 exploration programs targeted the “C” vein extensions below the existing mine workings. Collar locations for the holes drilled during the 2003 to December 31, 2005 as well as relevant holes from earlier drilling are plotted on Figure 13-1. Because there are 220 drill holes on the property it is not possible to plot all of these holes in page size format and still be legible. A more complete plan map is presented in Appendix III. This map is in PDF format and can be expanded to see detail using Adobe Acrobat 7 reader. An inclined section showing the pierce points of drill holes and grades of the 2003 to 2005 drilling is presented in Figure 13-2. Cross sections of the C vein to assist the reader in understanding the attitude, grades and thickness of the C vein system are presented in Figures 13-3 through 13-5.

The results of the 2003 to 2005 drilling of the C vein system confirmed the continuity of gold mineralization and the vein structure between the earlier drilled holes. As can be seen in the sections below, drill results show the “C” vein system to be an arc-like structure oriented east-west in the west swinging to a northeastern strike in the east. The change in strike occurs across the No.1 fault. To the east of the No.1 fault, the vein splays into two or more branches. The dip of the vein system is to the south and southeast and has an average dip of about 50º, although east of the No.1 fault the vein appears to flatten and thicken in a simoid-like feature. The exact nature of the apparent flattening of the vein’s dip is not clear and requires additional drilling to be resolved.

The thickness of the C veins varies from 0.30m to a maximum of 15.2 m (50 feet). The thicker parts of the vein occur to the east of the No. 1 fault where the dip of the vein flattens due to an apparent folding of the vein.

Table 13-2 lists the core length of the vein material cut in the drill hole. Depending upon the angle of intersection, the true thickness ranges from 100% to about 70%. The average core length thickness of the intersections is approximately 4.5 m (13 feet) and the average grade is 14.4g/t (0.4 opt) gold. The estimated average true thickness of the vein is 3.0 m (10 feet).

All of the holes in this period were drilled from surface and intersected a similar geologic sequence. From the collar, the holes penetrated from 15.2 m to 79.2 m (50 to 260 feet) of overburden followed by interlayered ash and lapilli tuff, volcanic wacke, and foliated andesite. The C vein system crosscuts the strike of the volcanic and volcaniclastic rocks at steep angles.”

Table 13-1 Assay Composites C Vein System

Hole-ID	From	To	Length (m)	Au_Comp (g/t)
04-1707E1	201.93	203.03	1.10	10.20
04-1707E1	209.46	213.42	3.96	5.10
04-1707E2	222.08	224.73	2.65	10.10
04-1707E2	246.64	248.72	2.07	11.80
04-1737E1	184.25	190.80	7.77	17.13
04-1737E1	200.25	207.57	7.32	4.00
04-1737E1	231.19	235.37	4.18	13.60
04-1737E2	233.66	233.96	0.30	30.90
04-1737E2	256.64	259.08	2.44	9.20
04-300SW1	180.75	181.72	0.98	11.20
04-300SW2	181.20	181.66	0.46	5.50
04-300SW2	206.53	208.94	2.41	14.80
04-300SW3	196.38	203.61	7.22	7.20
04-300SW3	223.88	230.73	6.86	14.50
04-330SW1	167.18	168.80	1.62	4.00
04-330SW1	172.36	174.49	2.13	35.30
04-330SW2	168.86	171.60	2.74	5.40
04-330SW2	194.22	202.27	8.05	31.90
04-360SW1	162.76	163.67	0.91	5.10
04-360SW1	179.10	193.43	14.33	11.60
04-360SW2	191.20	195.38	4.18	25.70
04-360SW2	198.24	201.87	3.63	5.20
05-1676E1	185.62	191.20	5.58	12.10
05-1676E2	210.01	213.27	3.26	8.90
05-300SW4	200.13	200.86	0.73	13.40
05-300SW4	261.34	269.05	7.71	17.40
05-300SW5	215.80	218.30	2.50	14.10
05-300SW5	241.10	246.74	5.64	21.60
05-300SW6	233.54	240.79	7.25	16.70
05-300SW6	260.91	266.09	5.18	18.20
05-330SW3	213.94	223.11	9.17	9.90
05-330SW4	221.89	233.00	11.89	19.88
05-330SW5	230.89	246.28	15.39	8.57
06-1676E-6	284.8	292.2	7.4	8.0
06-1707E-3	232.7	235.8	3.1	17.4
06-1707E-3	247.0	250.0	3.0	5.9
06-1707E-3	253.0	254.9	1.9	10.5
06-1707E-6A	312.0	313.4	1.4	18.8
06-1737E-3	242.3	244.3	2.0	24.6
06-1737E-3	252.7	254.7	2.0	10.6
06-240SW-8	263.5	270.0	6.5	7.7
06-240SW-8	297.7	299.2	1.5	8.8
06-240SW-8	316.1	317.55	1.45	8.3
06-270SW-4	263.1	268.85	5.75	12.3
06-270SW-4	310.8	313.7	2.9	4.9

Hole-ID	From	To	Length (m)	Au_Comp (g/t)
06-1615E-8	346.4	352.6	6.2	44.7
06-1768E-2	259.2	262.7	3.5	17.9
06-1768E-3	329.6	332.1	2.5	9.1
06-270SW-2	201.6	207.5	5.9	23.0
06-270SW-3	239.5	252.15	12.65	7.4
06-270SW-3	262.0	264.0	2.0	17.5
06-330SW-11	361.0	363.5	2.5	6.6
06-1768E-1A	225.8	242.3	16.5	23.1
06-1646E-6	258.3	265.5	7.2	15.5
06-1676E-5B	257.2	271.2	14.0	10.9
06-300SW-8	303.3	338.3	35.0	8.9
06-330SW-9	326.4	333.0	6.6	8.3
06-330SW-10	330.5	333.25	2.75	12.9
06-330SW-10	373.85	376.45	2.6	17.7
06-1585E-8	391.2	392.05	0.85	7.7
06-1615E-2	142.1	147.3	5.2	25.0
06-1615E-7.5	351.15	354.2	3.05	22.5
06-1615E-9	439.5	451.9	12.4	16.1
06-1813E-1	300.1	303.1	3.0	16.7
06-1813E-2	313.3	314.9	1.6	12.7
06-1813E-2	331.5	333.0	1.5	6.4
06-1813E-2	336.3	343.5	7.2	18.2
06-1813E-2	338.2	339.1	0.9	38.4
06-1813E-3	332.6	333.6	1.0	6.3
06-1813E-3	392.5	394.35	1.85	6.0
06-1813E-3	405.7	407.25	1.55	7.3
06-1707DE-1	425.6	430.1	4.5	11.8
06-1768DE-1	473.3	474.0	0.7	12.6
06-1768DE-1	487.4	491.9	4.5	9.5
06-1768DE-2	538.4	548.6	10.2	7.1
06-1859E-2	297.5	299.1	1.6	14.9
06-1859E-2	336.2	338.3	2.1	15.6
06-1859E-2	417.4	420.8	3.4	11.0
P8918A	194.43	199.49	5.06	37.70
P91C01	251.83	258.17	6.34	7.80
P91C01	274.93	281.48	6.55	20.20
P91C02	263.35	268.10	4.75	25.70
P91C03	226.47	229.64	4.20	11.46
P91C04	282.55	288.74	6.19	4.80
P91C05	156.73	169.10	12.37	10.30
P91C06	228.66	231.31	2.65	22.40
P91C07	292.06	294.35	2.29	18.80
P92C08	124.05	127.35	3.29	8.90
P92C09	161.85	163.98	2.13	4.20
P92C12	216.26	218.08	1.83	4.20
P92C12	258.47	265.27	6.80	15.20
P92C13	190.56	192.39	1.83	15.00

Hole-ID	From	To	Length (m)	Au_Comp (g/t)
P92C14	246.10	248.69	2.59	6.90
P92C15	231.77	233.17	1.40	4.80
P92C15	236.89	238.66	1.77	18.70
P92C16	229.36	231.65	2.29	4.20
P92C17	209.70	213.48	3.78	21.80
P92C18	281.18	283.98	2.80	9.20
P92C19	282.64	285.51	2.87	23.80
P92C20	243.35	245.97	2.62	8.90
P92C20A	233.08	235.82	2.74	12.50
P92C21	172.15	178.06	5.91	18.70
P92C24	136.52	139.39	2.87	23.70
P93C26	108.02	110.95	2.93	26.60
P93C27	148.89	151.49	2.59	12.50
P93C28	140.27	142.43	2.16	10.50
P93C31	184.80	186.99	2.19	8.00
P94C33	266.43	269.53	3.11	6.90
P94C34	176.72	180.14	3.41	7.20
P94C36	261.21	262.28	1.07	6.40
P94C37	255.24	261.27	6.04	22.20
P94C38	255.54	257.13	1.58	20.10
P94C38	274.53	279.23	4.69	22.90
P94C39	262.01	266.58	4.57	28.70
P95C40	493.01	498.93	5.92	8.47
P95C40	727.77	735.24	7.47	11.31
P95C42	456.47	457.54	1.10	13.13
P95C43	482.04	484.51	2.47	15.89
P95C44	586.50	589.57	3.07	18.76
P95C44	640.08	641.70	1.62	12.41
P95C44	692.38	694.15	1.77	28.06
PC25A	137.31	141.00	3.69	5.60
PC44A	586.40	591.22	4.78	16.00
PC44A	726.40	730.60	4.18	6.00
PC9001	206.35	213.79	7.44	11.80
PC9002	277.31	279.96	2.65	29.30
PC9003	215.59	222.50	6.92	14.80
PC9003	227.17	232.87	5.70	31.20
PC9004	221.25	225.25	3.99	16.10
PC9005	189.43	190.96	1.52	18.90
PC9007	160.90	162.31	1.40	15.20

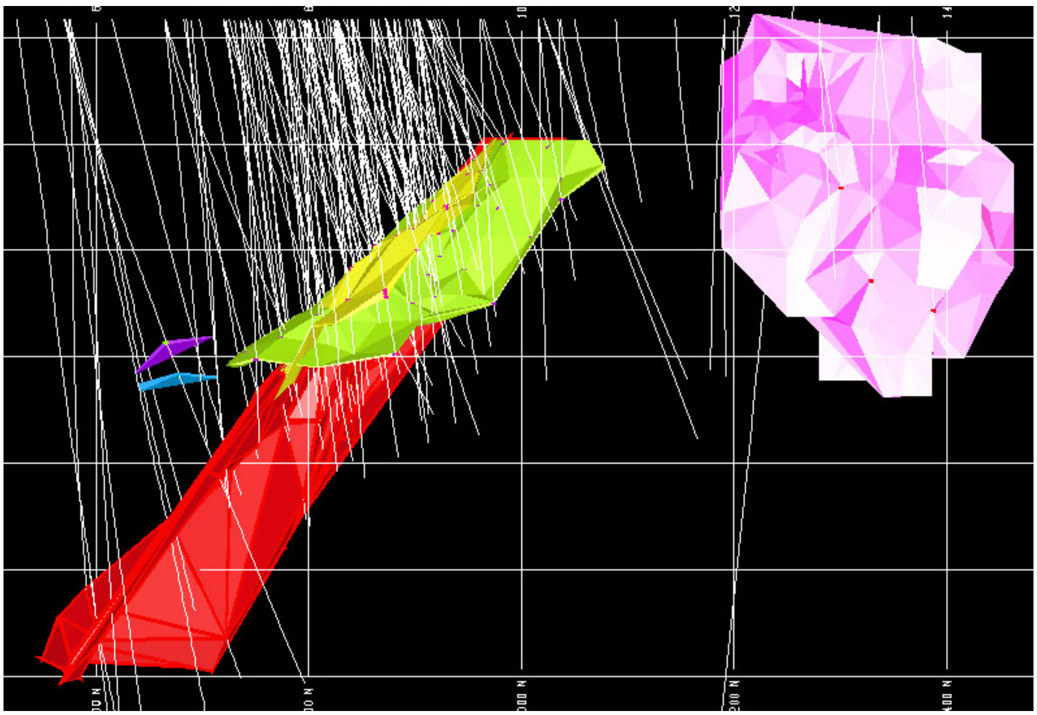


Figure 13-1 Drillholes, cross-section, viewed from the east
(C vein segments include; the red solid is CWM, green is CLOE, yellow is CHIE, the small blue and purple segments were not included in the resource estimate; the Y veins are to the north, shown as purple; the grid is 100m vertical and 200m horizontal; only drillholes since 1992 are shown)

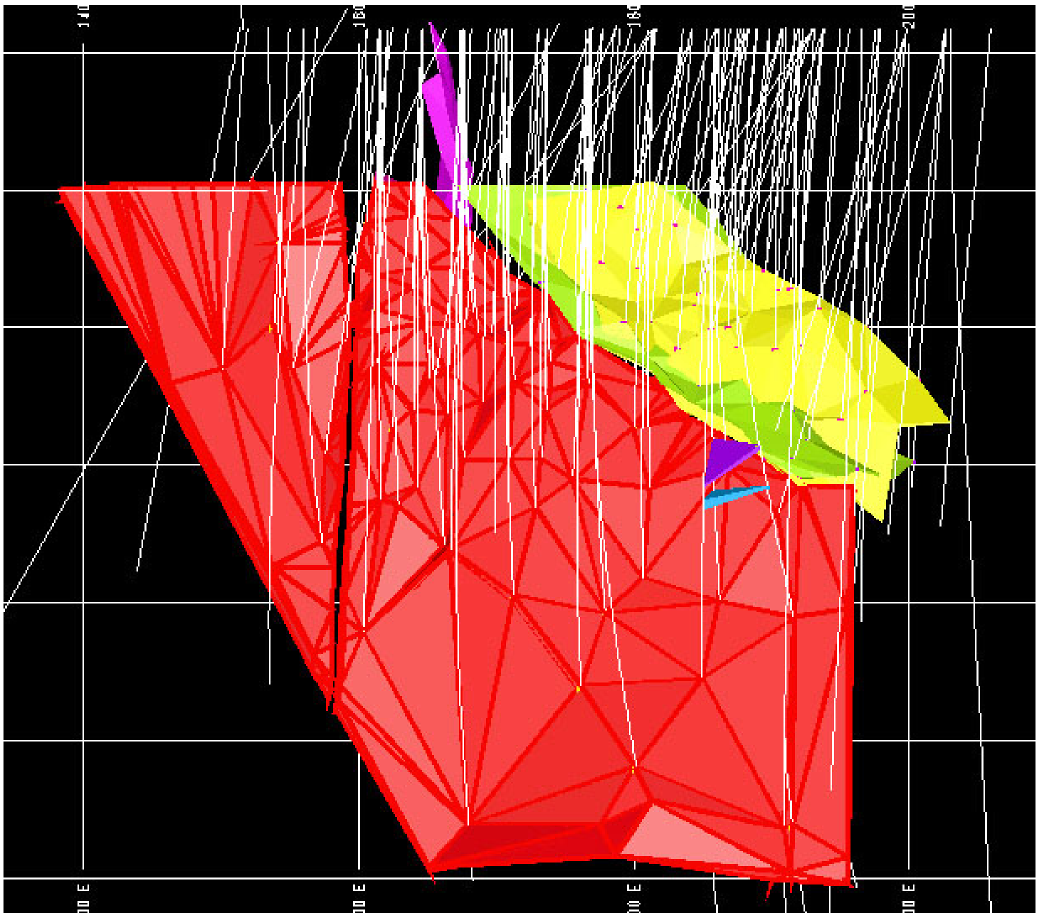


Figure 13-2 3D solids, cross-section, viewed from the south
(see Figure 13.1 for explanation)

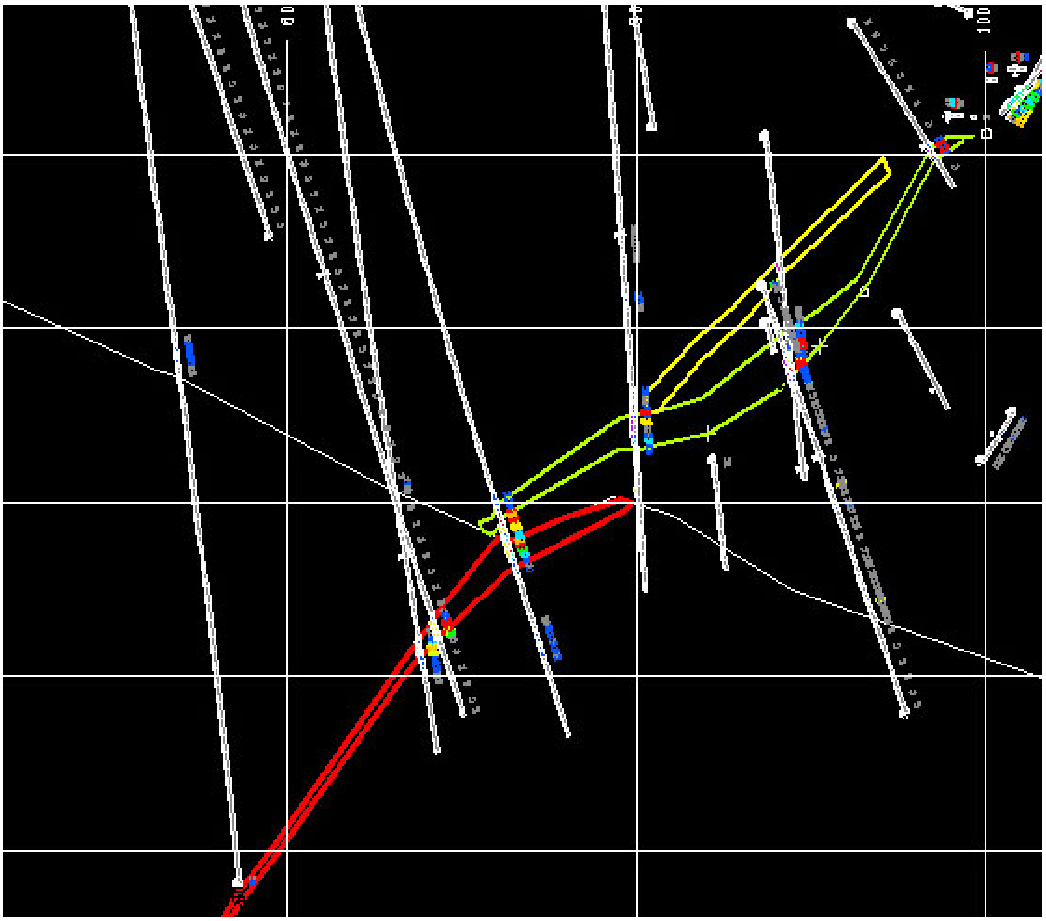


Figure 13-3 Cross-section East 1766
(shows Drillholes intersecting three segments of C vein, CWM is red, CLOE is green, CHIE is yellow; colour coding for the assay intercepts are >30g/t Au is red, 15-30g/t Au is orange, 10-15g/t is yellow, 5-10g/t Au is green, and light blue is 0-5g/t Au; the grid is 100m vertical and 200m horizontal; the white line separating the segments is No. 1 Fault)

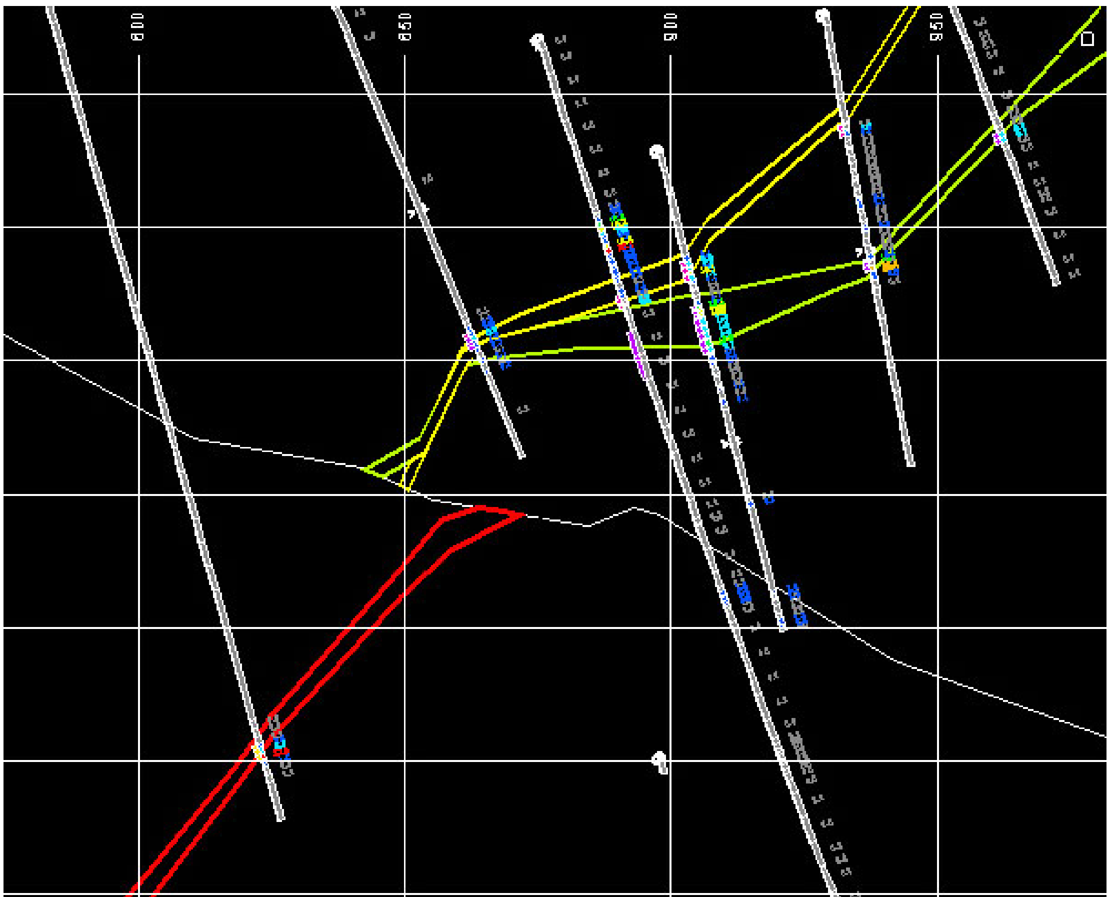


Figure 13-4 Cross-section East 1796
(see Figure 13.3 for explanation of colours)

14.0 SAMPLING METHOD AND APPROACH

Adapted from McClintock, 2006

“Sampling of the vein was done by wire line diamond drills using NQ-size rods. Drill collar locations were surveyed in by total station surveying method. Drilling azimuth and dip were set using a Brunton compass and clinometer. Routine downhole measurements of azimuth and dip were not done on the three holes drilled in 2003 and prior. In 2004, three different downhole survey systems were tried before settling on a Reflex system. The Reflex system was also used in 2005. The downhole surveying was operated by the Hytech’s drill crew. This information was entered into a GEMCOM program to plot the location of the collar and the pierce point of the veins.

Core recovery was very good and ranged from the low 90% to nearly 100% and is a fair sampling of the mineralization at the point where the drill hole pierced the vein.

The vein mineralization has well marked contacts with the wall rock. The transition from mineralized to nonmineralized rock occurs over a few centimeters. The mineralization consists of at least 3 stages of quartz veining. The initial stage of quartz-ankerite introduced into the structure was accompanied by a pervasive hydrothermal alteration of the immediately surrounding wall rock. Arsenopyrite, pyrite and lesser stibnite were deposited with the alteration. Later stages of quartz-ankerite veining are barren and have the effect of diluting the gold grades in the structure. The sulphide minerals are very fine-grained and disseminated in both the wall rock and early quartz and ankerite veins. Free gold is extremely rare and to the end of 2005 had not been recognized in core samples. The majority of the gold occurs in arsenopyrite and to a lesser extent in pyrite and stibnite. Because there is no visible gold and the host sulphides are very fine-grained and disseminated there is little nugget effect and gold values even over short intervals rarely exceed 1 opt. Out of 4,700 samples with greater than 0.03 opt gold collected from core and the underground workings, only 185 samples had a value greater than 1 opt, the highest being 3.69 opt. For this reason, no cutting of assays has been done in calculating composites nor are there many cases where a composite sample is carried by a single assay.

Determining intervals of core for sampling was done by the geologist during logging of the core. The mineralized vein structures were marked out. Selection of core intervals for sampling were based in the presence of veining and sulphide mineralization particularly arsenopyrite. Within the defined vein structure sample intervals ranged from 0.3 m to 1.5 m (1 foot to 5 feet). Divisions were based on intensity of mineralization and veining. Sampling of the core for 10’s of feet either side of the mineralized vein structures was also done to the point where hydrothermal alteration disappeared. No sampling of core from the unaltered rock was done.

The core was logged and stored in the camp. Access to the core was only available to the geologists and the core sampler. The core was brought from the drill to the logging facility by the geologist at the end of each shift. The core was geologically logged, recoveries calculated and samples marked out in intervals of 0.5 to 1 metre. The core was handed to the sample cutter who cut it with a diamond saw. Each sample was

> individually wrapped in plastic bags for shipment. The sample intervals were easily identified and correlate well with the drill logs."

Core logging, sample layout, cutting and bagging procedures were observed by Morris during the site visit. The procedures in the field remained as described by McClintock and follow accepted engineering standards.

Table 13.3, in the previous section, lists the relevant composite samples with gold values and sample length. True widths of the mineralized zone varies from 70% to 100%

15.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

15.1 2006 Program

The 2006 QA/AC program was similar to the previous programs in that samples were collected by employees of Canarc on site and shipped to ALS Chemex in Vancouver. For quality control and quality assurance, core samples were regularly mixed with blanks, duplicates, and standards. The program in the field was run in an efficient and proper manner following accepted engineering standards.

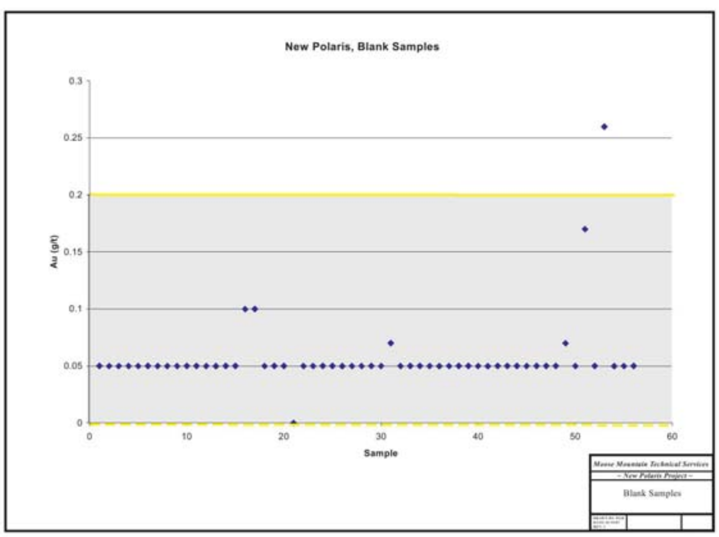
Blank samples represent material from the old mine, which is known to have a very low gold value. In total 56 blank samples were assayed. The sample statistics are shown in Table 15.1.

Table 15-1 Univariate Statistics, Blank Samples

Parameter	Result
Population	56
Minimum value	0.05
Maximum value	0.57
Mean value	0.07
Standard Deviation	0.08
CV	1.12

Three samples had gold values greater than three times the detection limit for gold (Sample C090930 with 0.57g/t, sample C 090800 with 0.26g/t, and sample C090770 with 0.17g/t). Figure 15.1 shows the test results of the blank samples (excluding the highest value sample). The three samples, and others from the same batch, should be re-tested by the laboratory.

Figure 15-1 Blank Samples, 2006



Duplicate samples were made by cutting ½ of the drill core into ¼ core and submitting the quarters as two different samples. In total 45 duplicate samples were assayed. The sample statistics are shown in Table 15.2.

Table 15-2 Univariate Statistics, Duplicate Samples

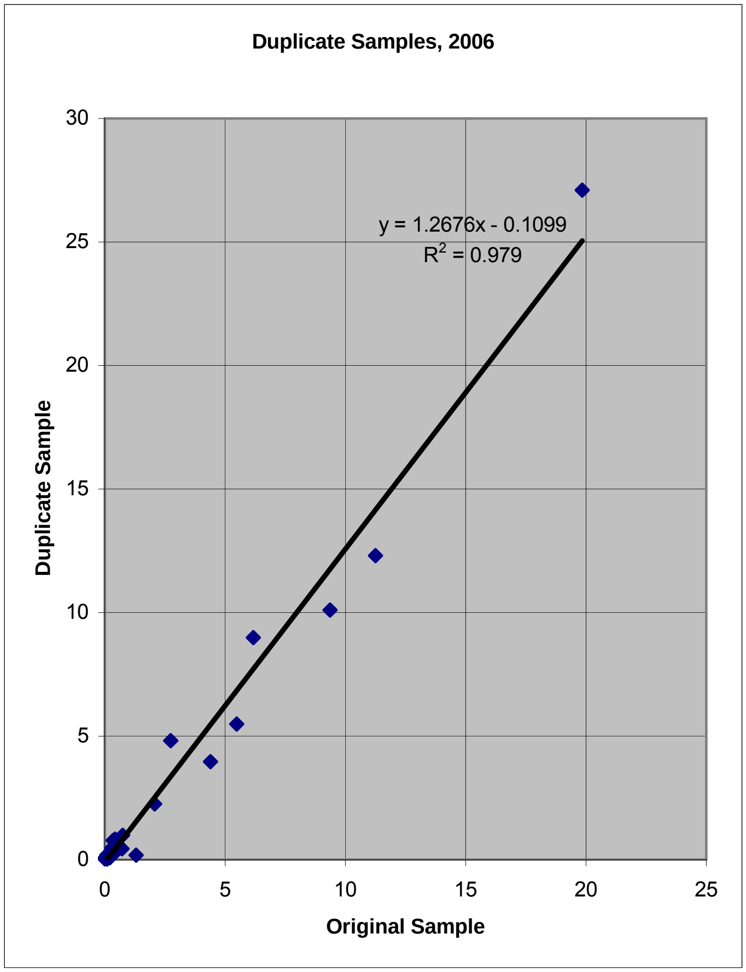
Parameter	Result, First Sample	Result, Duplicate Sample	Result, Sample Difference
Population	45	45	45
Minimum value	0.025	0.025	-1.1
Maximum value	19.85	27.1	7.25
Mean value	1.52	1.81	0.30
Standard Deviation	3.72	4.76	0.18
CV	2.45	2.63	4.09

Three tests were completed to compare the duplicate sample results: a Student's t-test, which is a comparison of the mean values; an F-test, which is a comparison of the variance; and a Paired t-test, which is a test for bias. The Student's t-test shows that the means of the duplicate samples are likely to be the same (the calculated t-value is -0.32, well below the critical value of 1.96). The F-test shows that the variances of the two sample sets are likely to represent the same population (the calculated F-value is 0.61, within the range 0.065 and 1.68 of F-values characteristic of a normal distribution). The Paired t-test shows that the difference between the two data sets is minimal, and they are distributed around the value zero (the standard error is calculated to be 0.18, while the mean, 0.30, is between –0.065 and 0.657).

The three statistical tests indicate that the results of the duplicate sampling are acceptable.

Figure 15.2 shows the duplicate sample results. As shown, there is a strong correlation between the two sample sets, with a coefficient of correlation of 0.98.

Figure 15-2 Duplicate Samples, 2006



(values are Au g/t along both axes)

Three standard samples were submitted randomly for assay throughout the program to test the accuracy of the laboratory.

Table 15-3 Standard Samples

Standard	Mean (Au g/t)	Standard Deviation	Upper Range	Lower Range
PM 165	6.51	0.10	6.71	6.31
PM 415	2.37	0.12	2.49	2.13
PM 916	12.7	0.09	12.9	12.5

Figure 15-3 Standard PM 165, 2006

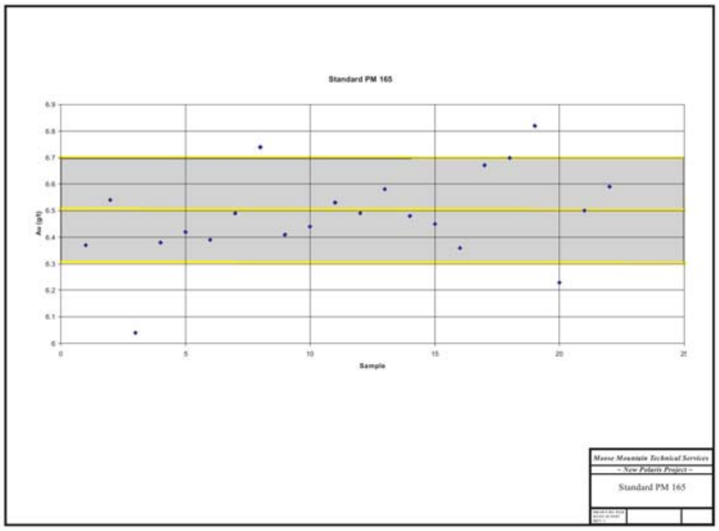


Figure 15.3 shows the results for standard PM 165. As shown, there are two samples with lower than acceptable values and two with higher than acceptable values. These samples, and others in the same batch, should be re-assayed.

Figure 15-4 Standard PM 415, 2006

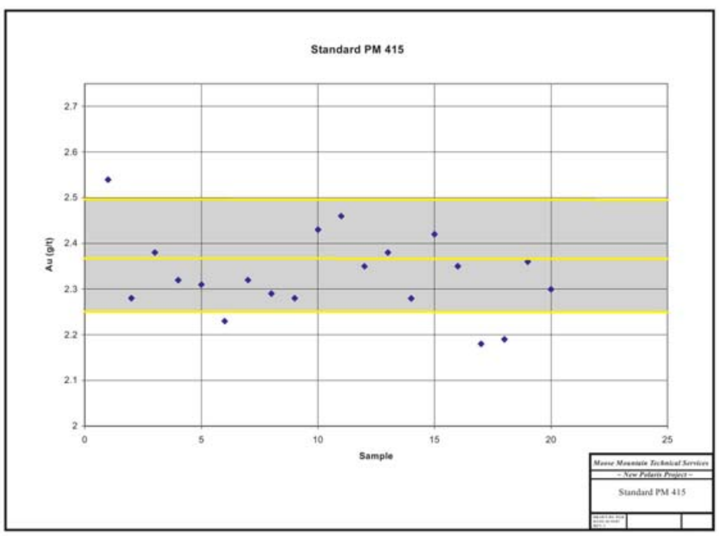


Figure 15.4 shows the results for standard PM 415. As shown, there is one sample with higher than acceptable values. This sample, and others in the same batch, should be re-assayed.

Figure 15-5 Standard PM 916, 2006

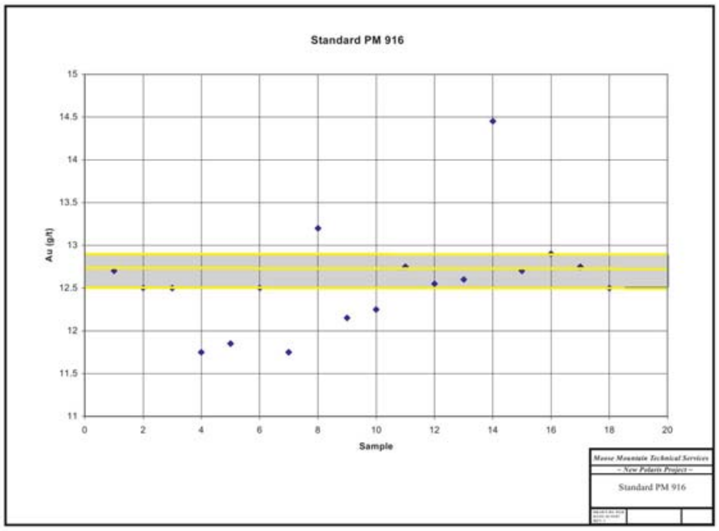


Figure 15.5 shows the results for standard PM 916. As shown, there are five samples with lower than acceptable values and two with higher than acceptable values. These samples, and others in the same batch, should be re-assayed.

A preliminary set of sample pulps was selected for re-assay by another laboratory. Acme Analytical Laboratory Ltd. (a highly accredited lab in Vancouver) was chosen as the second lab. The results are generally very consistent except for one sample in the Chemex Sample 1 set, which assayed 22g/t compared to 31.4g/t by Acme and 31.4g/t by Chemex the second time.

Table 15-4 Univariate Statistics, Round Robin Samples

Parameter	Chemex Sample 1	Difference Chemex 1 vs. Acme	Acme	Difference Acme vs. Chemex 2	Chemex Sample 2	Difference Chemex 1 vs. Chemex 2
Population	30	30	30	30	30	30
Minimum value	0.0	-9.43	0.01	-0.31	0.0	-9.4
Maximum value	40.7	0.36	42.11	2.04	41.4	0.8
Mean value	7.49	-0.50	7.99	0.19	7.80	-0.31
Standard Deviation	10.47	1.82	11.31	0.43	11.08	1.76
CV	1.40	3.65	1.42	2.25	1.42	5.70

Three tests were completed to compare the round robin sample results: a Student's t-test, which is a comparison of the mean values; an F-test, which is a comparison of the variance; and a Paired t-test, which is a test for bias. The Student's t-test shows that the means of the round robin samples are likely to be the same (the calculated t-value is -0.18 for the Chemex 1 vs. Acme samples; 0.11 for the Chemex 1 vs. Chemex 2 samples; and 0.07 for the Acme vs. Chemex 2 samples, well below the critical value of 1.96). The F-test shows that the variances of the three sample sets are likely to represent the same population (the calculated F-value is 1.04, for the Chemex 1 vs. Acme samples; 1.12 for the Chemex 1 vs. Chemex 2 samples; and 0.96 for the Acme vs. Chemex 2 samples, within the range 0.065 and 1.68 of F-values characteristic of a normal distribution). The Paired t-test shows that the difference between the Acme and Chemex 2 data set is minimal, and they are distributed around the value zero (the standard error is calculated to be 0.08, while the mean, 0.19, is between –0.065 and 0.657). In both cases, the Chemex 1 vs. Acme and Chemex 1 vs. Chemex 2 data sets, show a slight bias with the Chemex 1 samples being 0.32 to 0.33g/t lower than the Chemex 2 and Acme respectively.

The three statistical tests indicate that the results of the round robin sampling are acceptable.

Figure 15-6 Round Robin, Chemex 1 vs. Acme, 2006

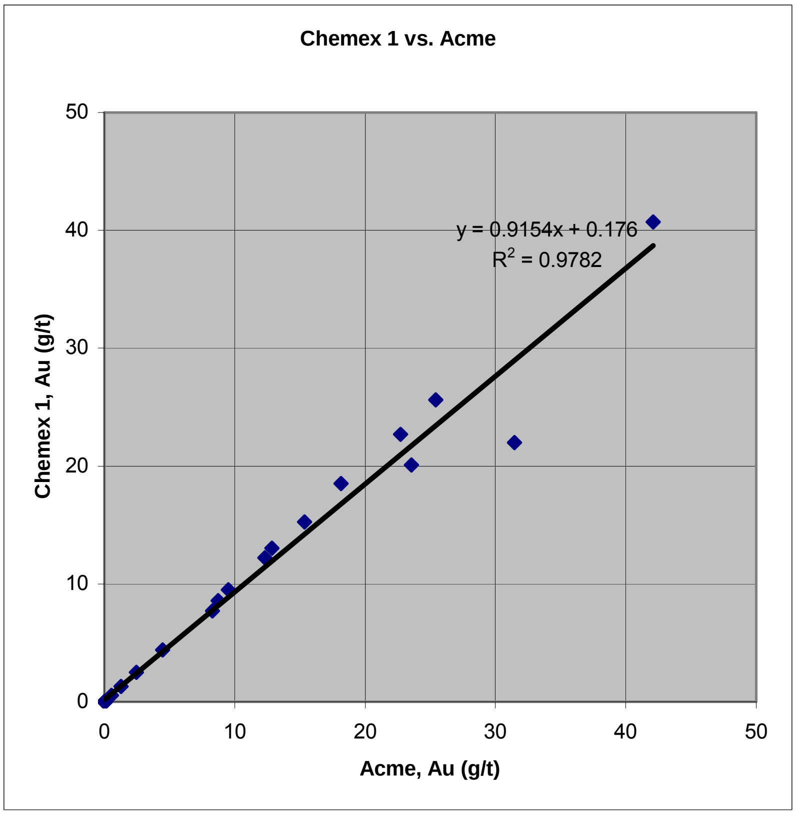


Figure 15-7 Round Robin, Chemex 1 vs. Chemex 2, 2006

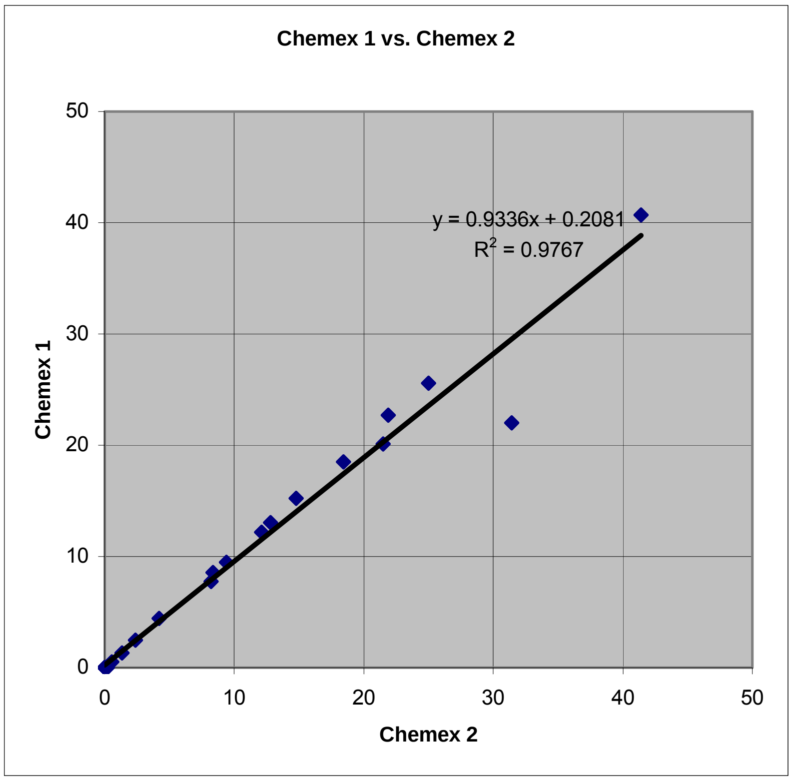
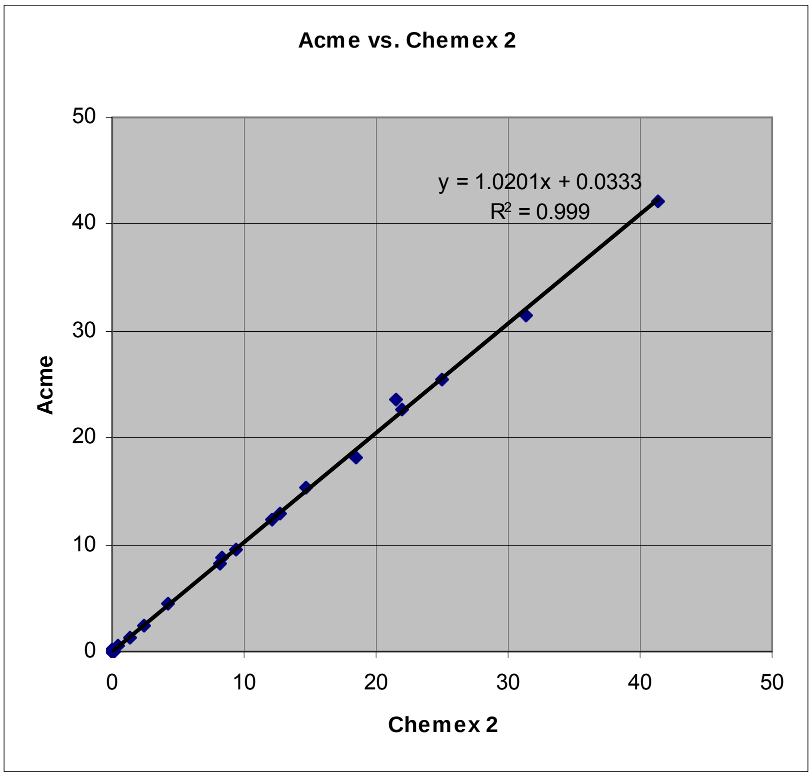


Figure 15-8 Round Robin, Acme vs. Chemex 2, 2006



16.0 DATA VERIFICATION

Morris spent two days on the New Polaris property. While on the property, the author examined underground workings to confirm the nature of mineralization, dimensions and extent of the vein system. The writer also viewed a selection of core from key holes drilled from the early 1990's to the end of 2006 and compared his observations with those documented in the drill logs. In both the case of the underground workings and the core, the author found that his observations confirmed that recorded in logs and sections. The author also confirmed that core had been properly cut and stored.

In addition to the site visit, a detailed review of the database was completed. Forty-one drill holes were selected from the C vein area, and the drill logs and assay sheets were compared with the database. Only minor differences were observed between the hard copy material and the database. As well, the input of the database into the modeling program was also checked.

The procedures used in the development of the database follow accepted engineering standards.

17.0 ADJACENT PROPERTIES

There are no adjacent properties with similar types of vein mineralization.

Redcorp Ventures Ltd. of Vancouver is developing the Tulsequah project and the nearby Big Bull deposit. In July of this year the first barge load of construction equipment was delivered to site and more loads were expected this summer.

The Tulsequah project is less than five kilometers north of the New Polaris project, up the Tulsequah River, while the Big Bull deposit is approximately six kilometers to the southeast. The development of the Tulsequah project should have a significant contributing impact on the New Polaris project with respect to access, infrastructure and other local supply issues.

18.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Recent metallurgical test work has been done by RDI in 2003 and 2004 followed by the current work. In the current work a metallurgical test program was conducted in 2006 & 2007 by Process Research Associates on selected samples from diamond drill core. The program was supervised by Jasman Yee who used the data from the testing program to design the treatment facilities that will be needed. The following are excerpts of the report "Metallurgical Report - Canarc Resources Corp. New Polaris Project" Jasman Yee P. Eng. May 20, 2007 which is on file at Canarc.

18.1 Test Work Review

The initial metallurgical test work on the Polaris-Taku (New Polaris) ore was performed by RDI in 2002 to early 2003. This test program included the review of all the historical test work done dating back to 1935. The review indicated that the gold may be associated with arsenopyrite and was refractory to direct cyanidation for gold extraction. Fine grinding followed by flotation using a simple reagent suite of potassium amyl xanthate and pine oil resulting in recovering 88% of the gold in a flotation concentrate with an approximate ratio of concentration of 4 and about 22% carbonates in the concentrate. The objectives of the RDI program were to determine the deportation of gold in the various minerals, reconfirm the flotation conditions and to produce a flotation concentrate with the maximum gold recovery with the minimum amount of carbonates for bio-oxidation studies. A bulk sample of about 200 kg's of was provided for this program from the AJ level.

The results of the RDI work were then used in developing the test procedures for the current program which took place at PRA. The aim of the current program was to maximize the gold recovery and to produce a flotation concentrate grading about 150 g Au/t. Depending on the quantity and type of minerals diluting the concentrate grade, it was conceivable that an arsenopyrite/pyrite separation may be required to achieve the desired flotation concentrate grade target, if it was not achievable with just the removal of the gangue minerals.

18.2 Process Metallurgy and Metallurgical Balance Results

Based on the PRA locked cycle test results and the use of mechanical cells for all cleaning stages, the metallurgical balance is presented in the table below. The final flotation tails shown is a combination of the rougher/scavenger and the 1st cleaner tailings. The balance is based on the assayed feed grade of the composite tested. The gold grade of the composite sample tested has a lower grade than the composite for the projected mill feed grade of 12.5g Au/t.

Table 18-1 Metallurgical Balance (Lock Cycle Results)

	% Wt	Assays				Recoveries			
		g Au/t	%As	%Sb	%S	Au	As	Sb	S
Feed	100.0	9.70	2.02	0.78	3.18	100.0	100.0	100.0	100.0
Flot. Conc.	12.5	72.30	14.54	5.47	22.63	92.0	92.0	87.7	89.0
Flot. Tail	87.5	0.89	0.23	0.11	0.40	8.0	4.1	12.3	11.0

By using columns for cleaning which is proposed in the design of the plant, the projected metallurgical balance is presented in the table 3.2. Again it is based on the assayed feed grade of the composite sample tested.

Table 18-2 Metallurgical Balance (Projected)

	% Wt	Assays				Recoveries			
		g Au/t	%As	%Sb	%S	Au	As	Sb	S
Feed	100.0	9.70	2.02	0.78	3.18	100.0	100.0	100.0	100.0
Flot. Conc.	9.8	90.00	18.48	6.95	28.75	91.0	92.0	87.7	89.0
Flot. Tail	90.2	0.93	0.22	0.11	0.39	9.0	4.1	12.3	11.0

The second table provides the anticipated process metallurgy of the designed plant. There is no gravity circuit as the test work indicated there is little or no gravity gold in the sample tested. The gold bearing concentrate produced by the plant will be shipped off site for further processing.

However, column flotation test work is required to confirm that the higher concentrate grades can be achieved.

The projected gold recovery is similar to the locked cycle test work with the potential of higher recovery if the projected mill feed is greater than 9.7g Au/t. In addition if the sulphur and the antimony grades are lower, then there is the likelihood of higher grade flotation concentrates.

19.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

19.1 Data Analysis

The database for the New Polaris resource estimated consisted of 1,056 diamond drill holes with a total of 31,514 sample intervals.

A total of 1,432 gold assays were within the mineralized zones outlined by Canarc geologists. The gold grade distribution is shown below in Figure 19.1 as a lognormal cumulative probability plot. Partitioning this plot shows 6 overlapping lognormal gold populations with mean grades and proportions of the total data set shown in Table 19.1.

Table 19-1 Summary of gold populations within mineralized zones

Population	Mean Au (g/t)	Proportion	Number of Assays
1	99.18	0.42 %	6
2	27.42	18.44 %	264
3	10.61	27.84 %	399
4	4.22	15.75 %	226
5	1.50	9.44 %	135
6	0.09	28.12 %	402

Population 1 consisting of only 0.42 % of the data are widely spaced throughout the mineralized zone and as a result can be considered erratic. A cap of 2 standard deviations above the mean of population 2, a level of 63 g/t Au, was used to cap 10 assays. The statistics for all assays both capped and uncapped are shown below in Table 19.2.

Table 19-2 Summary of gold statistics for uncapped and capped assays within mineralized zones

	Uncapped Au (g/t)	Capped Au (g/t)
Number of Assays	1,432	1,432
Mean Au	10.43	10.21
Standard Deviation	13.88	12.59
Minimum	0.001	0.001
Maximum	118.5	63.0
Coefficient of Variation	1.33	1.23

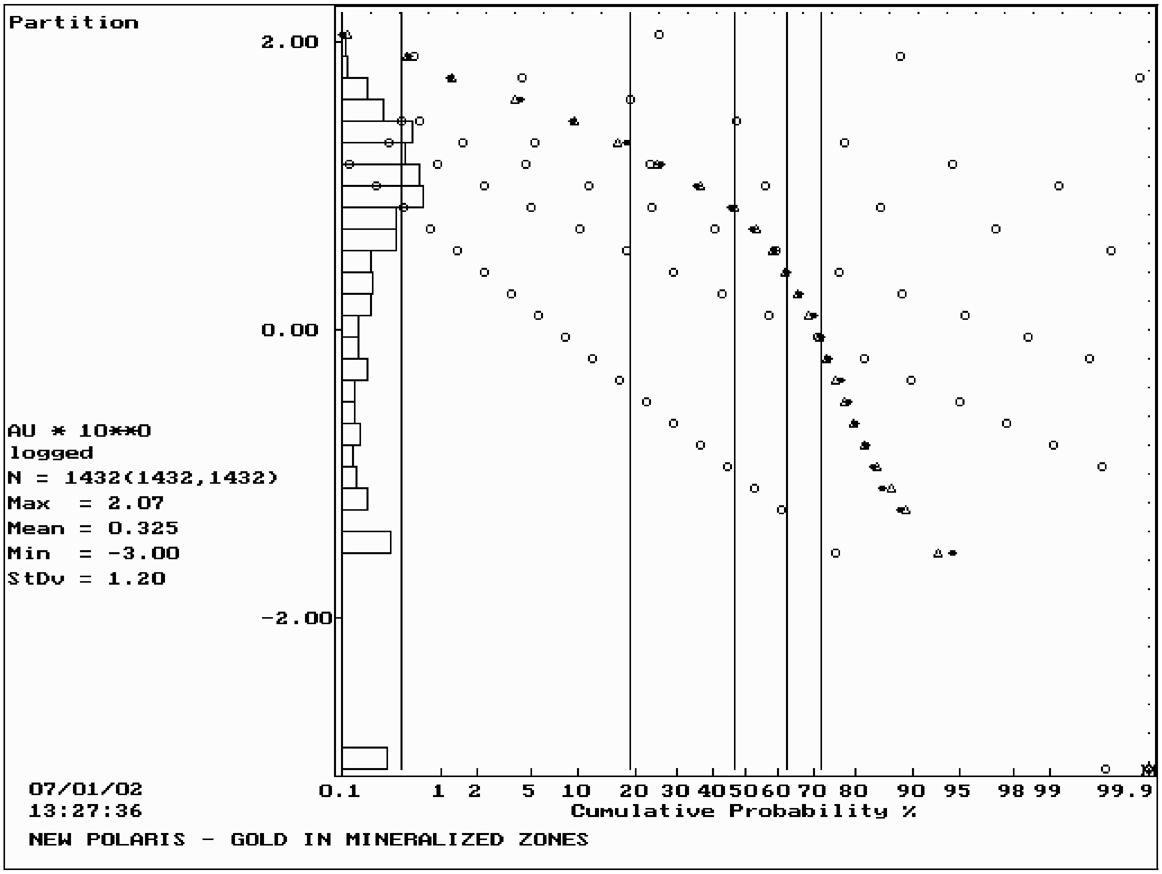


Figure 19-1 Gold Grade Distribution

19.2 Geologic Model

Using cross sections, Canarc geologists developed a three dimensional geologic model of the mineralized veins. The mineralized veins were subdivided into the following solids:

Table 19-3 Summary Description of Geologic Domains

Solid	Description	Number of drill hole intersections	Mean Au (g/t)
1	C High Vein East of Fault (CHIE)	186	6.80
2	C Low Vein East of Fault (CLOE)	439	12.09
3	C Vein Hanging Wall fragment	4	3.71
4	C Vein Hanging Wall fragment	4	14.67
5	C Vein West of Fault (CWM)	433	12.34
6	C Vein Mineralized Fragment	190	5.42
7	Y 19 Vein Striking N-S	133	10.59
8	Y 20 Vein Striking Azim 340	43	11.92

Three dimensional solids were created in GEMCOM software to constrain these mineralized zones, Figures 19.2 and 19.3.

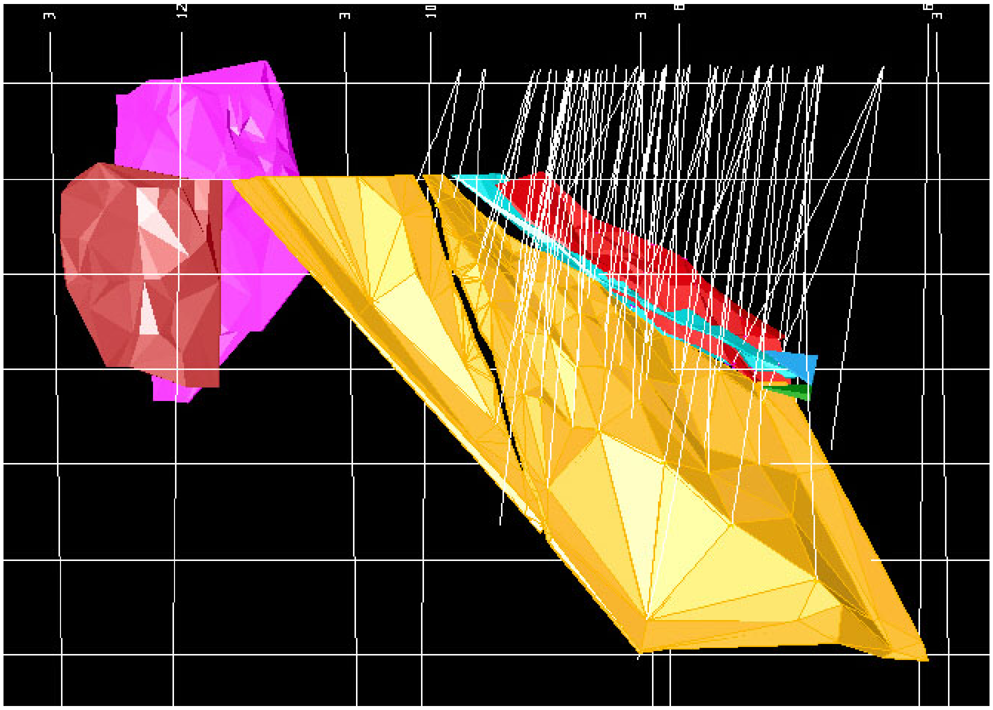


Figure 19-2 3D view, cross-section, looking to the northeast
(the gold coloured body is CWM, the “gap” is a cross-cutting felsic dyke, the red solid is CHIE, blue is CLOE, the small green and blue bodies are not used in the resource estimate, the two solids in the north are Y veins; the grid is 100m vertical; only 2006 drillholes are shown).

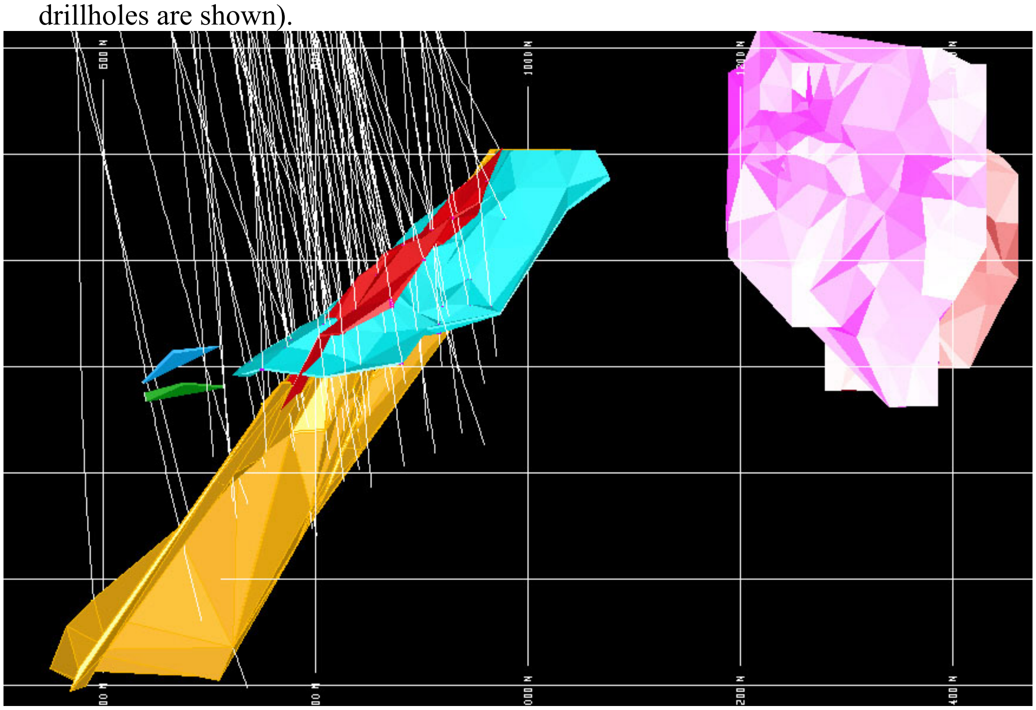


Figure 19-3 3D view, cross-section, looking to the west
(see Figure 19-2 for explanation of colours).

19.3 Composites

The drill holes were compared to the 3 dimensional solids and the point each hole entered and left each solid was recorded. Using these limits downhole composites 2.5 m in length were formed that honoured the domain boundaries. Intervals at the domain boundaries less than 1.25 m were combined with adjoining samples to produce composites of uniform support 2.5 ± 1.25 m in length. The statistics for each domain are summarized in Table 19.4.

Table 19-4 Summary of gold statistics in 2.5 m composites for geologic domains

Domain	1	2	3	4	5	6	7	8
Number	73	158	1	2	150	83	53	17
Mean Au (g/t)	5.62	10.11	3.61	13.88	11.38	4.61	6.23	13.60
Standard Deviation	5.12	9.59		0.59	9.25	7.81	7.64	11.40
Minimum	0.001	0.001		13.29	0.001	0.001	0.001	0.001
Maximum	24.37	44.09		14.47	46.30	42.89	35.50	36.34
Coefficient of Variation	0.91	0.95		0.04	0.81	1.69	1.23	0.84

19.4 Variography

Semi-variograms were produced along the three principal directions for each vein system; along strike, down dip and across dip. Where information on plunge of mineralization was available variograms were modeled in these directions. In all cases pairwise relative semivariograms were used and spherical models were fit to the experimental data. Domain 5 was split into an eastern segment and a steeper dipping western segment. The parameters for all models are summarized in Table 19.5. Individual models are shown in Appendix A.

Table 19-5 Summary of Semivariogram Parameters

Domain	Azimuth	Dip	Co	C_1	C_2	Short Range (m)	Long Range (m)
1	067	11	0.30	0.90		80	
	157	-48	0.30	0.90		60	
	337	-42	0.30	0.90		10	
2	069	4	0.40	0.55		80	
	159	-27	0.40	0.55		60	
	339	-63	0.40	0.55		12	
5 East	066	11	0.10	0.60		50	
	156	-50	0.10	0.60		90	
	336	-40	0.10	0.60		10	
5 West	087	14	0.10	0.60		50	
	177	-69	0.10	0.60		90	
	357	-21	0.10	0.60		10	
6	Omni Directional		0.60	0.75	0.10	10	60
7	0	0	0.20	0.80	0.40	12	40
	270	0	0.20	0.80	0.40	2	8
	0	-90	0.20	0.80	0.40	10	50
Waste	Omni Directional		0.40	0.75	0.28	25	50

Domains 3 and 4 had insufficient composites to model or estimate. Domain 8 also had too few composites to develop a model so the model for Domain 7 was used.

19.5 Bulk Density

A total of 87 specific gravity determinations were available for examination. All measurements were made from core drilled in 1996 and 1997. The list of results is contained in Appendix B. Unfortunately the majority of these samples were from parts of the deposit not included in this resource estimate (higher up on the C Vein). A total of 17 samples were measured from C Vein Domain 6 and 1 from Y Vein Domain 7. These 18 samples had an average specific gravity of 2.81 and this value was used for all estimated blocks.

More measurements should be taken from future drilling to better understand bulk density on this property.

19.6 Block Model

A block model consisting of 5 x 5 x 5 m blocks was superimposed on the geologic solids. The block model origin was as follows:

Lower Left Corner	1300 E	5 m block	180 columns
	450 N	5 m block	230 rows
Top of Model	300	5 m block	180 levels
No Rotation			

The block model was compared to the three dimensional geologic solids and the solid code and proportion of the principal solid in each block was recorded. In cases where two solids were present within the same block the minority solid was also recorded. Finally the amount of waste was calculated as 100 % - Prop. Solid 1 – Prop. Solid 2.

19.7 Grade Interpolation

Each geologic domain was interpolated separately using only composites from that domain. A gold grade was interpolated by ordinary kriging into all blocks containing some proportion of a geologic solid. A second gold grade, for blocks containing some proportion of a second solid, was then interpolated. Finally for blocks less than 100% within geologic solids, a gold grade for waste was interpolated using composites outside the geologic solids. The final gold grade for the block was a weighted average of these three values. Blocks within Domains 3 and 4 were not estimated since each domain contained only a single drill hole. Domain 5 was estimated in two sections to allow for the change in strike and dip. These two parts of Domain 5 and Domains 1 and 2 were all estimated using a soft boundary with both sides of the boundary seeing the same composites.

In all cases kriging was completed in a series of passes using expanding search ellipses whose dimensions and orientation were dictated by the semivariogram models. Pass 1 used dimensions for the search ellipse equal to ¼ the semivariogram range of the particular domain being estimated. Pass 2 expanded to ½ the semivariogram ranges while Pass 3 used the entire ranges. In an attempt to fill the limits of the geologic solids a fourth pass using twice the range of the semivariograms was completed in most cases. For pass 1 a minimum of 3 composites were required within the search ellipse to estimate a block. For passes 2 to 4 a minimum of 2 composites were required. In all cases the maximum number of composites used was 8 and if more than 8 were found, the closest 8 were used. The search ellipse directions and distances for each pass are shown below in Table 19.6.

Table 19-6 Summary of Kriging Parameters

Domain	Pass	Az/Dip	Dist. (m)	Az/Dip	Dist. (m)	Az/Dip	Dist. (m)
1	1	67/+11	20.0	157/-48	15	337/-42	5.0
	2	67/+11	40.0	157/-48	30	337/-42	5.0
	3	67/+11	80.0	157/-48	60	337/-42	10.0
	4	67/+11	160.0	157/-48	120	337/-42	20.0
2	1	69/+4	20.0	159/-27	15	339/-63	6.0
	2	69/+4	40.0	159/-27	30	339/-63	6.0
	3	69/+4	80.0	159/-27	60	339/-63	12.0
	4	69/+4	160.0	159/-27	120	339/-63	24.0
5 West	1	87/-14	12.5	177/-69	22.5	357/-21	5.0
	2	87/-14	25.0	177/-69	45.0	357/-21	5.0
	3	87/-14	50.0	177/-69	90.0	357/-21	10.0
	4	87/-14	100.0	177/-69	180.0	357/-21	20.0
5 East	1	66/+11	12.5	156/-50	22.5	336/-40	5.0
	2	66/+11	25.0	156/-50	45.0	336/-40	5.0
	3	66/+11	50.0	156/-50	90.0	336/-40	10.0
	4	66/+11	100.0	156/-50	180.0	336/-40	20.0
6	1	Omni Directional			15.0		
	2	Omni Directional			30.0		
	3	Omni Directional			60.0		
7	1	0/0	10.0	270/0	4.0	0/-90	12.5
	2	0/0	20.0	270/0	4.0	0/-90	25.0
	3	0/0	40.0	270/0	8.0	0/-90	50.0
	4	0/0	80.0	270/0	16.0	0/-90	100.0
8	1	340/0	10.0	250/0	4.0	0/-90	12.5
	2	340/0	20.0	250/0	4.0	0/-90	25.0
	3	340/0	40.0	250/0	8.0	0/-90	50.0
	4	340/0	80.0	250/0	16.0	0/-90	100.0
Waste	1	Omni Directional			12.5		
	2	Omni Directional			25.0		
	3	Omni Directional			50.0		
	4	Omni Directional			100.0		

19.8 Classification

Based on the study herein reported, delineated mineralization of the New Polaris Project is classified as a resource according to the following definition from National Instrument 43-101:

> "*In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by CIM Council on August 20, 2000, as those definitions may be amended from time to time by the Canadian Institute of Mining, Metallurgy, and Petroleum."*

> *"A **Mineral Resource** is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge."*

The terms Measured, Indicated and Inferred are defined in 43-101 as follows:

> *"A '**Measured Mineral Resource**' is that part of a Mineral Resource for which, quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity."*

> *"An '**Indicated Mineral Resource**' is that part of a Mineral Resource for which, quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed."*

> *"An '**Inferred Mineral Resource**' is that part of a Mineral Resource for which, quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes."*

For this project the geologic continuity has been well established through historic mining and diamond drilling. Grade continuity is best quantified by the semivariogram, which measures distances and directions of maximum continuity. For this study the classification for each block was based on the pass during which it was estimated which, was a function of the semivariogram range. In general, blocks estimated during pass 1 using ¼ of the semivariogram range, were classed measured although some of these blocks were later downgraded to indicated if they were based on a single drill hole. Blocks estimated during pass 2, using ½ the semivariogram range, were classed as indicated. All other blocks estimated during pass 3 or 4 were classed as inferred.

The results are presented as a series of grade-tonnage tables. Table 19.7 reports all material within the mineralized solids (undiluted) below -95 elevation in the CWM vein and below -140 elevation in the CLOE and CHIE veins (the lowest elevations of the old mine).

Table 19-7- New Polaris Resource, Undiluted
BLOCKS IN CWM BELOW –90m BLOCKS IN CLOE CHIE BELOW –135m ELEVATIONS

New Polaris Resource - Only Mineralized Portions of Blocks Tabulated						
	MEASURED RESOURCE			INDICATED RESOURCE		
Au Cutoff	Tonnes > Cutoff	Grade>Cutoff	Contained Metal	Tonnes > Cutoff	Grade>Cutoff	Contained Metal
(g/t)	(tonnes)	Au (g/t)	Au (ozs)	(tonnes)	Au (g/t)	Au (ozs)
1.00	410,000	9.09	120,000	1,330,000	10.61	454,000
2.00	390,000	9.48	119,000	1,280,000	10.97	451,000
3.00	360,000	10.13	117,000	1,240,000	11.23	448,000
3.50	340,000	10.37	113,000	1,220,000	11.41	448,000
4.00	330,000	10.62	113,000	1,180,000	11.65	442,000
4.50	320,000	10.87	112,000	1,130,000	11.95	434,000
5.00	304,000	11.19	109,000	1,090,000	12.21	428,000
5.50	287,000	11.55	107,000	1,057,000	12.45	423,000
6.00	271,000	11.89	104,000	1,017,000	12.71	416,000
7.00	233,000	12.77	96,000	910,000	13.45	393,000
8.00	203,000	13.54	88,000	806,000	14.22	368,000
9.00	173,000	14.42	80,000	696,000	15.11	338,000
	MEASURED PLUS INDICATED RESOURCE			INFERRED RESOURCE		
Au Cutoff	Tonnes > Cutoff	Grade>Cutoff	Contained Metal	Tonnes > Cutoff	Grade>Cutoff	Contained Metal
(g/t)	(tonnes)	Au (g/t)	Au (ozs)	(tonnes)	Au (g/t)	Au (ozs)
1.00	1,740,000	10.25	573,000	2,090,000	10.36	696,000
2.00	1,670,000	10.62	570,000	2,060,000	10.52	697,000
3.00	1,600,000	10.99	565,000	1,990,000	10.77	689,000
3.50	1,560,000	11.18	561,000	1,960,000	10.90	687,000
4.00	1,510,000	11.42	555,000	1,925,000	11.03	683,000
4.50	1,450,000	11.71	546,000	1,824,000	11.40	669,000
5.00	1,400,000	11.99	540,000	1,752,000	11.68	658,000
5.50	1,340,000	12.26	528,000	1,696,000	11.89	648,000
6.00	1,288,000	12.54	519,000	1,628,000	12.15	636,000
7.00	1,143,000	13.31	489,000	1,473,000	12.74	603,000
8.00	1,009,000	14.08	457,000	1,340,000	13.27	571,000
9.00	870,000	14.97	419,000	1,149,000	14.06	519,000

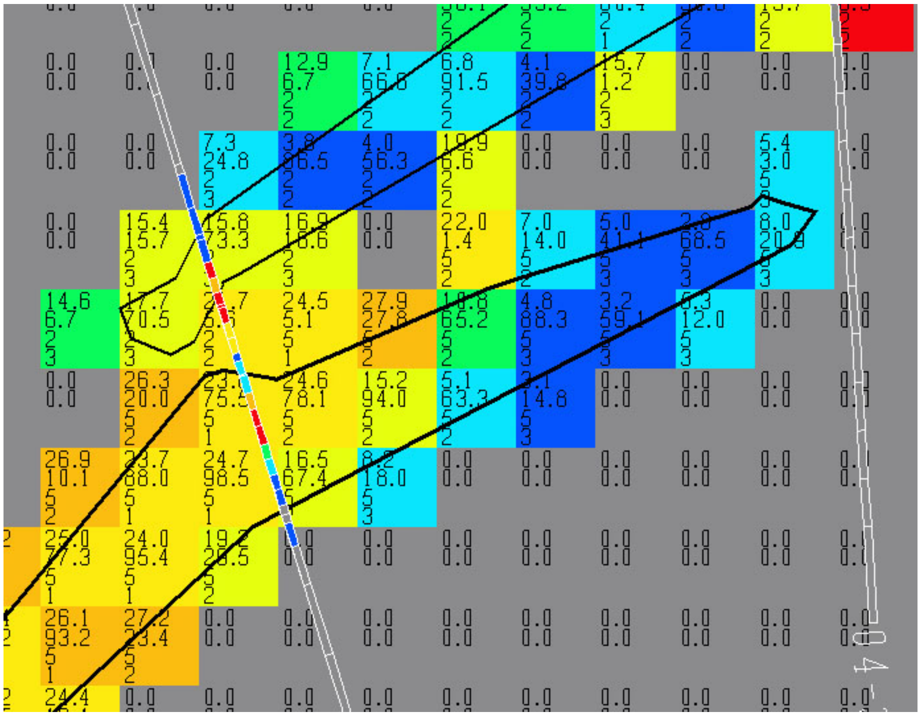


Figure 19-4 Cross-section 1766
Each block shows the Au grade, Ore %, Geology code, and Resource Classification (1=measured, 2=indicated, 3=inferred); the block size is 5x5x5m.

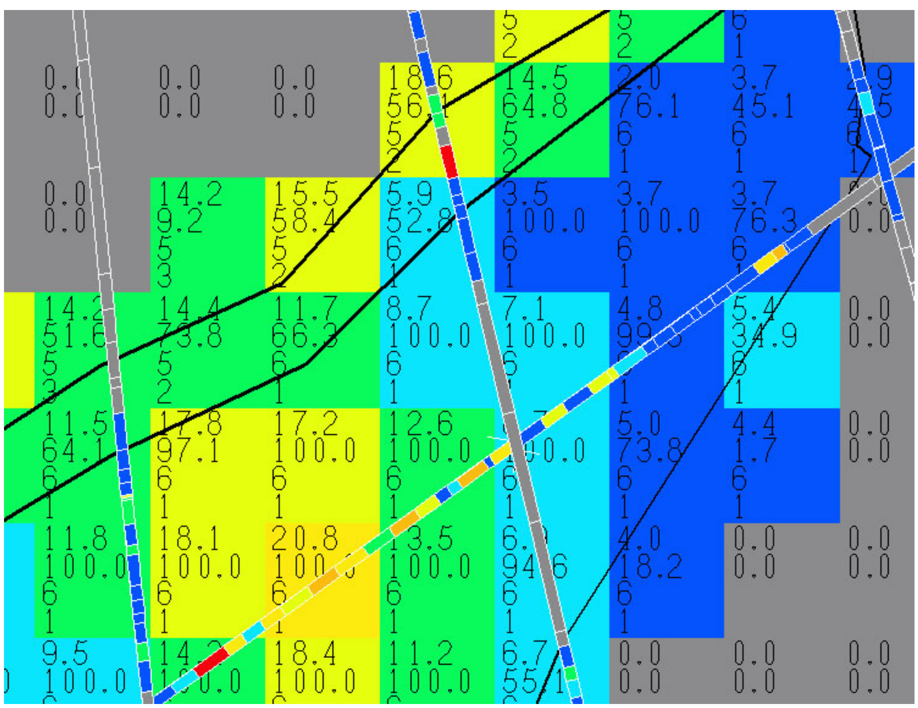


Figure 19-5 Cross-section 1706
(see Figure 19.4 for explanation)

19.9 Mining Resource Estimate

Based on the Resource model above, a mine plan has been developed to define a mineable resource. At this stage of the project a mine plan at a Scoping Study level of accuracy has been developed which is suitable for this Preliminary Assessment.

19.9.1 Mining Limits

The geology of the mineralization has been modelled, but a cutoff grade based on the break even point at which costs will equal revenues is applied to avoid zones of uneconomic material where possible. The cutoff grade is based on the following design factors:

Processing Costs = CDN$24/tonne.

G&A and Camp Costs = CDN$16/tonne

Surface Equipment and Facilities Costs = CDN$15/tonne

Ore to Plant Mining Cost = CDN$70/tonne

Total Combined Costs for the items above = CDN$125/tonne

Net Smelter Price = CDN$19.22/g, the calculation of this item is included in Appendix 1: NSP Calculation

Mining Dilution = 13%

Process Recovery = 91%

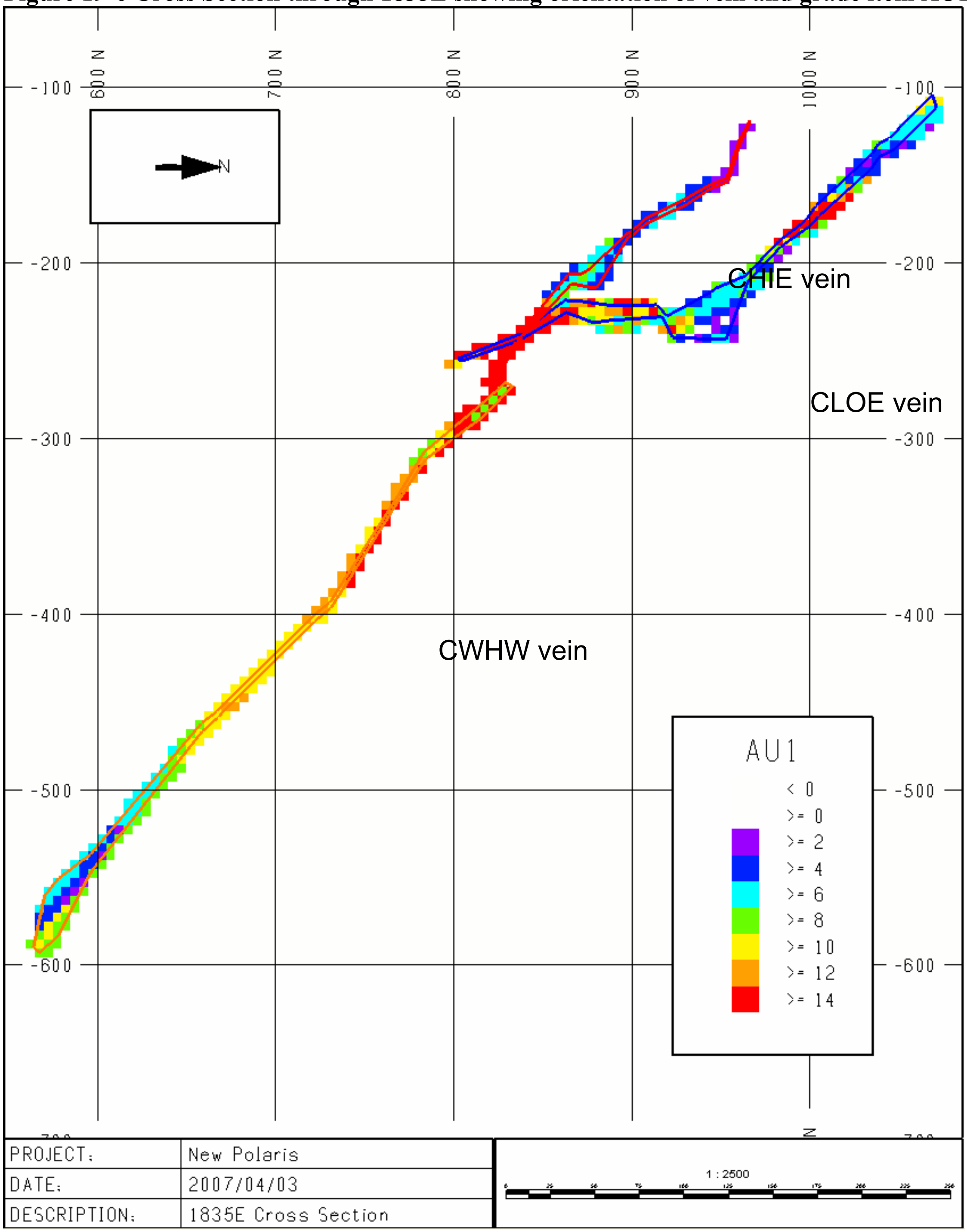
The cutoff grade is then = [(Costs to Delivery Ore to the Gate) / (NSP * Process Recovery)] * (1 + Dilution)

Cutoff Grade = [(CDN$125) / (CDN$19.22 * 91%)] * (1+0.13) = 8.1 g/t

Therefore, the goal is to mine out, using the mining methods chosen, all material grading higher than 8.1 g/t, and leave the material below this grade.

The following figure, shows a view of the block model, and the first grade item contained within the geologic veins in the block model. As described in the block model description above, there are three estimated gold grade items in each block in the model. The first grade item, AU1, represents the portion of the block that is contained within mineralization. If a block is five meters wide and the mineralization passing directly through the block is only two meters wide, the portion of the block that is mineralized is ~40%. This first grade item, AU1 represents the grade of the ~40% of the block that is mineralized. The average whole block grade would then be diluted over the other ~60% of the block. Therefore the first grade item, AU1, would be more representative than the whole block grade, AU, when running a cutoff grade determination. The second grade item, AU2, represents any portions of the block contained within a second vein of mineralization (occurs when two veins pass through the same block). The third grade item, AU3, represents the grade of material surrounding the mineralization outside the zone 3d solid. In the example above, it represents the grade of the remaining 60% of the block. The whole block grade, AU, is a weighted average of these three grade items, and represents the whole 5mX5mX5m block.

Figure 19-6 Cross Section through 1835E showing orientation of vein and grade item AU1.



A mining outline is created around all mining blocks grading ≥9 g/t, using the first grade item from the model as the basis for these cutoff drawings. This drawing is then clipped to the outlines of the geologic veins themselves, ultimately showing all material contained within the veins that include a ≥9 g/t mineralized grade. This clipped outline is shown in the drawing below, along with the location of the main shaft and '600 level' for reference.

These drawn out outlines are used as the mining limits, assuming all material within these outlines is mined out with an appropriate dilution factor which is described later in the report. It is assumed that stopes will fit within these limits, as appropriate.

Figure 19-7 All Material in C veins ≥9g/t, looking north

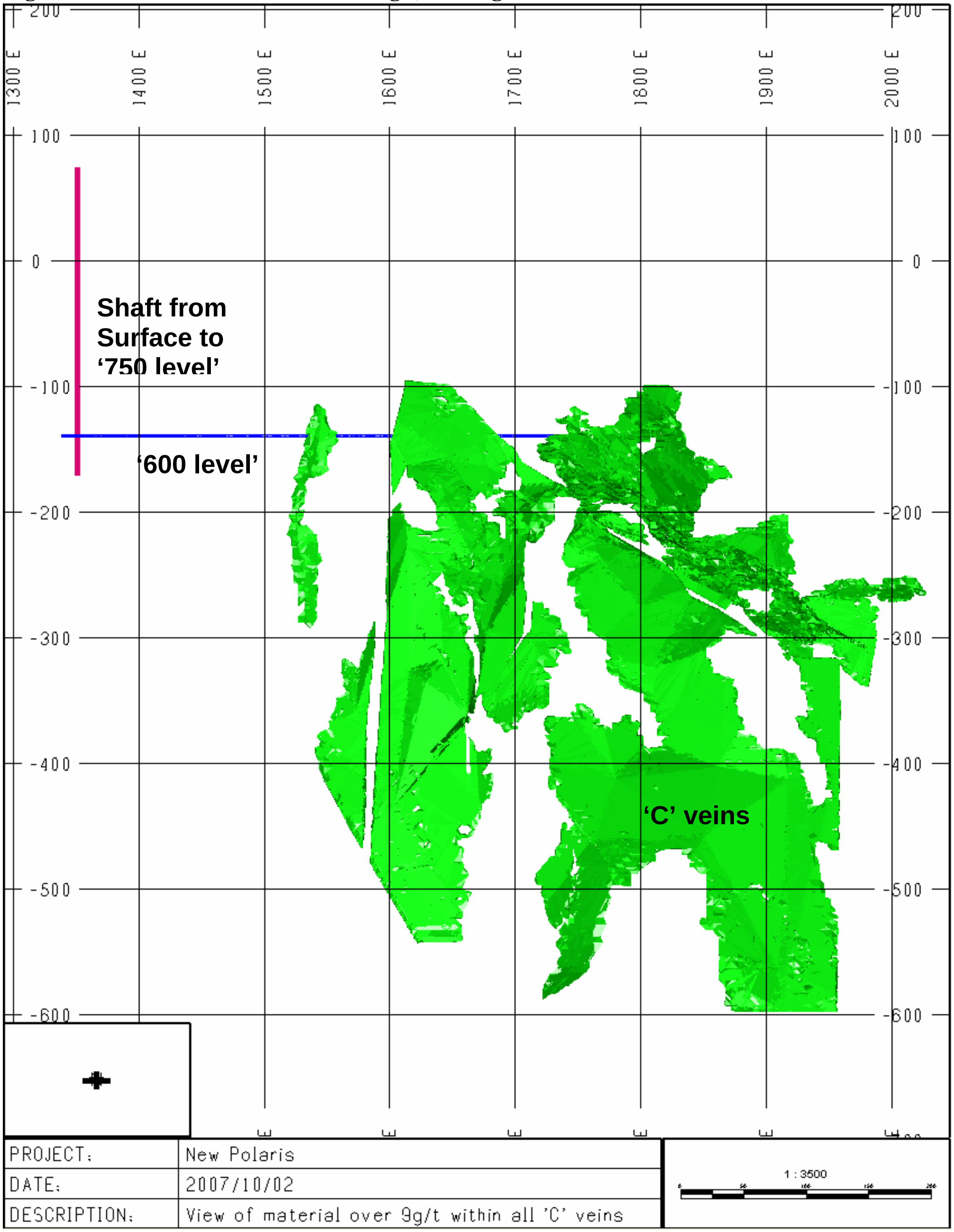
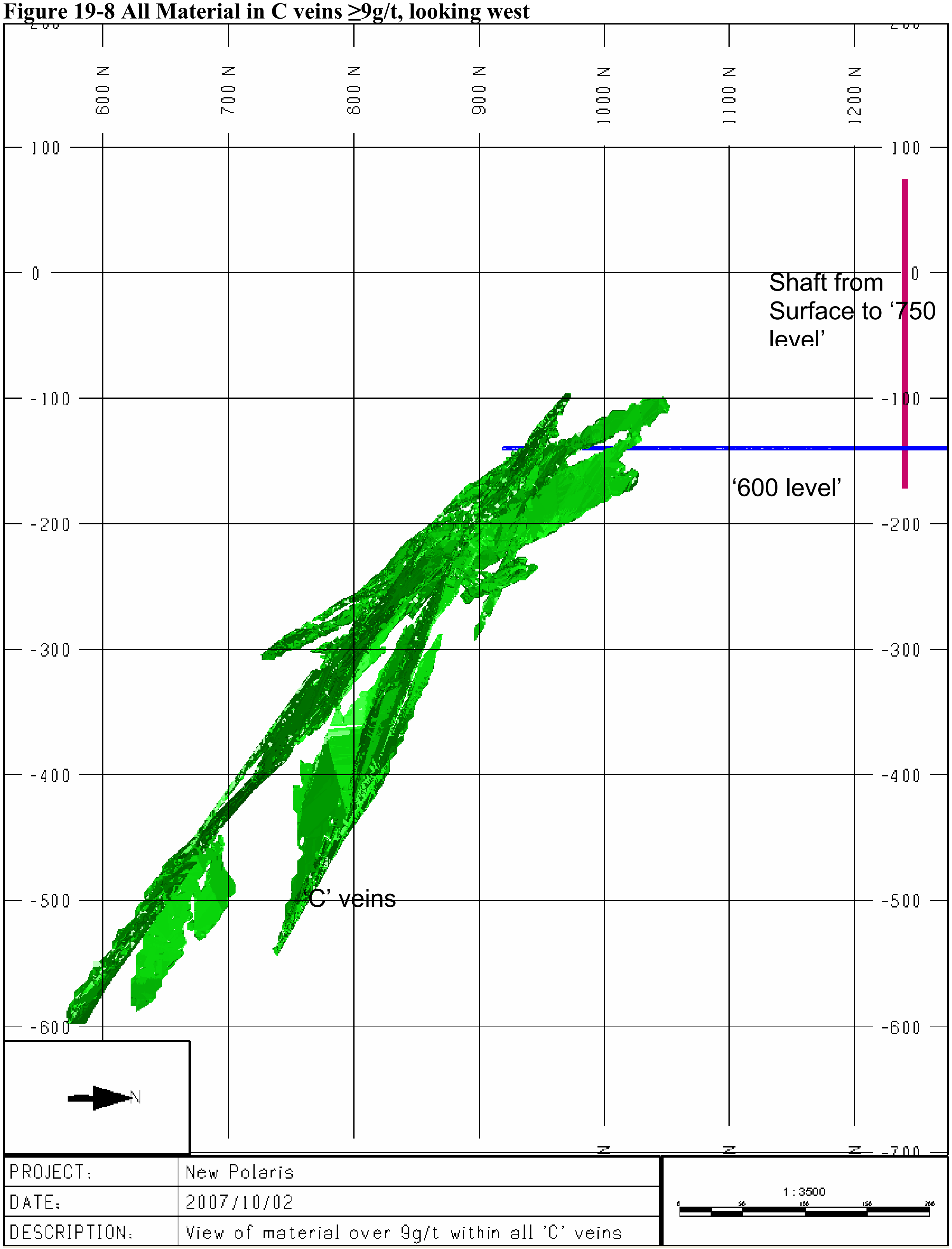
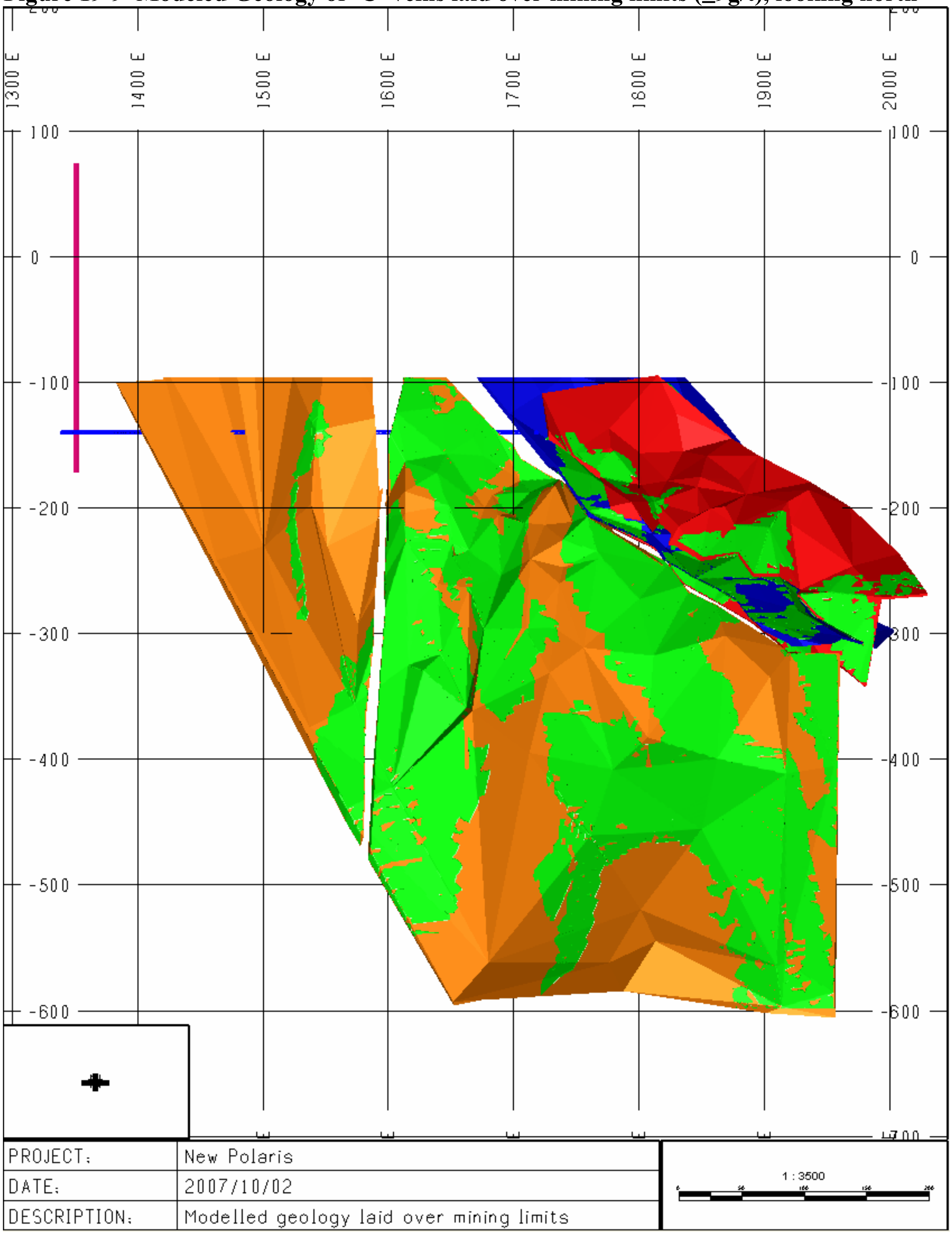


Figure 19-8 All Material in C veins ≥9g/t, looking west



The figure below shows an overlay of the mining limits to the modeled geologic outlines of the mineralized veins. This illustrates the areas of mineralization that are below 9g/t (not within the green mining limits) and therefore would not be targeted for mining unless future information warrants it.

Figure 19-9 Modeled Geology of 'C' veins laid over mining limits (≥9g/t), looking north



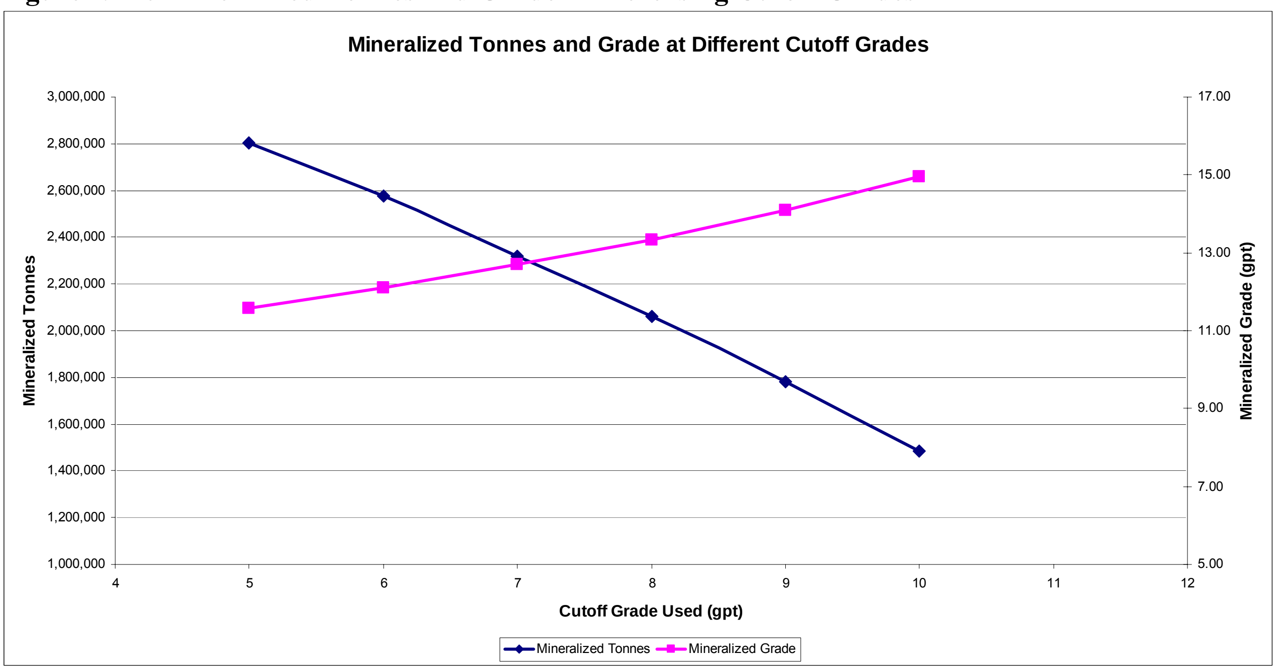
19.9.2 Effect of Altering Cutoff Grade

The effect of cutoff grade is analyzed in the table below summarizing the material contained in the model above each cutoff grade. The mineralized tonnage and the mineralized grade are weighted averages from the first two grade items in the model, AU1 and AU2 (if a second vein encountered). The surrounding grade is based on the third grade item contained in the model, AU3 and is used for estimating dilution.

Table 19-8 Amount of material and grade at different cutoff grades

Cutoff Grade	Mineralized kTonnes	Mineralized Grade	Surrounding Grade
5gpt	2,804.3	11.57	0.39
6gpt	2,577.3	12.09	0.39
7gpt	2,318.0	12.70	0.39
8gpt	2,061.2	13.33	0.38
9gpt	1,779.9	14.08	0.37
10gpt	1,485.0	14.96	0.36

Figure 19-10 Mineralized Tonnes and Grade at Increasing Cutoff Grades



As can be seen in the table and figure above, the rate of change in mineralized tonnes and grade does not vary when altering the cutoff grade. As the cutoff grade is increased, the mineralized tonnes decreases and the grade increases; but at no point is there a significant change in either value. Therefore there is no obvious selection for choosing a cutoff grade based on its greatest impact on mineralized tonnes or mineralized grade that will be mined. Rather, the cutoff grade is chosen purely on a basis of revenue minus costs.

19.9.3 Mining Dilution

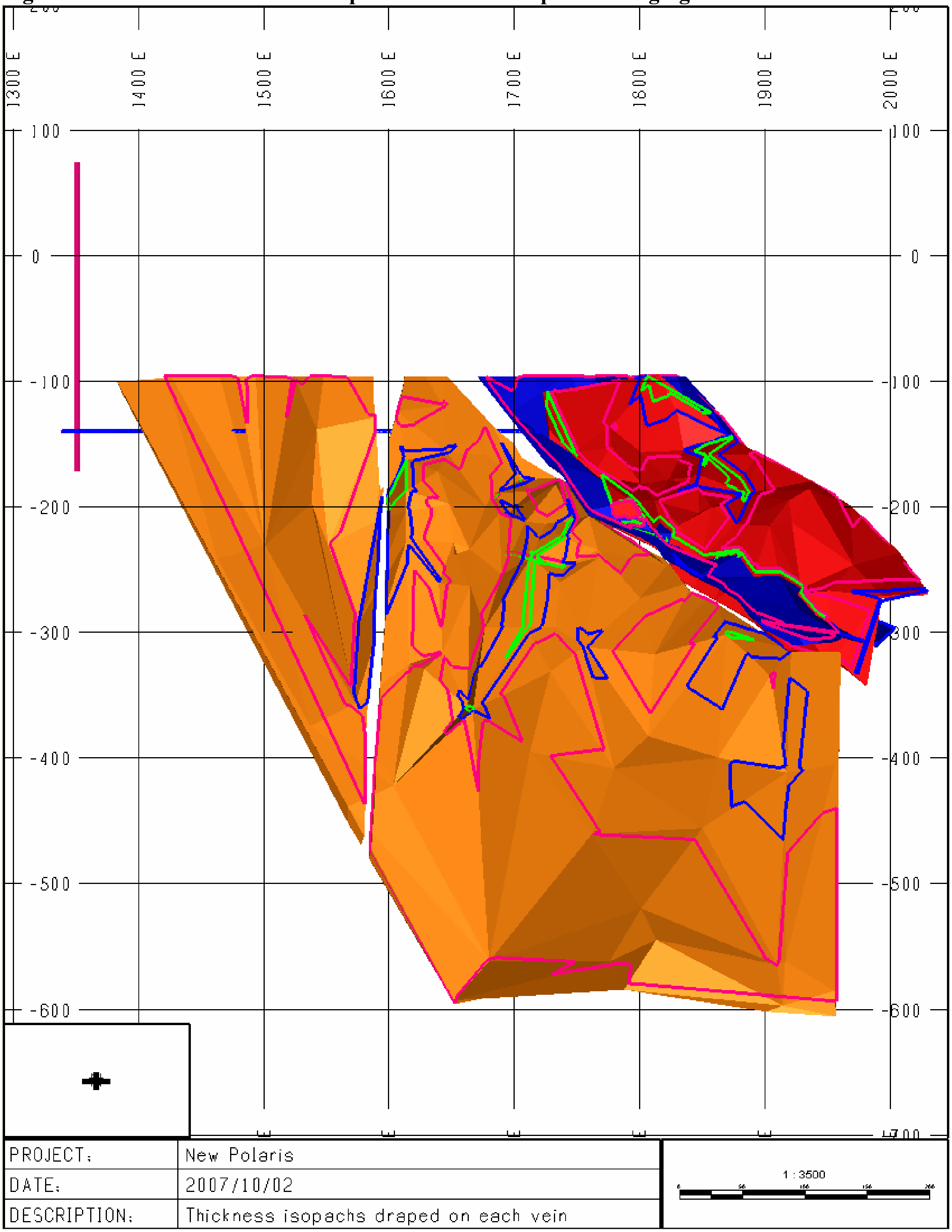
Dilution has been modeled within the mining limits by adding a thickness of surrounding material to the mineral zone footwall and hanging wall. Different thicknesses of dilution are added to different thicknesses of the zones to reflect more selective mining techniques in thinner areas. The zone thickness sub-divisions and the footwall/hangingwall addition is given in the table below.

Zone Sub-division thickness	Dilution added on FW & HW
0 m to 2.5m	0.25m
2.5m to 3.5m	0.35m
3.5m to 5.5m	0.5m

The diluted areas of the >9gpt mining zone account for ~13% of the total volume to be mined. With dilution considered, the total tonnage contained within the >9gpt mining zone is 2,032 kTonnes, averaging 12.48gpt.

These isopachs have also been used to estimate areas where longhole and shrinkage stoping would be used. For areas greater than 5.5m in thickness, longhole stoping is used; and in areas less than 5.5m in thickness Alimak assisted shrinkage stoping methods are used. In the figure below, the areas contained within the green lines are below 2.5m in vein thickness, and the material is assumed to be unrecoverable by conventional mining methods; the areas contained between the blue and green lines are between 2.5m and 3.5m in vein thickness, and the material is assumed to be recovered by traditional shrinkage stoping techniques; the areas contained within the blue and pink lines are between 3.5m and 5.5m in vein thickness, and the material is assumed to be recovered by Alimak assisted shrinkage methods; the areas outside the pink line are greater than 5.5m in vein thickness and are assumed to be recovered by longhole stoping methods.

Figure 19-11 Modeled vein with isopachs thickness draped on hanging wall



19.9.4 Mining Recovery

A mining recovery rate of 90% is assumed. This is based on 8% left behind as pillars, and an additional 2% from mining activity.

19.9.5 Mineable Resource

The following table shows the level by level diluted tonnages and grades that are contained in the mining limits set by the 9gpt cutoff grade and within the planned mine development levels. (85 ktonnes are 'lost' outside the development levels.)

Table 19-9 Mining Tonnage and Grade by Extraction Level

Level	Upper Elevation	Lower Elevation	Mineralized and Diluted		**Recovered**	
	m	m	kTonnes	Grade (gpt)	**kTonnes**	**Grade (gpt)**
1	-140	-190	293.8	12.88	**264.4**	**12.88**
2	-190	-240	356.8	13.37	**321.1**	**13.37**
3	-240	-290	299.5	12.83	**269.5**	**12.83**
4	-290	-340	190.0	13.22	**171.0**	**13.22**
5	-340	-390	148.8	14.06	**133.9**	**14.06**
6	-390	-440	216.4	12.55	**194.8**	**12.55**
7	-440	-490	182.8	10.60	**164.5**	**10.60**
8	-490	-540	137.8	10.27	**124.0**	**10.27**
9	-540	-590	121.3	10.58	**109.1**	**10.58**
Total			**1,947.0**	**12.51**	**1,752.3**	**12.51**

The mineralized and diluted tonnes and grade are a product of the material within the >9gpt mining limits. The recovered tonnes are a sum of the mineralized and diluted tonnes, and factored down based on the mining recovery.

20.0 OTHER RELEVANT DATA AND INFORMATION

No relevant data or information has knowingly been omitted by the author.

21.0 INTERPRETATION AND CONCLUSIONS

The New Polaris deposit was mined by underground methods from 1938 to 1942, and from 1946 to early 1951, producing a total of 740,000 tonnes of ore at an average grade of 10.3g/t gold.

The deposit is composed of three sets of veins (quartz-carbonate stringers in altered rock), the "A-B" veins are northwest striking and southwest dipping, the "Y" veins are north striking and dipping steeply east and finally the "C" veins are east-west striking and dipping to the south to southeast at 65º to 90º. The "C" veins appear to hook around to the north and south into the other two sets of veins so that their junctions form an arc. The gold is refractory and occurs dominantly in finely disseminated arsenopyrite grains that mineralize the altered wallrock and stockwork veins. The next most abundant mineral is pyrite, followed by minor stibnite and a trace of sphalerite. The zones of mineralization range from 15 to 250 metres in length and 0.3 to 14 metres in width.

Exploration, from 1988 to 1997, and in-fill drilling from 2003 to the end of 2006, has confirmed the continuity of the "C" vein system and improved the confidence that the mineralized vein systems are continuous between the earlier widely spaced holes. The "C" vein system appears to be the larger and most continuous of the vein systems found to date on the property.

Preliminary metallurgical test work confirmed the refractory nature of the mineralization and that up to 96% of the gold can be recovered into concentrate. Bio-oxidation and pressure oxidation also yielded positive results with 96 % and 93 % recoveries respectively. Further sampling and testing was completed in 2006.

An updated resource estimate was prepared by Giroux Consultants Ltd. using ordinary kriging of 192 recent drillholes and 1,432 gold assay intervals constrained within four main vein segments as modeled in 3D by Canarc geologists. The total New Polaris database consists of 1,056 diamond drillholes with a total of 31,514 sample intervals.

The geologic continuity of the C vein has been well established through historic mining and diamond drilling. Grade continuity was quantified using Geostatistical semivariograms, which measures distances (ranges) and directions of maximum continuity. The four principle veins in the semivariogram model produced ranges between 50 and 90 metres, both along strike and down plunge.

For this study, the classification for each resource block was a function of the semivariogram range. In general, blocks estimated using ¼ of the semivariogram range were classed as measured, blocks estimated using ½ of the semivariogram range were classed as indicated, and all other blocks estimated were classed as inferred.

The following tables list the undiluted resource estimate, including the "C" vein west (CWM) from the –90m elevation and below, and the "C" vein east (CLOE and CHIE) from the –135m elevation and below.

Measured, undiluted resource

Cutoff grade, g/t Au	Tonnes > Cutoff (tonnes)	Grade > Cutoff Au (g/t)	Contained Metal (oz)
2.00	390,000	9.48	119,000
4.00	330,000	10.62	113,000
6.00	271,000	11.89	104,000
8.00	203,000	13.54	88,000

Indicated, undiluted resource

Cutoff grade, g/t Au	Tonnes > Cutoff (tonnes)	Grade > Cutoff Au (g/t)	Contained Metal (oz)
2.00	1,280,000	10.97	451,000
4.00	1,180,000	11.65	442,000
6.00	1,017,000	12.71	416,000
8.00	806,000	14.22	368,000

Measured + Indicated, undiluted resource

Cutoff grade, g/t Au	Tonnes > Cutoff (tonnes)	Grade > Cutoff Au (g/t)	Contained Metal (oz)
2.00	1,670,000	10.62	570,000
4.00	1,510,000	11.42	555,000
6.00	1,288,000	12.54	519,000
8.00	1,009,000	14.08	457,000

Inferred, undiluted resource

Cutoff grade, g/t Au	Tonnes > Cutoff (tonnes)	Grade > Cutoff Au (g/t)	Contained Metal (oz)
2.00	2,060,000	10.52	697,000
4.00	1,925,000	11.03	683,000
6.00	1,628,000	12.15	636,000
8.00	1,340,000	13.27	571,000

This latest estimate shows an important gold resource at the New Polaris deposit, which should be further tested with infill drilling and underground exploration.

The Preliminary Assessment demonstrates that, at a US$650 per oz gold price, the project generates a pre-tax undiscounted net present value (NPV) of CA$60.4 million and a pre-tax tax internal rate of return (IRR) of 14.9%, or an after tax NPV of $40.9 million and an after IRR of 11.1%.

The Preliminary Assessment indicates that the New Polaris base case has potential for positive results and therefore further work is recommended to optimize the project and complete a feasibility study.

The Preliminary Assessment is based on resources, not reserves, and a portion of the modeled resources to be mined are in the inferred resource category. Resources are considered too speculative geologically to have economic considerations applied to them so the project does not yet have proven economic viability.

The New Polaris current resources were previously disclosed in a news release dated February 1, 2007 and in a NI 43-101 technical report filed on SEDAR on March 15, 2007.

Capital costs include CA$19 million to complete a feasibility study, as well as the capital needed to purchase equipment, further develop the mine and construct the plant and site infrastructure. Once a feasibility study is completed, the $CA19 million will drop out of the capital cost estimate.

Cash costs include site related costs prior to the shipping and sale of concentrates. Offsite costs for concentrate transportation and processing are treated as deductions against sales.
The Net Present Values are life of mine net cash flows shown at various discount rates. The Internal Rates of Return assume 100% equity financing.

The New Polaris project is sensitive to the price of gold. At US$600 per oz gold, the pre-tax NPV is CA$26 million with an IRR of 7%, or an after tax NPV of CA$18 million and an IRR of 5% and at US$750 gold, the pre-tax NPV jumps to CA$130 million and the IRR increases to 29%, or the after tax NPV increases to CA$87 million and the IRR increases to 22%.

The project is also sensitive to the US$/CA$ exchange rates and gold recoveries. Each 1% change in the US$/CA$ exchange rate indicates a change of approximately $0.3 million CA$ in after tax undiscounted NPV. Changes in recoveries produce similar results.

Opportunities exist to improve the base case model such as:

- Increasing resources and therefore mine life;
- Increasing gold recoveries and concentrate grades;
- Increasing production to enhance economies of scale;
- Reducing transportation costs; and
- Reducing offsite processing costs.

The main cost risks include:

- Rising engineering and construction labour and equipment costs due to limited availability;
- Escalating capital costs if there are project delays;
- Rising operating costs due to inflation and commodity shortages; and
- Fluctuations in US$/CA$ exchange rates.

22.0 RECOMMENDATIONS

Exploration completed on the property to the end of 2006, as recommended by Walton and McClintock, has demonstrated the continuity and grade of the "C" vein system.

Further work should include a preliminary assessment of the mineable resource to determine the nature of the "C" vein, a better estimate of the SG, and the collection of representative samples for metallurgical testing. As well, more infill drilling should be completed from underground stations.

It is recommended that humidity cell testing of core rock continues to operate to twenty-six weeks to provide a longer term assessment of metal loadings and provide long term data for predicting time to sulphide and carbonate depletion.

Additional flotation test work is underway to try and improve gold recoveries and concentrate grades. Autoclave and bio-leach test results will also be completed within the next month or two, which should allow management to initiate discussions with potential buyers of the New Polaris concentrates.

The base case model includes conventional barging of concentrates off-site during the summer season. However Canarc's neighbor, Redcorp Ventures, has applied for permits to operate year-round air cushion barges, ("ACB's") towed by amphibious vehicles ("Amphitracs"). Such technology may also be advantageous for Canarc and will be evaluated as part of a feasibility work program.

Future work should include driving an access ramp from surface to access the C vein ore and drifting and raising in ore on various levels to increase knowledge of the zone and upgrade the resources to the proven category. During this phase representative samples of ore should be collected for further metallurgical testing.

Canarc is contemplating proceeding with the above test work at an estimated cost of CA$18.7 million. The results of this work will be used as the basis for a future Feasibility Study. It is recommended that this work be expedited to advance the New Polaris project to a Pre-Feasibility or Feasibility Study Reserve and to pursue project financing.

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24.0 DATE AND SIGNATURE PAGE

I, James H Gray P.Eng., do hereby certify that:

1. I am a Principal of Moose Mountain Technical Services My office address is 1584 Ever green Hill SW Calgary Alberta T2Y 3A9.
2. I have read the definition of a "qualified person" and fulfill the requirements of a Qualified Person as specified in National Instrument 43-101 of the Canadian Securities Administrators.

   I have received a Bachelor of Applied Science from the University of British Columbia, Vancouver, 1975 in Mining Engineering

   I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia (11919).

   I am a member in good standing of the Association of Professional Engineers, Geologists and Geophysicists of Alberta (M47177).
3. I am a member of the Canadian Institute of Mining and Metallurgy.
4. I have been practicing as a Professional Engineer for over 25 years with relevant experience for the Technical Report including:

   1975 to 1978 Underground stope mining, Mine Supervision, and Mine Engineering positions in operations in Canada and Australia

   1978 to 1989, mine site engineering, operations and management positions, costing, evaluating new mineral projects and development properties.

   1989 to present, mine engineering consultant work on assessment and feasibility studies of numerous coal, base metal, industrial mineral, and precious metal deposits in Canada, United States, Mexico, Chile, Argentina, Peru, Turkey, Iran, and Australia.
5. I am responsible for the compilation of the work of the other Qualified Persons, and the other experts listed to produce this Technical Report titled "New Polaris Project - Preliminary Assessment" dated Oct 4th 2007.
6. I have not visited the property
7. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
8. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 4th day of October 2007.

___(Signed)________________________

J.H. Gray PEng

Moose Mountain Technical Services

I, Robert J. Morris, M.Sc., P.Geo., do hereby certify that:

1. I am a Principal of Moose Mountain Technical Services, 6243 Kubinec Road, Fernie BC V0B 1M1.
2. I graduated with a B.Sc. from the University of British Columbia in 1973.
3. I graduated with a M.Sc. from Queen's University in 1978.
4. I am a member of the Association of Professional Engineers and Geoscientists of B.C. (#18301).
5. I have worked as a geologist for a total of thirty-three years since my graduation from university.
6. My past experience with gold exploration and mining includes work in the Bralorne area, China, Argentina, and Northern Saskatchewan.
7. I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" as defined in National Policy 43-101.5.
8. I am responsible for the geology and resource review and verification and preparation of the technical reports titled "Resource Potential, New Polaris Project", dated 5 March 2007 and "New Polaris Project, Preliminary Assessment" dated October 4th, 2007.
9. I completed a site visit of the New Polaris Property during the period 22-24 August 2006. My prior involvement with the project includes a site visit 21 August 1988, and work on the geology, resource estimate, and exploration program during 1988 and 1989, while with Beacon Hill Consultants.
10. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
11. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
12. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Date this 4th day of October 2007,

____(Signed)______________
Signature of Qualified Person

Robert J. Morris, M.Sc., P.Geo.
Print Name of Qualified Person

G.H. Giroux

I, G.H. Giroux, of 982 Broadview Drive, North Vancouver, British Columbia, do hereby certify that:

1. I am a consulting geological engineer with an office at #1215 - 675 West Hastings Street, Vancouver, British Columbia.
2. I am a graduate of the University of British Columbia in 1970 with a B.A.Sc. and in 1984 with a M.A.Sc. both in Geological Engineering.
3. I have practiced my profession continuously since 1970. I have completed resource estimation studies for over 30 years on a wide variety of base and precious metal deposits including several narrow vein mesothermal deposits.
4. I am a member in good standing of the Association of Professional Engineers of the Province of British Columbia.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Policy 43-101.5
6. The report titled Resource Potential New Polaris Project and dated March 5, 2007 ("Technical Report") is based on a study of the available data and literature for the New Polaris (formerly the Polaris-Taku) Deposit. I am responsible for the resource estimation section of this report. The work was completed in Vancouver during September 2006 to February 2007. I have not visited the property.
7. This report titled "New Polaris Project – Preliminary Assessment" dated October 4th, 2007 is based on the same resource model referenced in 6. above.
8. I have previously completed resource estimations on this property in 1991 and 1995.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
10. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 4th day of October, 2007

GIROUX CONSULTANTS LTD.
Per:

G. H. Giroux, P.Eng., MASc.

(Signed)

25.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

This Technical Reports is based on Geology information and Resource model discussed early in this report as well as other studies commissioned by Canarc on the property. Information from the following reports is available from Canarc.

Metallurgical Report Canarc Resources Corp. New Polaris Project Atlin, BC - May 20, 2007
Jasmin Yee, P. Eng.

New Polaris Gold Mine Ltd. Surface Infrastructure Cost Estimate – April, 2007
Ken Chamberlin

Ongoing ARD/ML Kinetic Testing by URS Canada Inc. These test are currently in progress and tests are not complete but early indications are available.

Internal Financial Models – September, 2007
P. A. Stokes P. Eng.

The results of these studies are summarized below.

25.1 Operations

Operations will include year round underground mining activities and onsite processing to produce bulk concentrate, and seasonal barge shipping of concentrates and supplies. Onsite support for the operations and management of a local camp with fly-in fly out service to the landing strip onsite are included in the cost estimates. Details of the mining, processing, and onsite facilities are included in the Canarc documents referenced above with a summary description of the operations below.

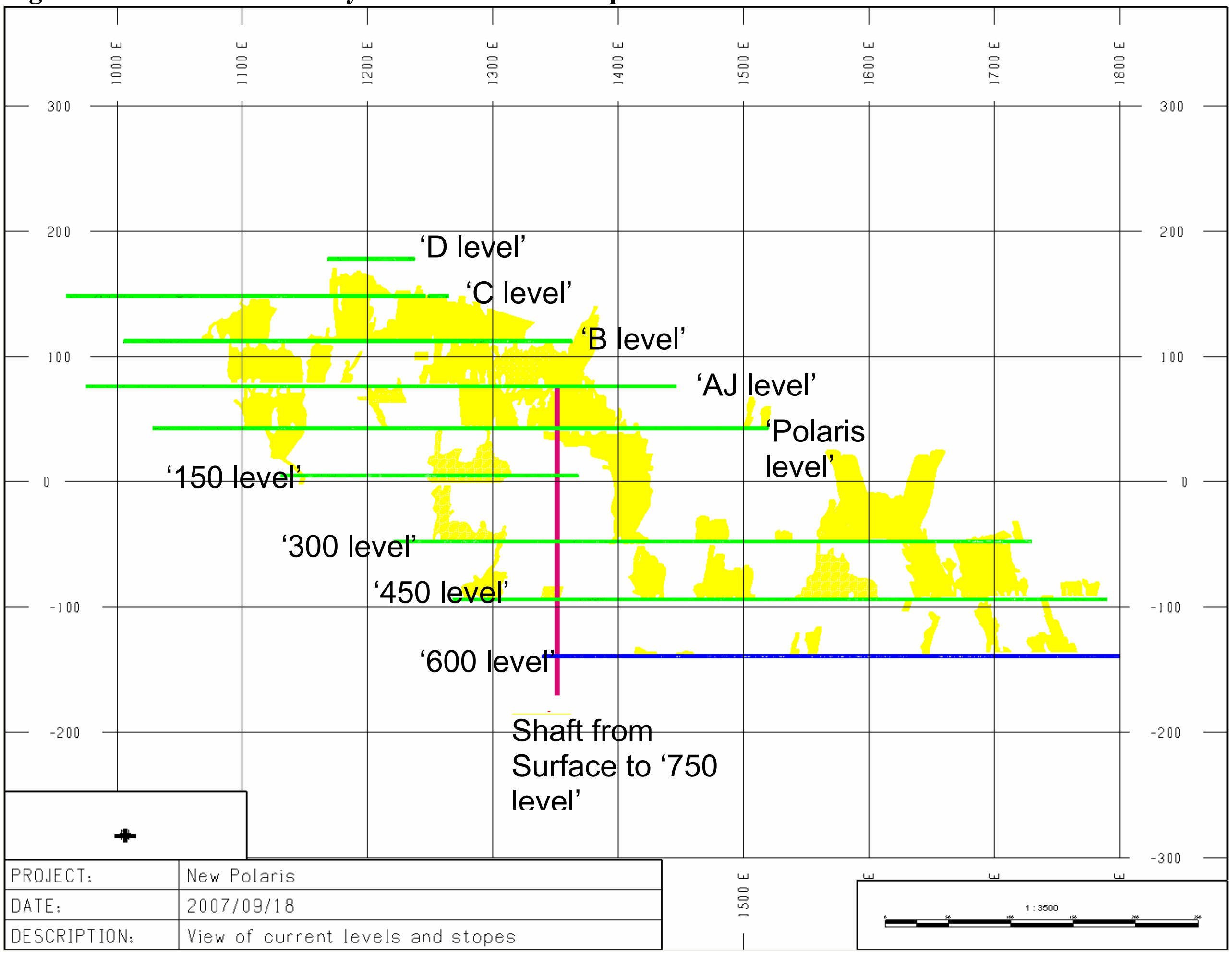
25.1.1 Mining

The Mine design is based on the Mineable resource outline as described in Section 19.9. Details are available in the report "New Polaris Mine Plan & Scoping Study Summary." An internal report at Canarc.

25.1.1.1 Existing Underground Infrastructure

The figure below shows the currently mined levels and stopes looking north. The shaft from the surface to the '750 level' is also shown. All previously mined stopes were mined using a shrinkage stoping methods. For levels 'D' to 'AJ', ore was transported to surface along the levels. For levels below 'Polaris' ore was transported up the shaft and along 'Polaris' to surface.

Figure 25-1 View of currently mined levels and stopes



25.1.1.2 Layout of New Development

25.1.1.2.1 Ramp from Surface

A ramp from surface is driven at -15% for ~165m vertical and a total length of 1,225m. It ties in with the expansion of the '600 level' described below. The ramp is driven with an equivalent height of 3.7m and a width of 4m.

25.1.1.2.2 Main Ramp

Starting at the 600 level, from an intersection at 940m N and 1700m E, the main ramp runs approximately 3770m in length and ends at a point 595m N and 1870m E. Winding down to the lower portion of the ore body at a 15% grade, the 3.7m high x 4m wide ramp provides access to the ore zone at a variety of locations at 50m vertical level intervals.

A new drift will be driven to connect the original mine workings with the top of the new ramp.

Figure 25-2 Surface and Main Ramp, looking south

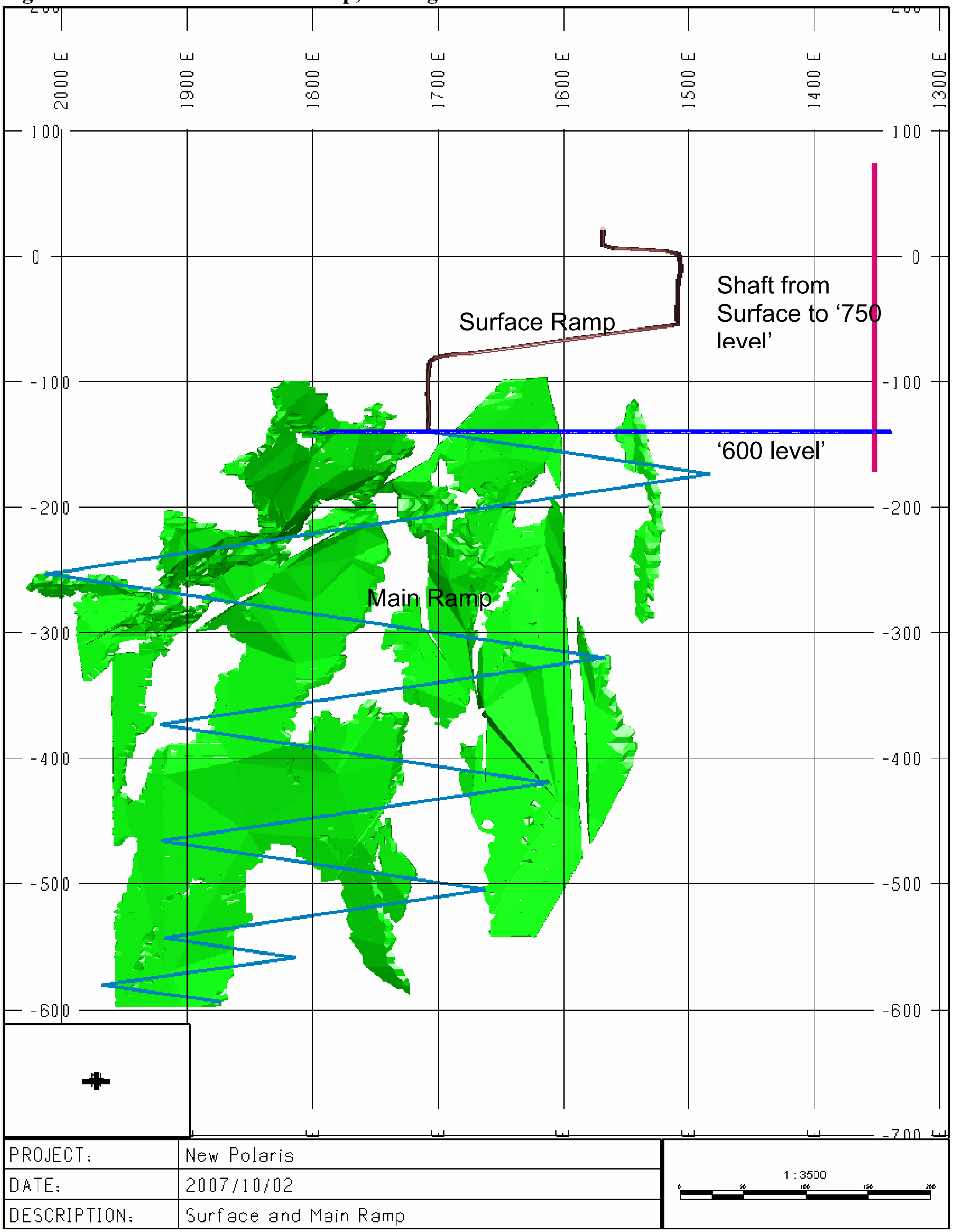


Figure 25-3 Main Ramp, Plan View

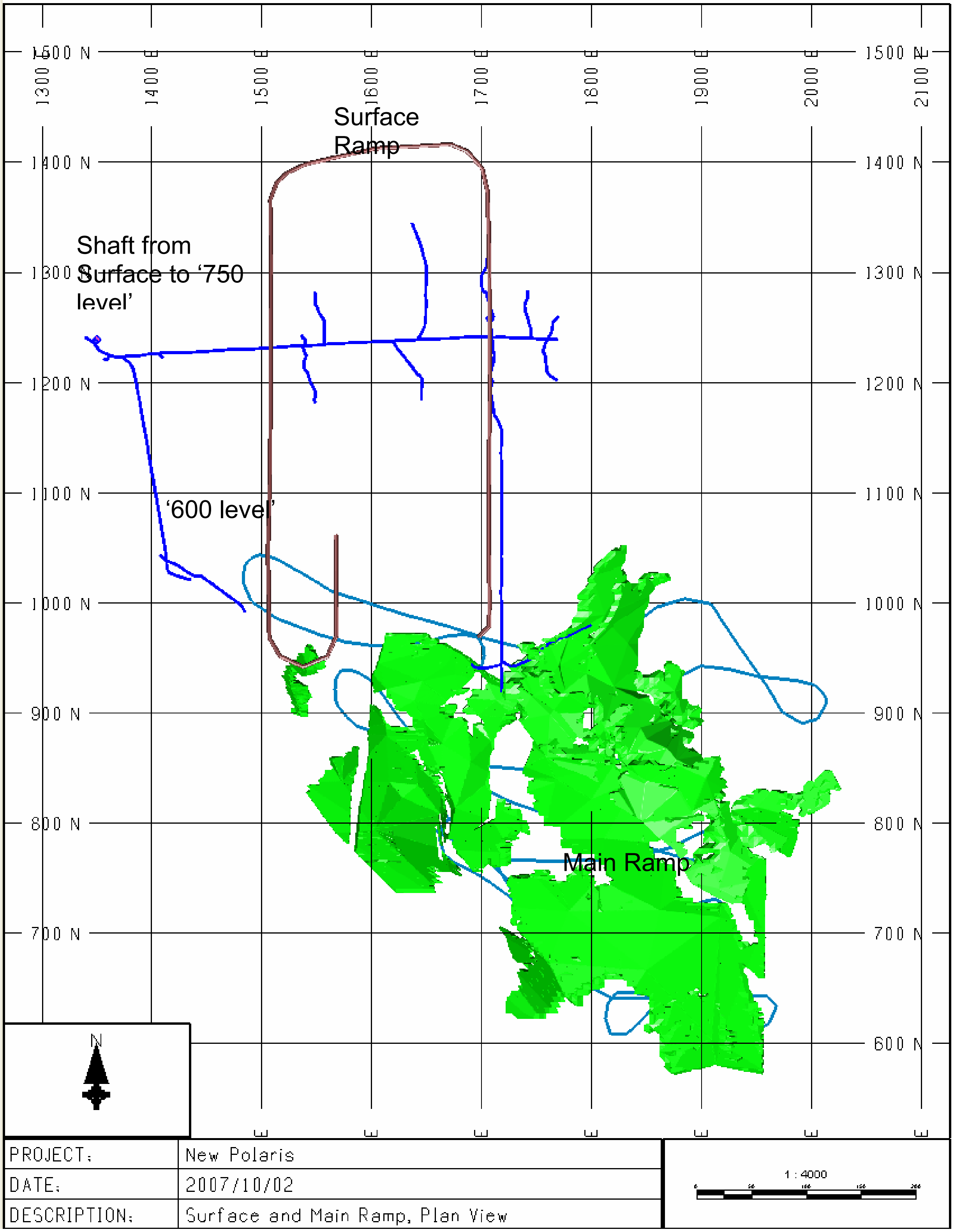
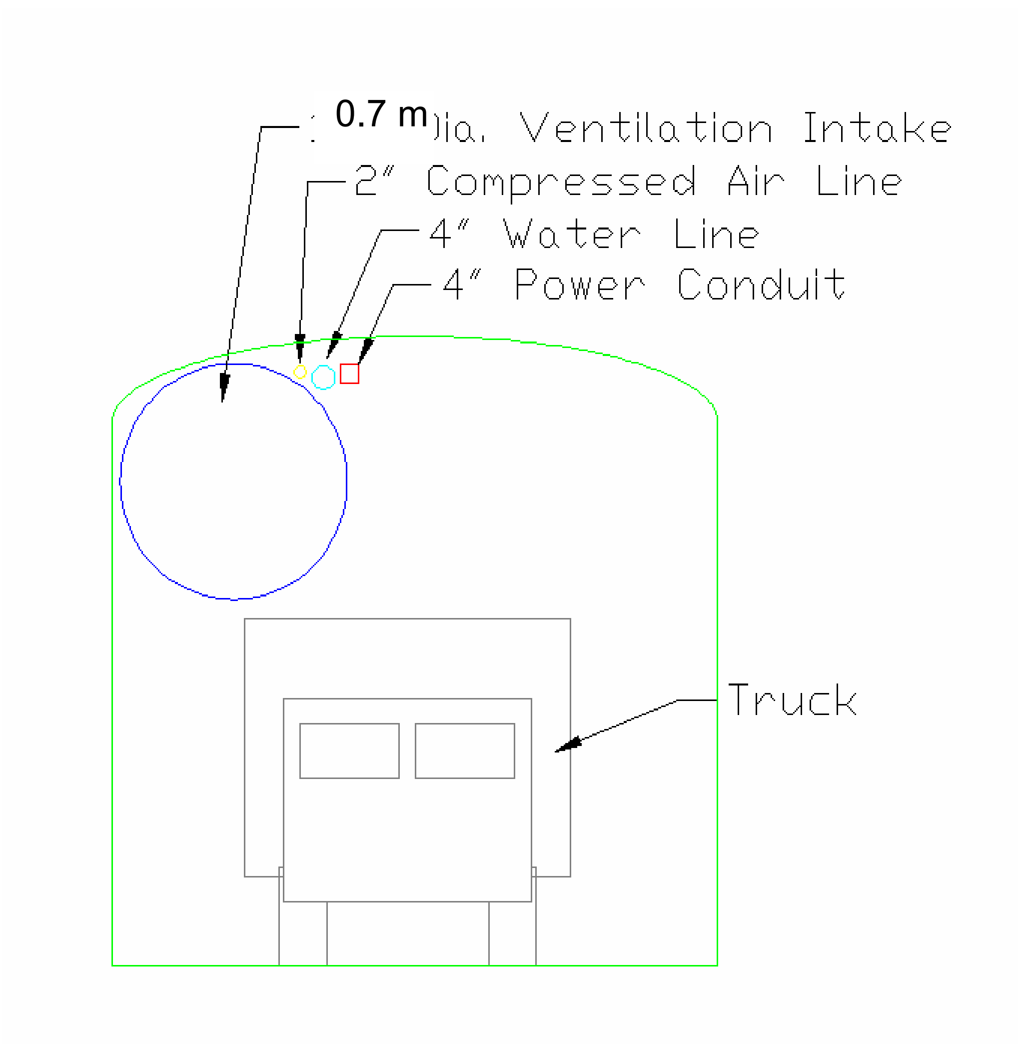


Figure 25-4 Cross Section of Main Ramp



25.1.1.2.3 Extraction Drifts

Extraction drifts will be driven at 25m vertical intervals in longhole stoping areas and 50m vertical intervals in Alimak shrinkage stoping areas along the entire main ramp. They are designed to give access to the footwall of the ore body, from which stopes can be started and ore can be hauled. The extraction drift is designed 10m from the footwall of the mineralized zone. Drawpoints will be driven at 10m horizontal intervals from the extraction drift to the ore body.
Using the 9g/t mining limit, the number of stopes to develop for each extraction drift is estimated. The extraction drifts are driven out to the edges of this mining limit and can be extended if the economics dictate.

Figure 25-5 Extraction Drifts

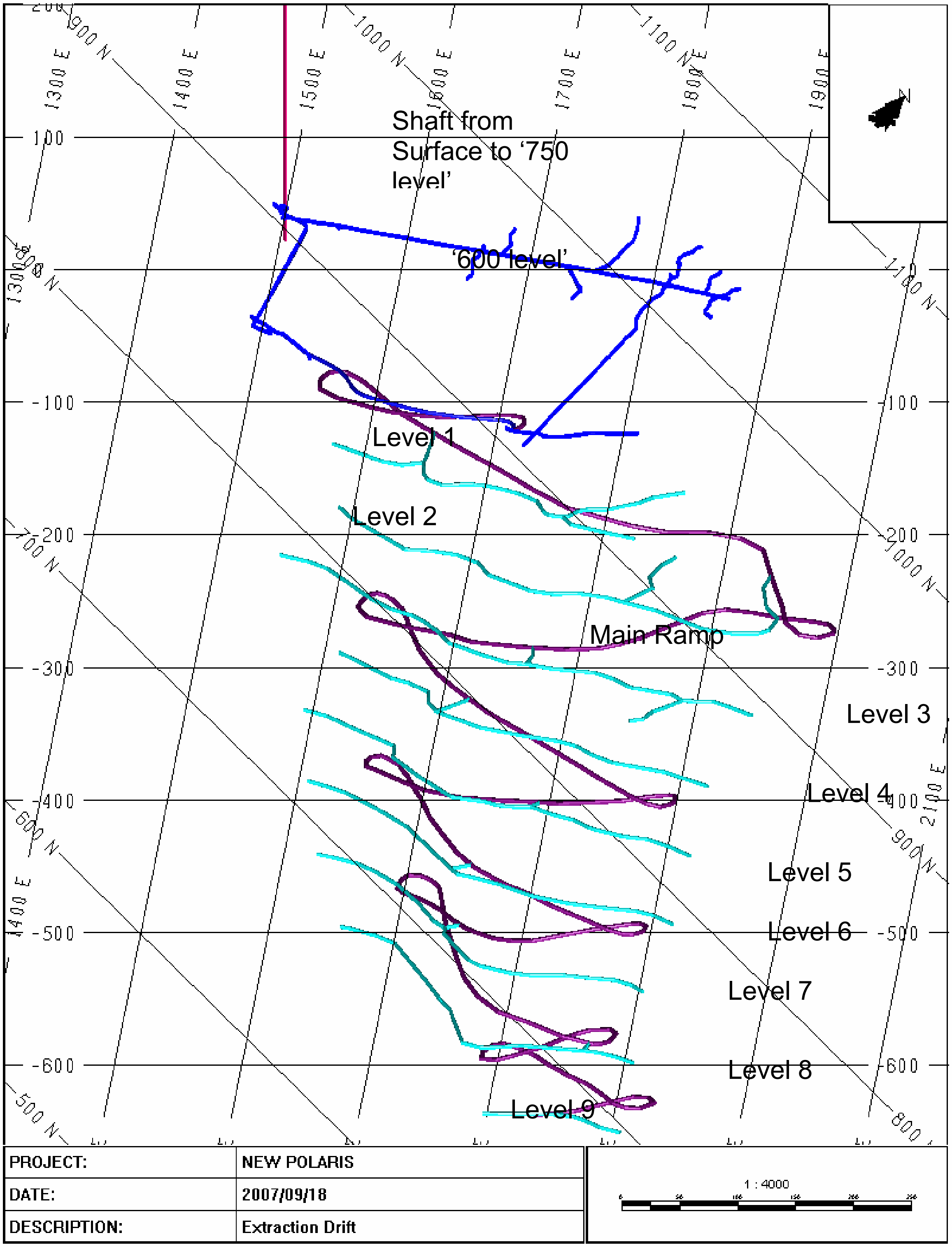


Table 25-1 Extraction drift length and estimated number of stopes

Level	Elevation	Length	Estimated number of Stopes
	m	m	
1	-190	625	7
2	-240	665	9
3	-290	620	8
4	-340	455	7
5	-390	465	8
6	-440	450	8
7	-490	430	7
8	-540	425	6
9	-590	135	2

25.1.1.3 Development Schedule

The following development items are scheduled:

- Surface ramp to the existing 600 level.
- Main ramp from the existing 600 level down to the lower portion of the mineralized ore body.
- Re-muck bays spaced as needed along the entire surface and main ramp.
- Ventilation raise that ties into the main ramp progressing from the current 600 level down to the bottom of the ramp.
- On each extraction level:
    - The extraction drift itself.
    - Drawpoints accessing the undercut of the stopes from the extraction drifts.
    - Crosscuts into the raises on either end, and where required to muck the ore.
    - Re-muck bays spaced along the extraction drift.
    - Service raises on each end of each stope that is developed, and a third raise in all stopes below level 2 (this material reports to ore production, whereas all other development items report to waste).

The production schedule is used to determine when each item of development is required. In the year before the ore production from a level is required, all access to the stopes on that level is scheduled to be complete, as well as development of the stopes themselves.

The first five items listed above are scheduled together. In Year -2, the first 10% of this development is completed. In Year -1, another 50% of this development is completed and provides access to all of the exploration levels. The remaining 40% is completed in 10% increments over the next 4 years.

The following table shows the years in which the development for each extraction level is scheduled to be completed.

Table 25-2 Development Schedule by Level

Extraction Level	Year Development is Completed	Years level is mined
1	-1	1 to 2
2	1	2 to 3
3	2	3 to 4
4	3	4 to 5
5	4	5 to 6
6	5	6 to 7
7	6	6 to 7
8	7	7 to 8
9	8	8

The following table shows a year by year breakdown of the development meters, waste tonnage and ore tonnage related to the development work required.

Table 25-3 Development Schedule, Lengths and Tonnages

	Total Ramps and Drifts			Total Drawpoints/Crosscuts			Total Re-Muck Bays			Total Development Raise			Total Stope Raise			
	Meters	Waste kT	Ore kT	Meters	Waste kT	Ore kT	Meters	Waste kT	Ore kT	Meters	Waste kT	Ore kT	Meters	Waste kT	Ore kT	Total Meters
Year -3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Year -2	1,126	47	0	0	0	0	15	2	0	40	1	0	0	0	0	1,181
Year -1	3,752	156	0	1,120	47	0	122	15	0	200	3	0	0	0	0	5,194
Year 1	3,045	127	0	0	0	0	15	2	0	40	1	0	2,400	0	27	5,500
Year 2	2,063	86	0	560	23	0	38	5	0	40	1	0	1,200	0	13	3,900
Year 3	1,676	70	0	490	20	0	33	4	0	40	1	0	1,050	0	12	2,289
Year 4	1,505	63	0	560	23	0	33	4	0	40	1	0	1,200	0	13	3,338
Year 5	755	31	0	560	23	0	14	2	0	0	0	0	1,200	0	13	2,529
Year 6	760	32	0	490	20	0	14	2	0	0	0	0	1,050	0	12	2,313
Year 7	771	32	0	420	17	0	14	2	0	0	0	0	900	0	10	2,105
Year 8	154	6	0	140	6	0	5	1	0	0	0	0	300	0	3	599
Year 9	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Year 10	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	15,607	649	0	4.340	180	0	302	38	0	400	7	0	9,300	0	105	105

25.1.1.4 Description of Mining Operation

The mining operation is split into 3 major areas: Development mining, Production mining, and General Mine Expense (GME).

The production and development mining is separated into subcategories for drilling, blasting, loading, hauling, ground support, operational support, utilities, services and unallocated labour. Each section accounts for all equipment consumables and parts, manpower required (both operating and maintenance) and all material costs (blasting, ground support, services).

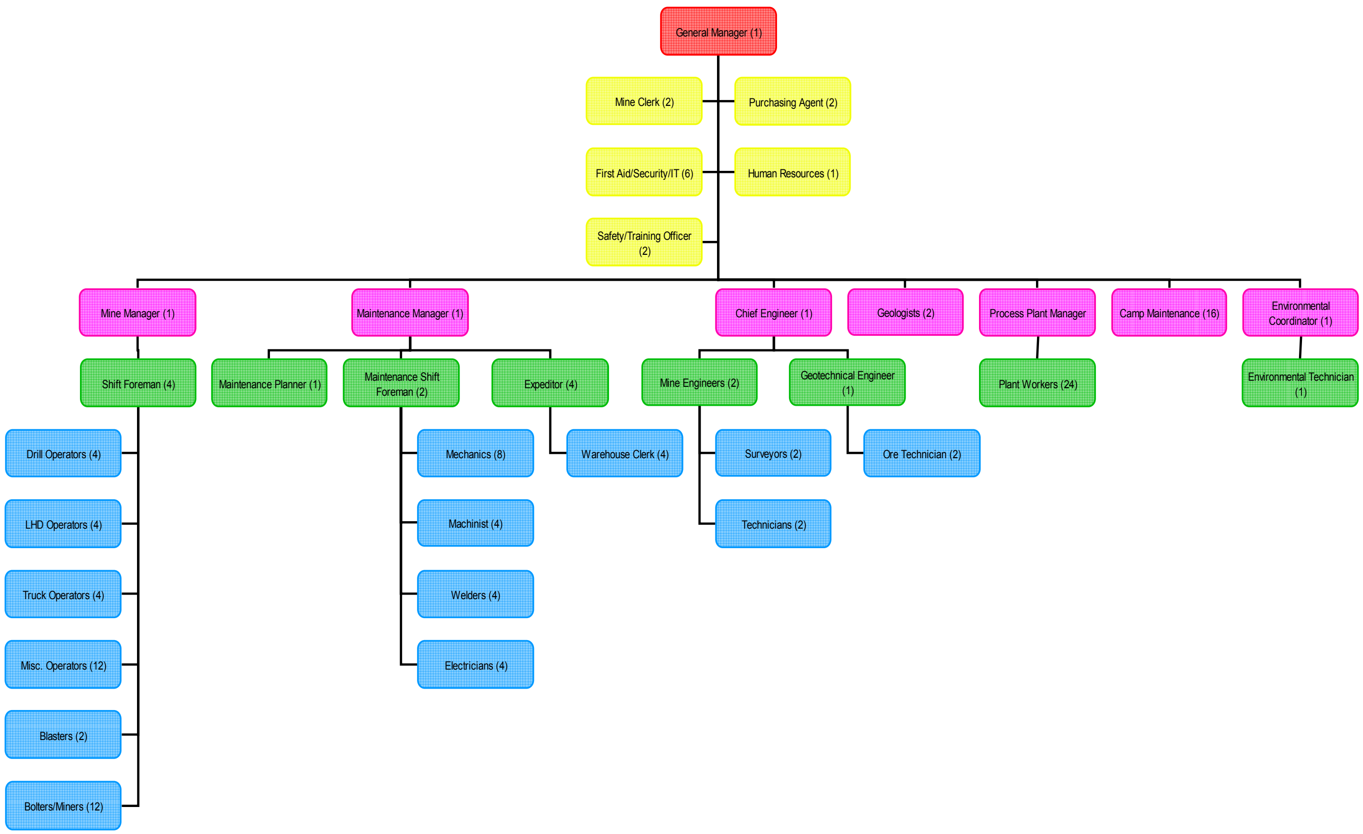
GME includes the supervision and training for the direct mining activities as well as technical support from Mine Engineering and Geology functions. More detailed descriptions of the mine organization and the unit mining activities follows.

In this study the development and production operations are completed using an owner's fleet and owner's workforce.

25.1.1.4.1 Mining Operations Organization

Details of Mine Operations are given below showing the breakdown of the Direct Mining, Mine Maintenance, and GME functions.

Figure 25-6 Mine Operations Organizational Chart



25.1.1.5 Mining Costs

Mine capital and operating costs are derived from a combination of supplier quotes and historical data collected by MMTS. This includes the labour, maintenance parts, fuel, tire and other consumables costs.

The unit operating costs are used as a constant basis over the schedule periods and estimates are input for Sustaining and Replacement Capital.

25.1.1.5.1 Mining Capital Costs

The mining capital purchases are listed in the table below: Overall Cost, Fleet Manpower, and Supplies Summary. The cost for replacement of each piece of equipment is calculated from the expected life of the piece of equipment, the cost of the item, and the number of pieces that require replacement.

Table 25-4 Summary of Mining Capital Purchases

Mining	
Size Description	Capital Cost ($CDN)
Jackleg Stoper Drill, 16kg	$5,000
Development Drill, Jumbo	$500,000
Raise System	$550,000
LHD, 3m^3 bucket	$450,000
30 tonne underground hauler	$700,000
Crane Truck/Pallet Handler	$270,000
Scissorlift,1818 kg lift capacity, 3.5m lift	$230,000
Grader,108 kW, 3m blade width	$315,000
Pickup, Passenger Carrier	$35,000
Flatbed Truck	$30,000
Mine Rescue Gear	$200,000
Mine Rescue Station	$100,000
Air Hoists w/slusher blades and ropes	$20,000
Utilities	
Size Description	Capital Cost ($CDN)
Primary Vent Fans, 80kW	$15,000
Aux. Vent Fans (Stopes), 20kW	$7,000
Aux. Vent Fans (Development), 25kW	$7,000
Concrete Bulkheads with Air Control Doors	$50,000
Mine Office and Dry	$2,000,000
Dewatering Pumps, 25 kW	$30,000
Aux. Sump Pumps, 100 kW	$100,000
Underground Workshop	$500,000
Electrical Substations, 5 MW	$500,000
Oil Fired Air Heater, 30L/hour	$50,000
Breaker	$250,000
Shop, Warehouse, Fuel Tanks, etc.	$2,000,000
Air Compressor, 375 kW, with receiver	$76,380

25.1.1.5.2 Estimates of Equipment Hours

Scheduled Hours and Operating Hours

For each year of operation, 365 days are scheduled with two 12-hour shifts per day. This amounts to 8,760 hours of scheduled mining each year.

For all mobile equipment listed above, the operating hours are de-rated by the following factors:

- For each shift, 1.5 hours is spent on lunch, shift changes, safety meetings, etc. An 87.5% shift utilization factor is applied.

- An availability factor of 82% is applied, which is a combination of 85% mechanical availability, and 97% use of availability.

For the mine rescue gear, mine rescue station and all the listed utilities the utilization and availability factors are set at 100%.

For the drilling, loading, and trucking equipment, a 90% operator efficiency factor is applied.
An operations efficiency factor is also applied to the items listed above. Operations efficiency accounts for all time that an item is available and ready for operation, but is not used because it is not required; or the operator is performing auxiliary tasks; or any other operational delay that may occur underground. For all drilling, loading and trucking units, an 87.5% operations efficiency factor is applied.

With the utilization, availability, operator efficiency and job efficiency considered, the drilling, loading and trucking units have a maximum amount of effectively operated hours of 4,977/year.
For the support and utility items, the following operational efficiency factors are applied. The resultant effective operating hours each year are also shown.

Table 25-5 Utilization of Support and Utility Items

Unit	Job Efficiency	Maximum Effective Operating Hours
Toolcarrier (Getman Pallets/Crane)	20%	1,264
Scissor Lift	20%	1,264
Motorgrader	70%	4,424
Crew Trucks	50%	3,160
Flatbed Truck	75%	4,740
Mine Rescue Gear	87.5%	7,665
Mine Rescue Station	1%	88
Primary Vent Fans	100%	8,760
Aux. Vent Fans (Stopes)	100%	8,760
Aux. Vent Fans (Development)	100%	8,760
Air Compressor	100%	8,760
Mine Office and Dry	100%	8,760
Dewatering Pumps	100%	8,760
Aux. Sump Pumps	100%	8,760
Underground Workshop	100%	8,760
Electrical Substations	100%	8,760
Oil Fired Air Heater	100%	8,760
Concrete Bulkheads for Air Control	100%	8,760
Shop, Warehouse, Fuel Tanks, etc.	100%	8,760
Air/Electric Hoists for Stopes	60%	3,792

Utilization of Development Equipment

For each Advance Round of the ramp development, the following cycle schedule has been determined:

Table 25-6 Ramp Development Cycle

Jumbo	Drill - hours	2.35
Blaster	Load & blast - hours	4.00
LHD	Muck out - hours	1.05
Bolter	Scale, bolt - hours	3.00
Utility	Services - hours	0.45
	Total	10.85

The jumbo and LHD hours are based on the calculated productivities from above. The ground support and service installation is from productivities described in the sections below. The blasting hours are the remainder.

When the main ramp and extraction drifts are being developed, the utilization of the jumbo drill the LHD and the trucks are limited to the portions of this cycle that they would be applied.

- The drill jumbo utilization =22%, when in ramp and drift development.
- The LHD utilization =10%, when in ramp and drift development.
- The truck utilization =10%, when in ramp and drift development.

Otherwise these items are utilized by the design basis amount of 87.5%.

25.1.1.5.3 Applying Costs to Operating Hours

For each item scheduled an hourly cost is applied. The equipment productivity and the scheduled production are used to calculate the required equipment operating hours for drilling, blasting, loading and trucking. These are multiplied by the hourly consumables consumption rates and unit operating costs to calculate the total equipment operating costs for each time period. The fuel cost is estimated CDN$1.00/L required. The power cost used is CDN$0.15/kW required.

Blasting costs are based on a per kg cost of explosives. The all-in blasting costs, including supply, delivery, loading and initiation are CDN$3.00/kg required. Blasting labour costs are added on top of this.

Labour factors in Man Hours/equipment operating hour are also assigned to each of the items. Labour costs are calculated by multiplying the labour factor by the item's operating hours, and labour costs are allocated to the items where labour has been assigned. The total hours required for each job type on all the items are summated and any additional labour required to complement a crew is assigned to unallocated labour. The mine hourly and salaried labour schedules are listed in the tables below.

Table 25-7 Summary of Mine Hourly Labour Required

SUMMARY	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5
MINE OPERATIONS							
Drill Operator	2	4	4	4	4	4	4
Blasters	2	2	2	2	2	2	2
LHD Operator	2	4	4	4	4	4	4
Truck Driver	4	4	4	4	4	4	4
Misc. Operator (includes surface operators)	8	12	12	12	12	12	12
Jackleg/Bolter/Utilility Miner	1	4	12	12	12	12	12
MINE MAINTENANCE							
Electrician	2	4	4	4	4	4	4
Mechanic	2	4	8	8	8	8	8
Machinist	2	4	4	4	4	4	4
Welder	2	4	4	4	4	4	4
NON-MINING							
Plant Workers	0	0	24	24	24	24	24
Camp Maintenance	2	16	16	16	16	16	16
TOTAL HOURLY	27	62	98	98	98	98	98

Table 25-8 Summary of Salaried Labour Required

SALARIED LABOUR SUMMARY	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5
ADMINISTRATION							
General Manager - Day Only	1	1	1	1	1	1	1
Mine clerks - Day Only	1	2	2	2	2	2	2
First Aid/Security/IT	0	6	6	6	6	6	6
Human Resources	0	1	1	1	1	1	1
Purchasing Agent - Day Only	0	2	2	2	2	2	2
Safety/Training Officer	1	2	2	2	2	2	2
Environmental Coordinator - Day Only	0	1	1	1	1	1	1
Environmental Technician - Day Only	0	1	1	1	1	1	1
MINE OPERATIONS							
Mine Manager - Day Only	0	1	1	1	1	1	1
Shift Foreman -	2	4	4	4	4	4	4
MINE MAINTENANCE							
Maintenance Manager Day Only	0	1	1	1	1	1	1
Mine Maintenance Foreman	1	2	2	2	2	2	2
Maintenance Planner - Day Only	0	1	1	1	1	1	1
Warehouse Clerk	0	4	4	4	4	4	4
Expeditor	0	4	4	4	4	4	4
MINE ENGINEERING							
Chief Engineer - Day Only	0	1	1	1	1	1	1
Mine Engineer - Day Only	1	2	2	2	2	2	2
Technicians - Day Only	0	2	2	2	2	2	2
Surveyor - Day Only	1	2	2	2	2	2	2
TECHNICAL SERVICES							
Mine Geologist - Day Only	0	2	2	2	2	2	2
Geotechnical Engineer - Day Only	0	1	1	1	1	1	1
Ore Grade Technicians	0	2	2	2	2	2	2
TOTAL SALARIED	8	29	29	29	29	29	29

The Mine plan has been developed to a higher level of detail than a Preliminary Assessment because of the variability of the Mineral zone. This level of detail gives assurance that the plan is achievable. It is recommended that future detailed studies attempt to simplify the plan and consolidate the mining methods if possible. Future planning should also attempt to define the mining loss and dilution on a stope by stope basis.

25.1.2 Processing

The Process design by Jasmin Yee is based on the PRA test work discussed above.

(The following is from "Metallurgical Report Canarc Resources Corp. New Polaris Project Atlin, BC" May 20, 2007 Jasmin Yee P. Eng.)

25.1.2.1 Process Design Criteria

The plant design is based on operating at the rate of 600 tonnes per 24 hr day at 92% availability for 12 months of the year. The design criteria have been drawn from various sources and where data is unavailable, professional judgment has been used.

Run of Mine Ore Characteristics	**Design**
Maximum Size	500mm
Specific Gravity	2.8
Moisture Content	3%
Bulk Density	1.6 t/m3
Angle of Repose	35 deg.

Met Test Composite Feed Grade	**Design**	**Source**
Au g/t	9.7	PRA
Ag g/t	1.2	PRA
As %	2.02	PRA
Sb %	0.78	PRA
S(total) %	3.18	PRA
C (total) %	3.75	PRA

Projected Recovery

Au in flotation Concentrate	91.0%	Best Estimate

Projected Flotation Concentrate Grades

Au g/t	90	Calculated
Ag g/t	9	Calculated
As %	18.5	Calculated
Sb %	6.9	Calculated
S(Total) %	28.8	Calculated

Plant Design

Annual Plant Rate	tonnes	219,000
Daily Plant Rate	tonnes	600
Design Operating Rate	t/h	22.6
Plant Availability		92%

Operating Schedule	365 days per year 2 shifts per day 12 hrs per shift

25.1.2.2 Plant Equipment

The general plant flow sheet consists of:

- Underground ore and any surface stockpiled ore will be delivered to the coarse ore bin.
- The process plant includes the unit operations for 2 stages of crushing, grinding, flotation (rougher with 2 stages of cleaning), dewatering, concentrate handling, classification of tailings for backfilling and slimes for delivery to a tailings pond.
- Flotation concentrate produced will be packaged in 3t bulk bags and removed on a regular basis to a concentrate storage shed until the commencement of the shipping season.
- Backfill produced will be stored in a 200t tank until advised by underground when a pour is needed. Excess tailings and slimes will report to the tailings pond. The delivery system for backfill to underground has not been included in the estimate.
- Mill process water will be either from the mine, or if clear and free of sediments the tailings decant water from the tailings dam.
- Power for the process plant will be from the MCC room located in the mill building. Cost for the MCC are included in the plant capital estimate.

The equipment list follows. Details can be found in the Jasmin Yee report at Canarc's office.

Crushing	Primary Jaw	(1) 600 X 900
	Secondary – Shorthead Cone	(1) 1275
Grinding	Ball Mill 3m Ø X 3.9	650 hp
Flotation	Rougher – 2 banks	2.8 m3 /cell
	Cleaners – 2 stages Columns	(2) 1.0m Ø X 8.0m high
Dewatering	Thickener	6 m Ø
Conc. Handling	Shed	3t bags
Backfill	Cyclones 2 stages	various
Tailings	Slurry pumps	(2) 125 X 100mm

It is recommended that further test work be done to refine the process design and that more detailed process design be undertaken to upgrade the accuracy of the estimate in future studies.

25.1.3 Local Infrastructure

The on site facilities will be located where the townsite for the previous operations were located. The facilities are based on supporting the underground and milling operations including on-site power generation feeding a local electrical distribution system. The ongoing operation is based on a fly-in fly-out camp using barging of major supplies up the Taku River on a seasonal basis from Juneau Alaska. Personnel transportation and short term supplies will be delivered via fixed wing aircraft to the landing strip on site. Since access to the site for major supplies will use shallow barges up the Taku River it is likely that a large number of supplies may be purchased in Juneau. Supplies not available in Juneau will be purchased in Southern BC, Canada or the United States and shipped to site through Juneau. Barging will be done utilizing independent contractors who have the required equipment and the necessary experience with this type of service.

The site facilities will include a tailings pond for mill tailings which cannot be used as backfill. Quantification and costing studies for paste back fill will be conducted in future studies.

(from "New Polaris Gold Mine Ltd. Surface Infrastructure Cost Estimate" – Ken Chamberlin April 2007 on file at Canarc)

> This surface infrastructure costing has been conducted without an inspection of the proposed operational site. All equipment design, sizing and selection have been done without review by a professional engineer. Equipment and their proposed use are for pricing and discussion only, no environmental discussion or input has been received. All building and equipment positioning has been done using the existing site topographical plans; the final locations are yet to be determined. The insitu soil conditions are estimated based upon information provided by Canarc Resources employees. A loaded daily labour wage is based on $650 per employee. These construction cost estimates take into account that the process plant construction equipment i.e.; crane, Hi-ab, D8 dozer etc. are available for use on the surface infrastructure during the Process Plant construction, additional costs for this equipment has been included in the Installation Costs. It is also assumed that the service haul road to the Taku River barge landing is restored and can be traveled with a tractor trailer carrying 80,000 lbs. at an average speed of 15 kilometres per hour. Items valued over $1M have been RFP to multiple vendors. All vendor quoted prices are within +/- 20% and no negotiations to lower the cost have been done. Prices shown in ($00.00 Budget) style refers to unquoted items with pricing from verbal or historic data. All new items quoted have brochures and documents attached in the Appendix. The Airport Improvements are a budget amount with further study required to economically improve air transportation to site. Portable Fire Fighting equipment from 10 & 20 lbs. CO_2 and chemical powder to 305 lbs. wheeled units are included as required throughout the budget. The Total Cost figure is the Capital Cost + Shipping + Installation + Taxes and is the figure used in the last column of the Cost Summary and is used in the Design Criteria document.

The following items are included:

- 100 Man Camp (13 units)
- 3 Megawatt Generators (3 units)
- 600V Surface Distribution
- Compressors 1500 cfm (4 units)
- Sewage Treatment (1 unit)
- Waste Sludge Dewatering (1 unit)
- Waste Incinerator (1 unit)
- Truck Maintenance Shop and Mine Dry
- Potable Water Supply (Utillador)
- Fuel Tank Farm (14 units and Berm)
- Fuel Transport System (Used Tanks)
- Motor Grader / 950 Loader (2 units)

- Concentrate Truck & 2 – 48’ Trailers
- Three Site Services Pickups
- Mine & Mill Crew Bus
- Two Fork Lift Units
- Haul Road Restoration
- Airport Improvements

Future studies are recommended for:

- Using local power and road facilities if and when they become available form other local projects.
- Use of paste back fill to reduce tails pond disposal requirements
- Detailed site layout and facilities design for a higher level of cost estimate.
- Include water management plan, and reclamation planning in the site layout design

25.2 Environmental

The project will be designed to minimize environmental impacts during the construction and operating phases of the mine and to minimize any long term environmental impacts.

Canarc has retained URS Canada Inc to assess the tailings and development rock. Testing is currently underway by static testing conducted on 27 “fresh” rock drill core samples, collected from sections of the C vein, and flotation testing was conducted on 5 test tailings samples. A criterion of 2.0 weight % sulphide sulphur has been developed to distinguish between non-acid generating and potentially acid generating materials.

Kinetic testing (humidity cell) of the vein rock over a twelve week period indicates the sulphide oxidation is occurring, however, carbonate dissolution (i.e. acid neutralization) is also occurring in response to the sulphide oxidation. Arsenic and antimony loading from vein rock is low. Loadings from other potential metals of concern (Cr, Cu, Ni) are very low.

Sub-aqueous tailings column testing indicates the sulphide oxidation and acid neutralization as carbonate dissolution are occurring. Loadings are highest in tailings pore water, however, loadings are also increasing in the overlying cover water due to geochemical interactions between tailings, pore water and cover water. Antimony loadings were particularly elevated in tailings pore water, however, acid neutralization due to carbonate dissolution is occurring. Limited molar carbonate ratios for the first three weeks indicates that and pore water and cover water indicate that carbonate dissolution may not be adequate for complete neutralization, however, the pH of pore water and cover water were near-neutral and showed a slight increase over the first twelve weeks.

It is recommended that humidity cell testing continues to operate to twenty-six weeks to provide a longer term assessment of loadings and provide long term data for predicting time to sulphide and carbonate depletion.

Tailings storage facilities will be constructed on the flood plain adjacent to the site, above the high water level of the river. The dikes will be protected with Rip-Rap to protect them from erosion during extremely high water events.

Waste rock and tailings use for filling underground openings following mining will be maximized as well to minimize the amount of surface storage needed.

The waste rock and tailings currently show that they are both non acid generating in nature, due to high carbonate content, and will pose no environmental risk from acid generation. It is recommended that the testing continue and additional samples are taken as well as a full property environmental baseline and reclamation plan is developed as the project advances.

25.3 Marketing

Concentrate production of 25,000 to 30,000 tonnes annually will be shipped from site using 3 tonne bulk bags destined for ocean shipment from Juneau to other processing facilities in the United States or Asia. Facilities will be constructed at site for safe storage of the concentrate between October and May. Concentrate will be shipped on the return leg on the barges used to haul bulk supplies to the site.

The cost model used in this study assumes the concentrate is shipped to a contract autoclave and refinery in Nevada. Since part of the shipping is ocean transport, the option to use Asian refineries is being pursued if they improve project economics.

It is recommended that further studies be done with respect to refining and marketing the concentrates.

25.4 Socio-Economic

The mine is located in an area that has low usage for mining, exploration, hunting, fishing, trapping and logging activities. The New Polaris site was previously mined between 1938 and 1956 with remnants of the old activities still being present at the site.

The project is located within the land claim and traditional territory of the Taku River Tlingit First Nation (TRTFN).

No formal agreements are in place with the Taku River Tlingit First Nation. Discussions have taken place in previous years but no agreements completed. It is expected that the project will enhance opportunities for the people of the TRTFN during the construction and operation. These will include opportunities for employment during the construction and operating life of the mine. Operational training as well as trades training opportunities will also be made available for the members of the TRTFN on a preferential basis. A number of other benefits will accrue to Atlin through funding of social events, scholarships for higher education, and community enhancement programs.

During the exploration phase of the project a high percentage of employees have been from Atlin and the surrounding area. It is the intention of Canarc to continue to operate in a fashion that ensures the local community and its citizens continue to benefit from the construction and operation of the mine.

The sourcing of qualified and experienced underground miners, process personnel, and tradesmen is a concern, particularly with the current labour shortages in Western Canada. Canarc will continue to source appropriate personnel as the project advances. A contract mining company will also be pursued.

Since access to the site for major supplies will use shallow barges up the Taku River it is likely that a large number of supplies may be purchased in Juneau. Supplies not available in Juneau will be purchased in Southern BC, Canada or the United States and shipped to site through Seattle and Juneau. Barging will done using independent contractors who have the required equipment and have the necessary experience with this type of service. The major items needed for the operation will be diesel fuel, ground support supplies, mill reagents & supplies, explosives and a variety of components for equipment maintenance.

The current plan is based on a fly-in fly-out rotation and an onsite camp. Air transportation will be used for transporting employees and perishable items or small items needed to sustain the operation. As other projects in the area are developed the opportunity may arise to use access and infrastructure developed for these other projects.

Continued work in negotiations with First Nations, manpower planning, and the impact on local infrastructure is recommended.

25.5 **Production Schedule**

25.5.1 Mining Rate

A milling rate of 600 tonnes per day for 365 days/year is targeted. Therefore, a total of 219 kTonnes of material are scheduled to be processed each year.

The following table shows the level by level tonnages and grades that are contained in the mining limits using the 9gpt cutoff grade resource boundary.

Table 25-9 Mining Tonnage and Grade by Extraction Level

Level	Upper Elevation	Lower Elevation	Mineralized and Diluted		**Recovered**	
	m	m	Tonnes	Grade (gpt)	**Tonnes**	**Grade (gpt)**
1	-140	-190	293,774	12.88	**264,397**	**12.88**
2	-190	-240	356,797	13.37	**321,117**	**13.37**
3	-240	-290	299,486	12.83	**269,537**	**12.83**
4	-290	-340	189,977	13.22	**170,979**	**13.22**
5	-340	-390	148,750	14.06	**133,875**	**14.06**
6	-390	-440	216,427	12.55	**194,784**	**12.55**
7	-440	-490	182,760	10.60	**164,484**	**10.60**
8	-490	-540	137,766	10.27	**123,989**	**10.27**
9	-540	-590	121,253	10.58	**109,128**	**10.58**

The mineralized and diluted tonnes and grade are a product of the material within the >9gpt mining limits. The recovered tonnes are a sum of the mineralized and diluted tonnes, and factored down based on the mining recovery.

Mining will begin in Year 1 on level 1 for easy access, then progress downwards. The results of this scheduling strategy are shown in the table below.

Table 25-10: Production Schedule showing kTonnes and Grade by Year off each level

Mining Zone		Totals	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Level 1	Production (kT)	**264**	219	46						
	Mill Grade (g/t)	**12.88**	12.88	12.88						
	Total Au (kg)	**3,407**	2,818	588						
Level 2	Production (kT)	**321**		173	148					
	Mill Grade (g/t)	**13.37**		13.37	13.37					
	Total Au (g)	**4,295**		2,315	1,980					
Level 3	Production (kT)	**270**			74	199				
	Mill Grade (g/t)	**12.83**			12.83	12.83				
	Total Au (kg)	**3,459**			908	2,552				
Level 4	Production (kT)	**171**				20	151			
	Mill Grade (g/t)	**13.22**				13.22	13.22			
	Total Au (kg)	**2,260**				264	1,996			
Level 5	Production (kT)	**134**					68	66		
	Mill Grade (g/t)	**14.06**					14.06	14.06		
	Total Au (kg)	**1,882**					952	930		
Level 6	Production (kT)	**195**						129	66	
	Mill Grade (g/t)	**12.55**						12.55	12.55	
	Total Au (kg)	**2,445**						1,614	830	
Level 7	Production (kT)	**164**						24	141	
	Mill Grade (g/t)	**10.60**						10.60	10.60	
	Total Au (kg)	**1,743**						254	1,489	
Level 8	Production (kT)	**124**							12	112
	Mill Grade (g/t)	**10.27**							10.27	10.27
	Total Au (kg)	**1,273**							124	1,149
Level 9	Production (kT)	**109**								107
	Mill Grade (g/t)	**10.58**								10.58
	Total Au (kg)	**1,154**								1,130
Annual Totals	Production (kT)	**1,752**	219	219	219	219	219	219	219	219
	Mill Grade (g/t)	**12.51**	12.88	13.27	13.20	12.87	13.48	12.79	11.17	10.42
	Total Au (kg)	**21,918**	2,818	2,903	2,888	2,815	2,948	2,798	2,444	2,279
	Total Au (oz)	**704,671**	90,614	93,344	92,836	90,509	94,778	89,971	78,564	73,277

25.6 **Project Schedule**

The Project Schedule is presented as a Preliminary estimate of the general tasks required to complete a full Feasibility Study for the New Polaris Project. The schedule will be updated as the project progresses and more information is gained. Included in the schedule is:

25.6.1 Schedule Items

1. & 2 Initial Resource Estimate & Technical Report

NI 43-10 Resource Potential Technical Report March 16, 2007.

3. Preliminary Assessment

Technical Study based on MII resources including Mining, Metallurgy, Processing, Infrastructure, G&A, Environmental, ARD, Production scheduling, and CapEx/OpEx costing to a scoping level of detail.

4 to 7. Rehab old workings

Pumping out and rehabilitating the old workings to gain secondary egress and ensure no water or sludge hazards threaten work on deeper levels, and to establish adequate mine ventilation for ongoing work on lower levels. This was completed in August, 2007.

8 & 9. Advance to exploration level

Drive an exploration decline to access the C vein between the 600 and 1050 levels. This decline will also be used for future production.

10 to 12 Exploration sampling

Drive level development on the 600, 750, 900 and 1050 levels to test C vein, which will also serve as future production levels. Face samples and channel samples and muck samples will be taken for assay. Cross-cuts will be made to test the hanging wall and footwall contacts and future mining conditions.

13 to 15 Update resource model

The development work and the past DD results will be used to update the resource model.

16 & 17 Resource Estimate & Pre-Feasibility

The new resource model will be used to update the resource estimate. The Mine Planning for the Pre-Feasibility/Feasibility Study will then be started. It may be possible to start the mine planning if the vein delineation and partial grade estimation of the model is sufficient to define a portion of the mine plan. The risk is that changes to the final resource model may cause changes to the mine planning.

18 Feasibility Study

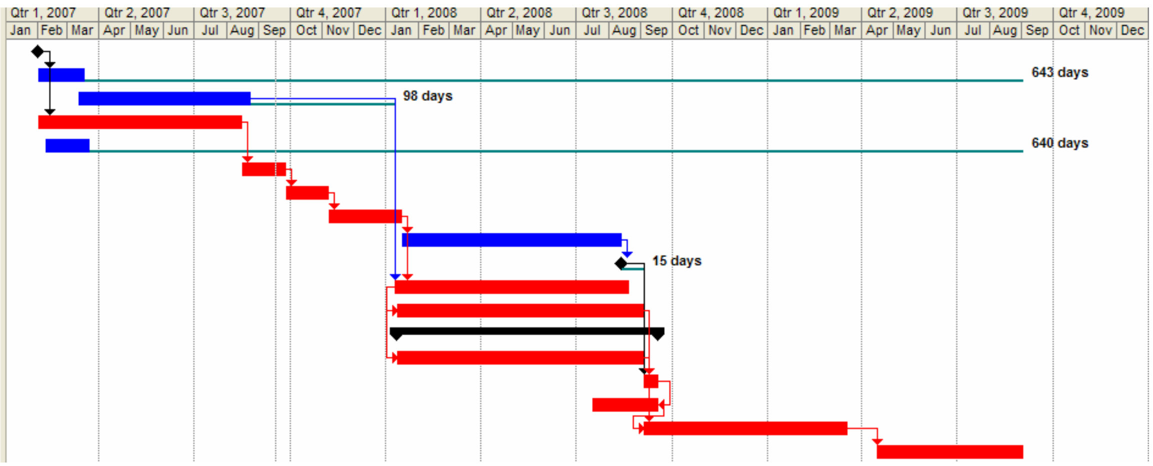
The Feasibility Study will identify opportunities and risks which will be addressed to improve the cost estimates. It may be possible to start the Feasibility Study before the Pre-Feasibility study is finished however it is assumed some management decisions will be needed between these stages of the project. It is also assumed that beyond the planned underground development and sampling program, no additional exploration will be needed for the Feasibility Study.

25.6.2 *Project Schedule*

The following project schedule includes all items discussed above

Table 25-11 General Project Schedule

Task Name	Duration	Start	'redecessor	Finish
Initial Resource Estimate	0 days	Feb 1 '07		Feb 1 '07
Technical Report - Resource Estima	32 days	Feb 1 '07	1	Mar 16 '07
Preliminary Assessment Study	120 days	Mar 12 '07		Aug 23 '07
Pumping out current workings	140 days	Feb 1 '07	1	Aug 14 '07
Mobilization	30 days	Feb 8 '07		Mar 21 '07
Rehab of old workings	30 days	Aug 15 '07	4	Sep 25 '07
Drift Dev	30 days	Sep 26 '07	6	Nov 6 '07
Development to C Vein (250m)	50 days	Nov 7 '07	7	Jan 15 '08
Exploration Drift in C Vein (750m)	150 days	Jan 16 '08	8	Aug 12 '08
Bulk Sample for Met Test	0 days	Aug 12 '08	9	Aug 12 '08
Exploration Drilling	160 days	Jan 9 '08	FS-5 days,3	Aug 19 '08
Assays of Exploration Work	168 days	Jan 11 '08	1SS+2 days	Sep 2 '08
Update Resource Model	**178 days**	**Jan 11 '08**		**Sep 16 '08**
Update Geology	168 days	Jan 11 '08	1SS+2 days	Sep 2 '08
Update Geostats and Re-Inte	10 days	Sep 3 '08	14,10	Sep 16 '08
Resource Estimate	45 days	Jul 16 '08	15FF	Sep 16 '08
Pre-feasibility	140 days	Sep 3 '08	-10 days,12	Mar 17 '09
Feasibility Study	100 days	Apr 15 '09	FS+20 days	Sep 1 '09



25.7 Financial Evaluation

The basis of the project cash flow analysis is summarized below. Details of the Capital and Operating cost estimates can be obtained from Canarc.

Economic Parameters in Study

Parameter	Value	Unit
Gold Price	$650	US/oz
Currency Exchange Rate	0.90	$US:$CDN
Net Smelter Price (after refining, transport, marketing, etc.)	$19.22	CDN/gram
Mining Cost (includes staff costs)	$70	CDN/tonne
Development Costs	$3,000	CDN/meter
Processing Costs	$24	CDN/tonne
General & Administration Costs	$10	CDN/tonne
Camp Costs	$6	CDN/tonne
Surface Equipment and Facilities Costs	$15	CDN/tonne
Mining Dilution	13%	
Mining Recovery	90%	
Mining Rate	600	tonnes/day
Process Recovery	91%	
Mining Cutoff Grade	9	grams/tonne
Average Grade Mined (after dilution)	12.51	grams/tonne
Total Tonnes Mined (including dilution)	1,750,000	tonnes

In line with the level of the preliminary assessment study, by NI43-101 standards, it is estimated that the data used, and the conclusions reached are within ±40% for this report.

The following table shows the results of a cash flow analysis for the project, using the inputs shown above, as well as the tax structure that would apply to the project. Based on this analysis, the project's undiscounted Net Present Value after 9 years of operations is $41 million, and would require $72 million of capital investment after the pre-feasibility program.

Based on the above positive results it is recommended that the project be further evaluated.

The preliminary assessment is based on resources, not reserves, and a portion of the modeled resources to be mined are in the inferred resource category. Resources are considered too speculative geologically to have economic considerations applied to them so the project does not yet have proven economic viability.

The New Polaris current resources were previously disclosed in a news release dated February 1, 2007 and in a NI 43-101 technical report filed on SEDAR on March 15, 2007.

Cash costs include site related costs prior to the shipping and sale of concentrates. Offsite costs for concentrate transportation and processing were treated as deductions against sales.

The Net Present Values are life of mine net cash flows shown at various discount rates. The Internal Rates of Return assume 100% equity financing.

The New Polaris project is sensitive to the price of gold. At US$600 per oz gold, the pre-tax NPV is CA$26 million with an IRR of 7%, or an after tax NPV of CA$18 million and an IRR of 5% and at US$750 gold, the pre-tax NPV jumps to CA$130 million and the IRR increases to 29%, or the after tax NPV increases to CA$87 million and the IRR increases to 22%.

The project is also sensitive to the US$/CA$ exchange rates and gold recoveries. Each 1% change in the US$/CA$ exchange rate indicates a change of approximately $0.3 million CA$ in after tax undiscounted NPV. Changes in recoveries produce similar results.

Opportunities exist to improve the base case model such as:

- Increasing resources and therefore mine life;
- Increasing gold recoveries and concentrate grades;
- Increasing production to enhance economies of scale;
- Reducing transportation costs; and
- Reducing offsite processing costs.

The main cost risks include:

- Rising engineering and construction labour and equipment costs due to limited availability;
- Escalating capital costs if there are project delays;
- Rising operating costs due to inflation and commodity shortages; and
- Fluctuations in US$/CA$ exchange rates.

Additional flotation test work is now underway to try and improve gold recoveries and concentrate grades. Autoclave and bio-leach test results will also be completed within the third quarter of 2007, which should allow management to initiate discussions with potential buyers of the New Polaris concentrates.

The base case model includes conventional barging of concentrates off-site during the summer season. However Canarc's neighbor, Redcorp Ventures, has applied for permits to operate year-round air cushion barges, ("ACB's") towed by amphibious vehicles ("Amphitracs"). Such technology may also be advantageous for Canarc and will be evaluated as part of a feasibility work program.

A feasibility work program to include driving a decline from surface down to the 1050 mine level (1000 feet below surface), developing one or more drifts and raises within the C vein, "trial" mining to extract a bulk sample, shipping and processing of a representative portion of the bulk sample for final metallurgical testing, finalizing the process flow sheet and completing a feasibility study at an estimated cost of CA$18.7 million, subject to financing.

Table 25-12 New Polaris After-Tax Project Cash Flow Analysis

Indicative Only — 13-Aug-07

NEW POLARIS MINE

Average Head Grade 12.51 gAu/t
Conceptual Financial Analysis after tax

$(000)s

Resources Tonnes (000)'s 1,750

Description	-2	-1	1	2	3	4	5	6	7	8	9	Total
Production Ton/year			218,750	218,750	218,750	218,750	218,750	218,750	218,750	218,750		1,750,000
Au Grade gAu/toz/t			12.88	13.27	13.20	12.87	13.48	12.79	11.17	10.42		12.51
Recovery Au % Concentrate			91.00%	91.00%	91.00%	91.00%	91.00%	91.00%	91.00%	91.00%		91.00%
Au ounces payable Concentrate			82,459	84,944	84,481	82,363	86,248	81,874	71,493	66,682		640,544
Gross Revenue			49,090	50,827	50,503	49,024	51,738	48,682	41,428	38,066		379,358
Operating Costs			29,376	27,856	26,884	26,804	29,735	29,824	29,803	28,067		228,349
Revenue Before Taxes			19,715	22,971	23,619	22,220	22,004	18,858	11,625	9,999		151,009
Federal Income Tax						2,094	3,113	2,804	1,556	1,431		10,998
BC Income Tax			394	459	472	1,491	1,997	1,779	1,011	915		8,519
Revenue Before Capital Exp.			19,320	22,511	23,147	18,634	16,893	14,275	9,058	7,652		131,492
Capital Expenditures												
- Development/Construction	$15,461	$56,151										71,612
- On-Going Capital			$5,488	$3,358	$3,920	$3,994	$420	$567	$100	$100	$1,000	18,948
Working Capital Change			4,896								(4,896)	
Salvage												
Total Capital	$15,461	56,151	10,384	3,358	3,920	3,994	420	567	100	100	(3,896)	90,560
Net Cashflow	(15,461)	(56,151)	8,936	19,153	19,227	14,640	16,474	13,708	8,958	7,552	3,896	40,931
Discounted NCF 5%	(15,461)	(53,477)	8,105	16,545	15,818	11,471	12,293	9,742	6,063	4,868	2,392	18,359
Discounted NCF 8%	(15,461)	(51,992)	7,661	15,204	14,132	9,964	10,381	7,998	4,840	3,778	1,805	8,311
Discounted NCF 10%	(15,461)	(51,046)	7,385	14,390	13,132	9,091	9,299	7,034	4,179	3,203	1,502	2,707

Rate of Return 11.08% Payback Initial 4.66 years

Notes:

1. Metal Prices US $ Au/oz $650.00
2. Capital requirements based on 100% equity.
3. All funds are in Canadian $ except where noted.
4. Taxes are approximate

NSR/t $216.78

Beacon Hill Consultants (1988) Ltd

A table of the project sensitivity is presented below.

Table 25-13: After Tax Cash Flow Sensitivity Results

NEW POLARIS GOLD PROJECT
SENSITIVITY ANALYSIS -AFTER TAX

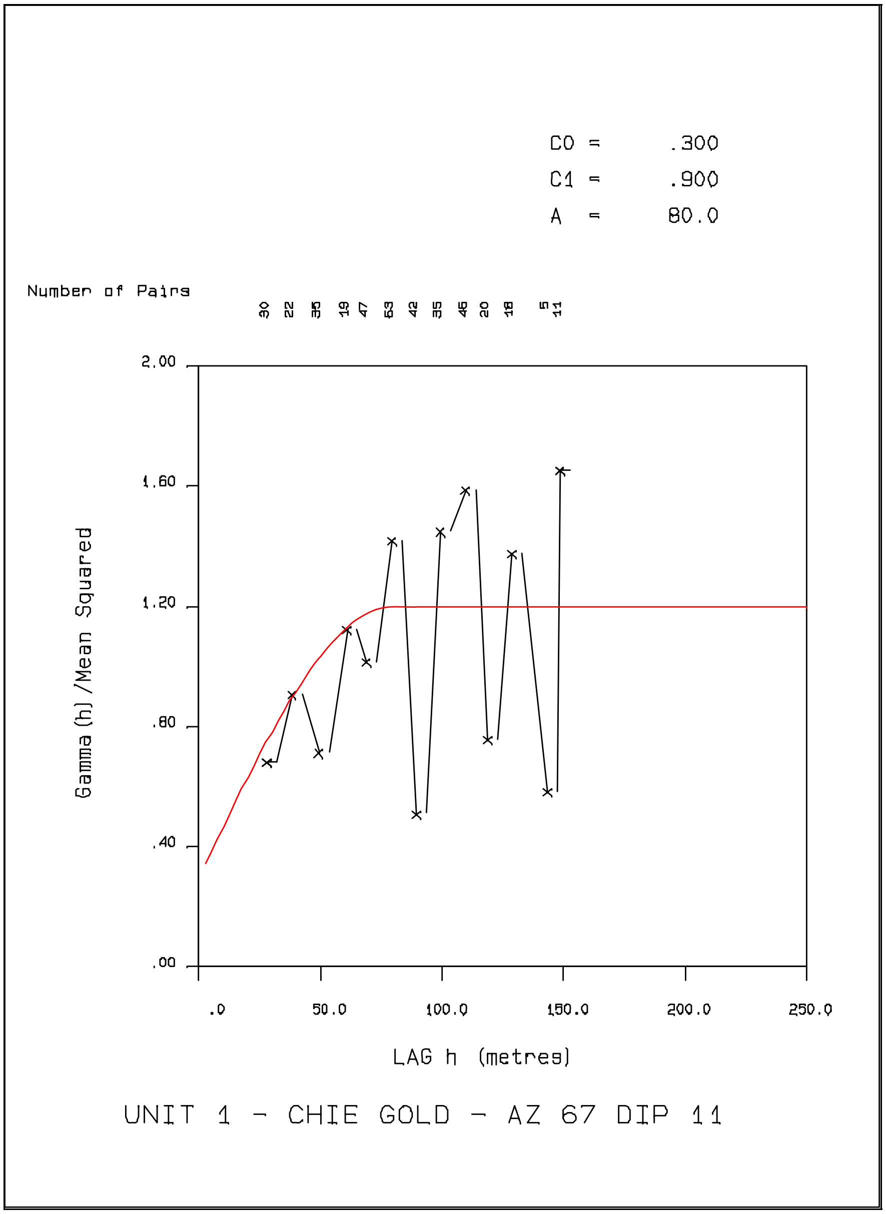
UPDATED AS OF 13-Aug-07

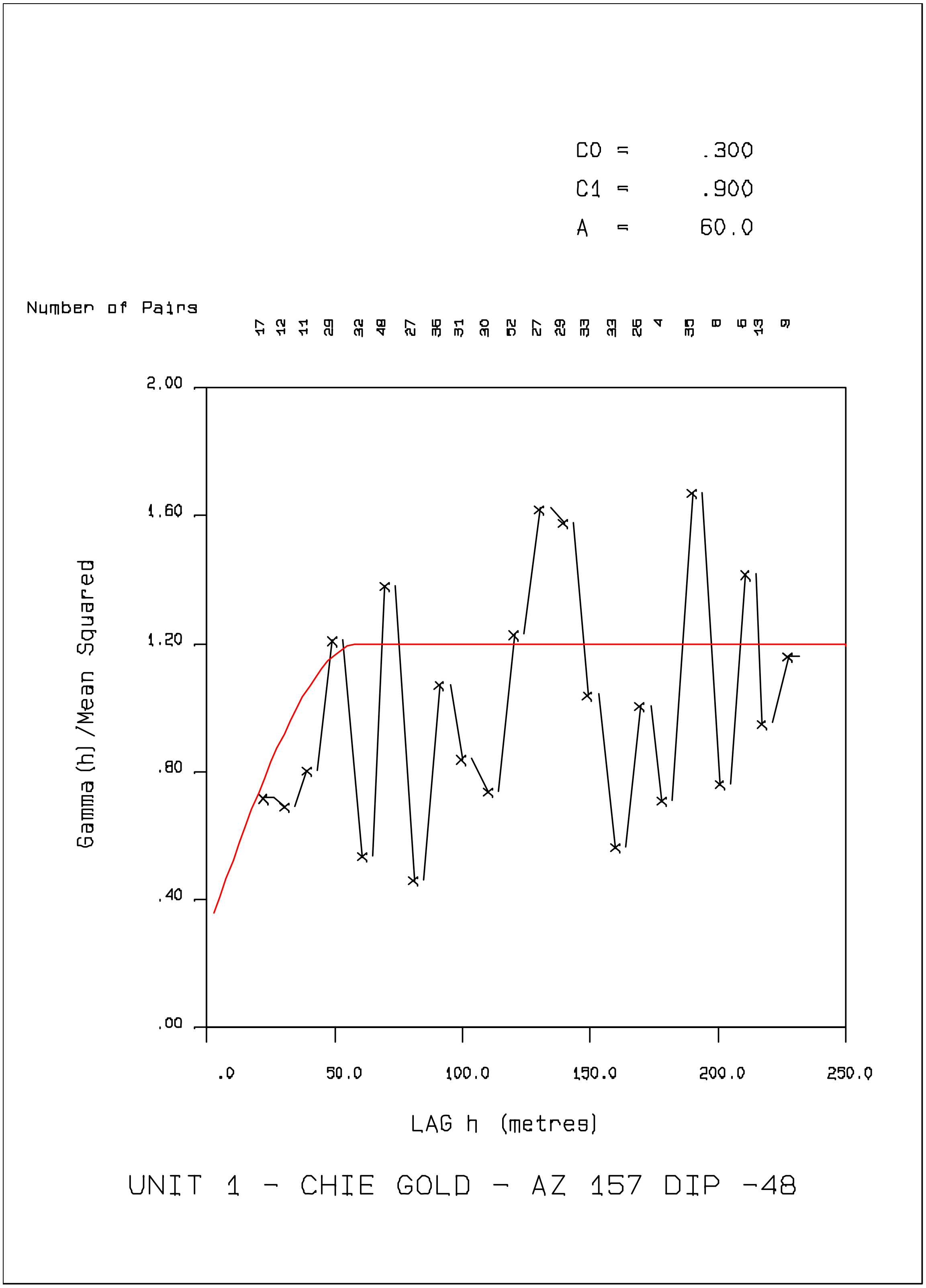
Case	Description of Sensitivity	NPV Dis.0%	NPV Dis.5%	NPV Dis.10%	IRR
		US$(000)s	US$(000)s	US$(000)s	%
CASE 1	**Base Case**	**$40,931.42**	**$18,358.80**	**$8,310.63**	**11.08%**
CASE2	Gold $550/oz	($10,582.61)	($22,420.30)	($27,517.01)	-3.18%
CASE3	Gold $600/oz	$17,856.13	$43.83	($7,814.28)	5.02%
CASE1	**Base Case**	**$40,931.42**	**$18,358.80**	**$8,310.63**	**11.08%**
CASE4	Gold $700/oz	$64,006.71	$36,451.89	$24,130.87	16.62%
CASE5	Gold $750/oz	$86,613.88	$54,166.05	$39,610.80	21.75%
CASE6	Grade -10%	$11,165.46	($5,351.91)	($12,603.06)	3.16%
CASE7	Grade -5%	$26,048.44	$6,587.71	($2,033.31)	7.23%
CASE1	**Base Case**	**$40,931.42**	**$18,358.80**	**$8,310.63**	**11.08%**
CASE8	Grade +5%	$55,814.40	$30,043.34	$18,535.07	14.71%
CASE9	Grade +10%	$70,697.38	$41,685.76	$28,700.98	18.16%
CASE10	Capital Cost -10%	$46,374.64	$23,947.37	$13,941.22	13.61%
CASE11	Capital Cost -5%	$43,653.03	$21,173.33	$11,154.06	12.31%
CASE1	**Base Case**	**$40,931.42**	**$18,358.80**	**$8,310.63**	**11.08%**
CASE12	Capital Cost +5%	$38,209.81	$15,544.27	$5,467.21	9.94%
CASE13	Capital Cost +10%	$35,488.20	$12,729.74	$2,623.79	8.90%
CASE14	Operating Cost -10%	$69,746.72	$41,038.13	$28,183.12	18.04%
CASE15	Operating Cost -5%	$55,737.02	$30,029.50	$18,546.26	14.73%
CASE1	**Base Case**	**$40,931.42**	**$18,358.80**	**$8,310.63**	**11.08%**
CASE16	Operating Cost +5%	$25,366.55	$6,011.05	($2,558.43)	7.03%
CASE17	Operating Cost +10%	$9,042.43	($7,115.00)	($14,193.47)	2.55%
CASE18	Exchange rate US$0.80=C$1.00	$72,465.59	$43,099.63	$29,949.68	18.59%
CASE19	Exchange rate US$0.85=C$1.00	$55,771.03	$30,023.84	$18,524.70	14.71%
CASE1	**Base Case**	**$40,931.42**	**$18,358.80**	**$8,310.63**	**11.08%**
CASE20	Exchange rate US$0.95=C$1.00	$27,653.87	$7,855.47	($920.02)	7.65%
CASE21	Exchange rate US$1.00=C$1.00	$15,704.08	($1,706.61)	($9,374.85)	4.42%

Beacon Hill Consultants (1988) Ltd.

Based on the above positive results it is recommended that the project be further evaluated.

APPENDIX A SEMIVARIOGRAM MODELS





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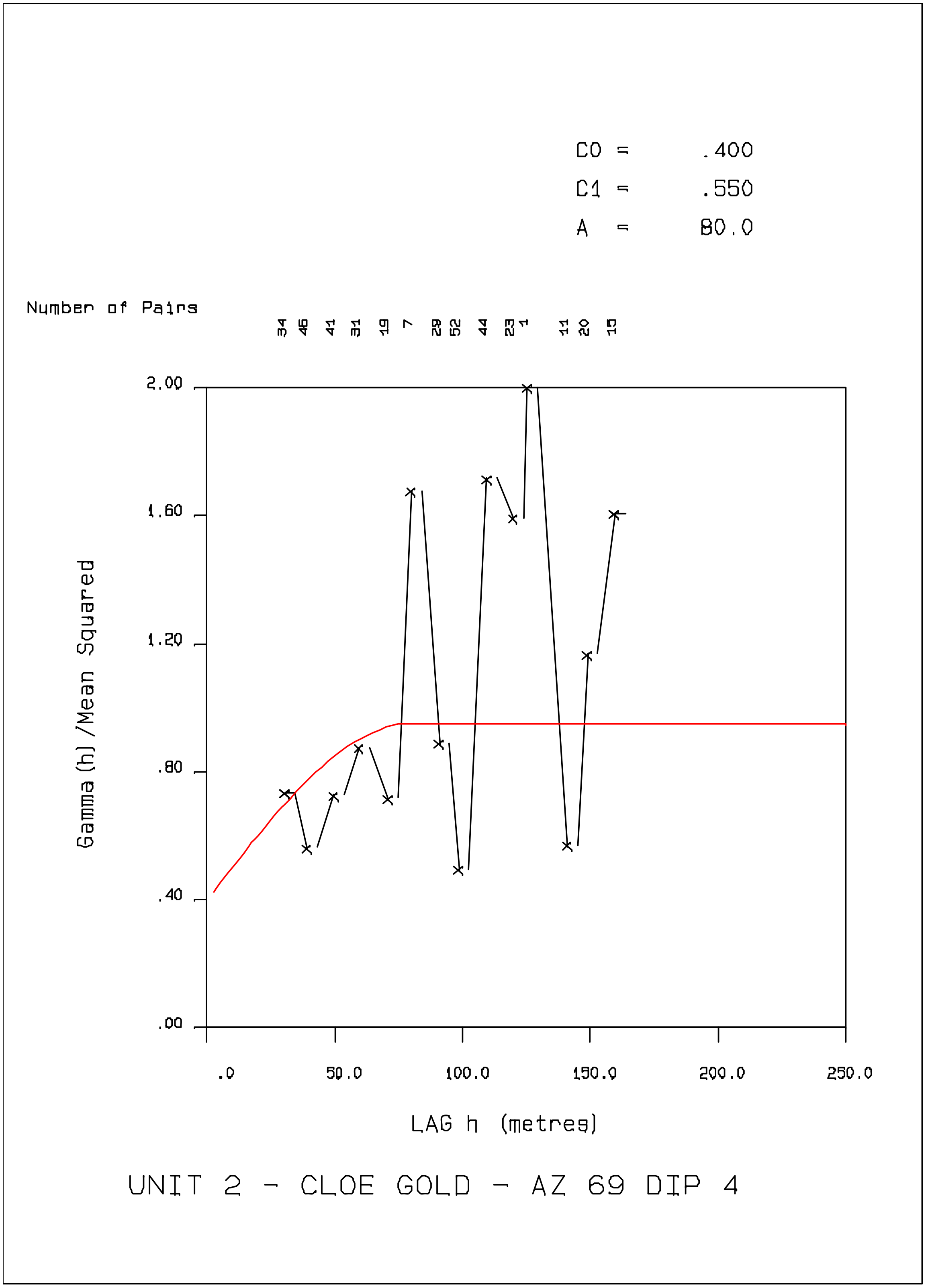
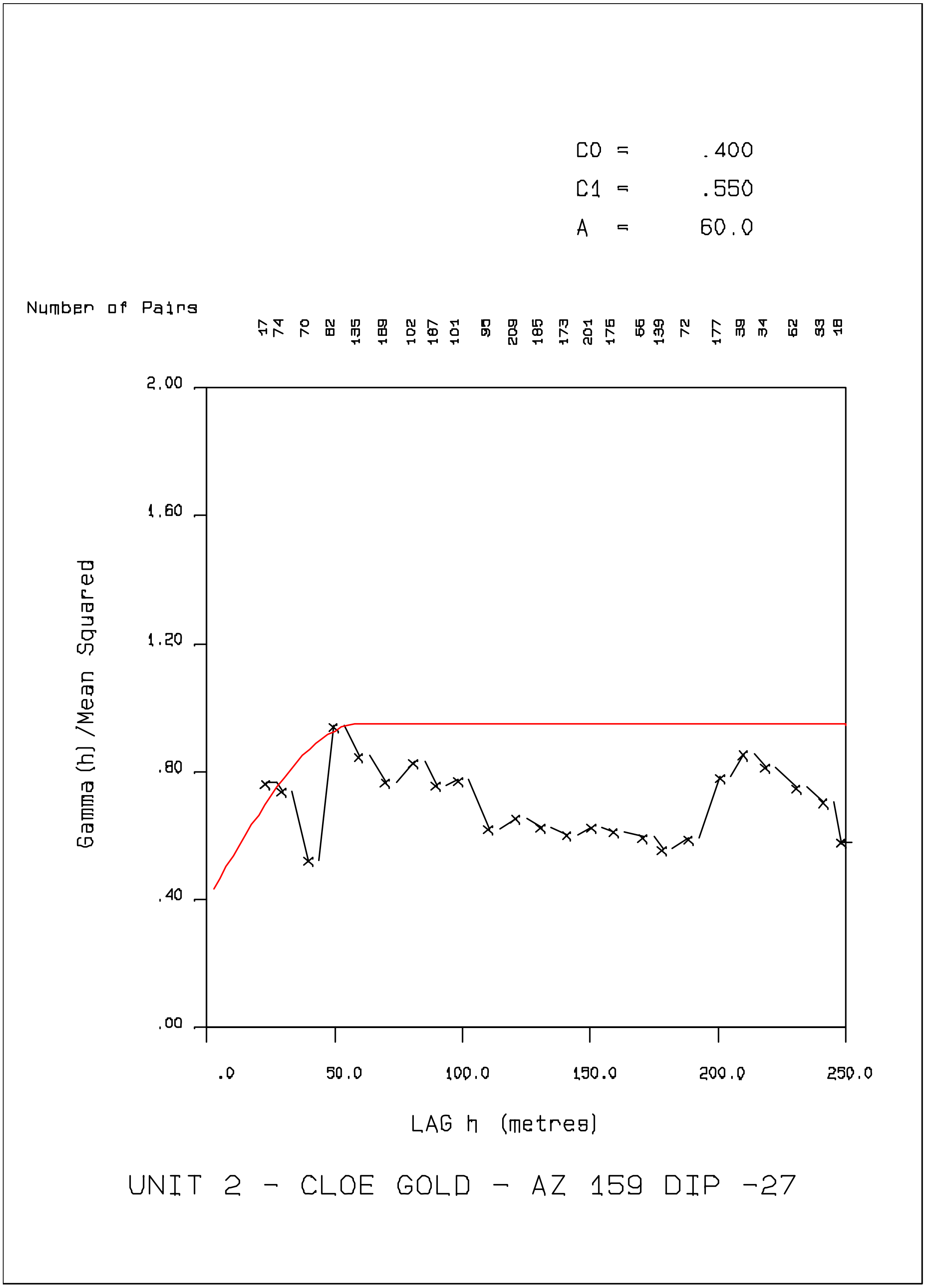
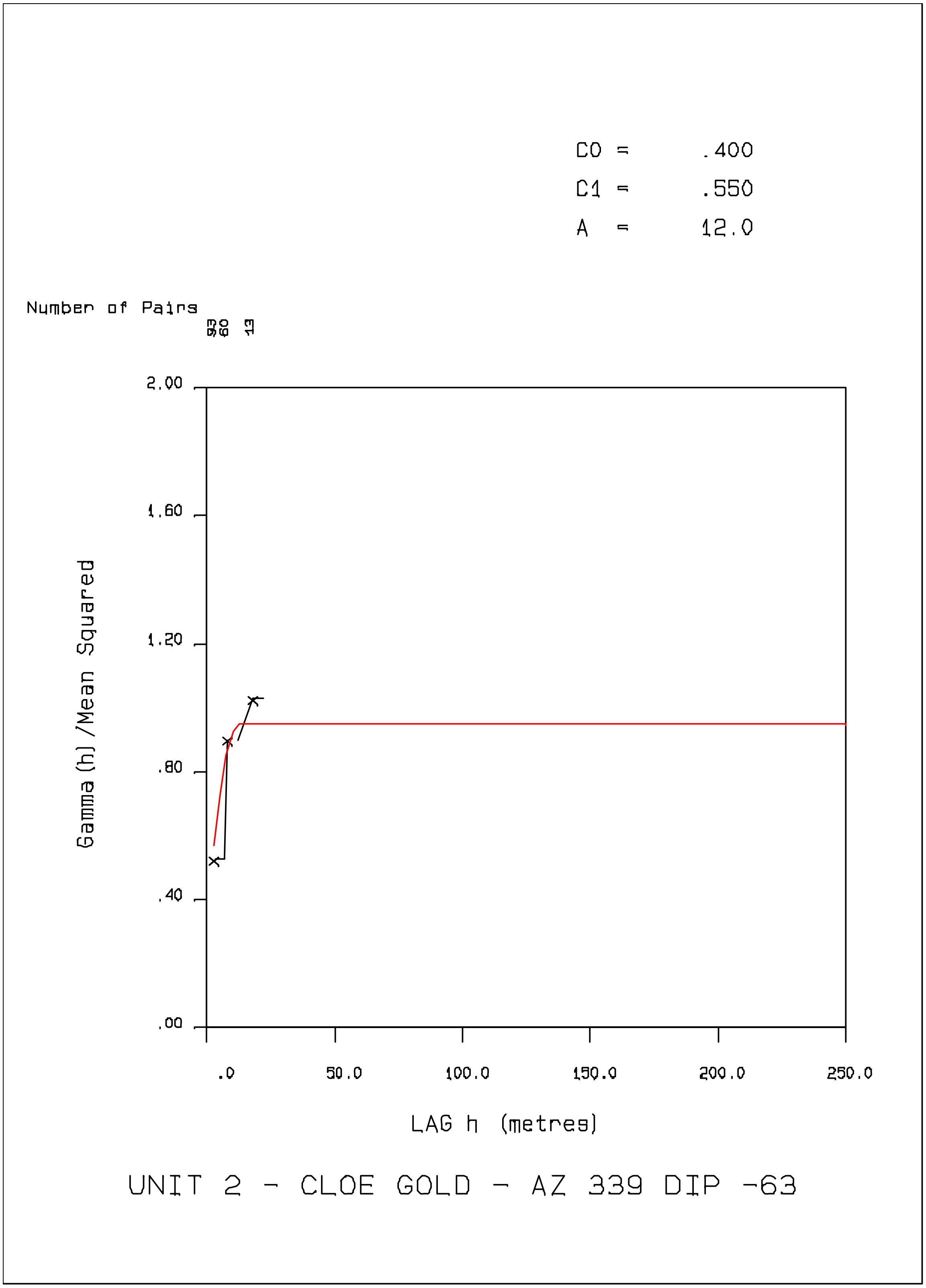
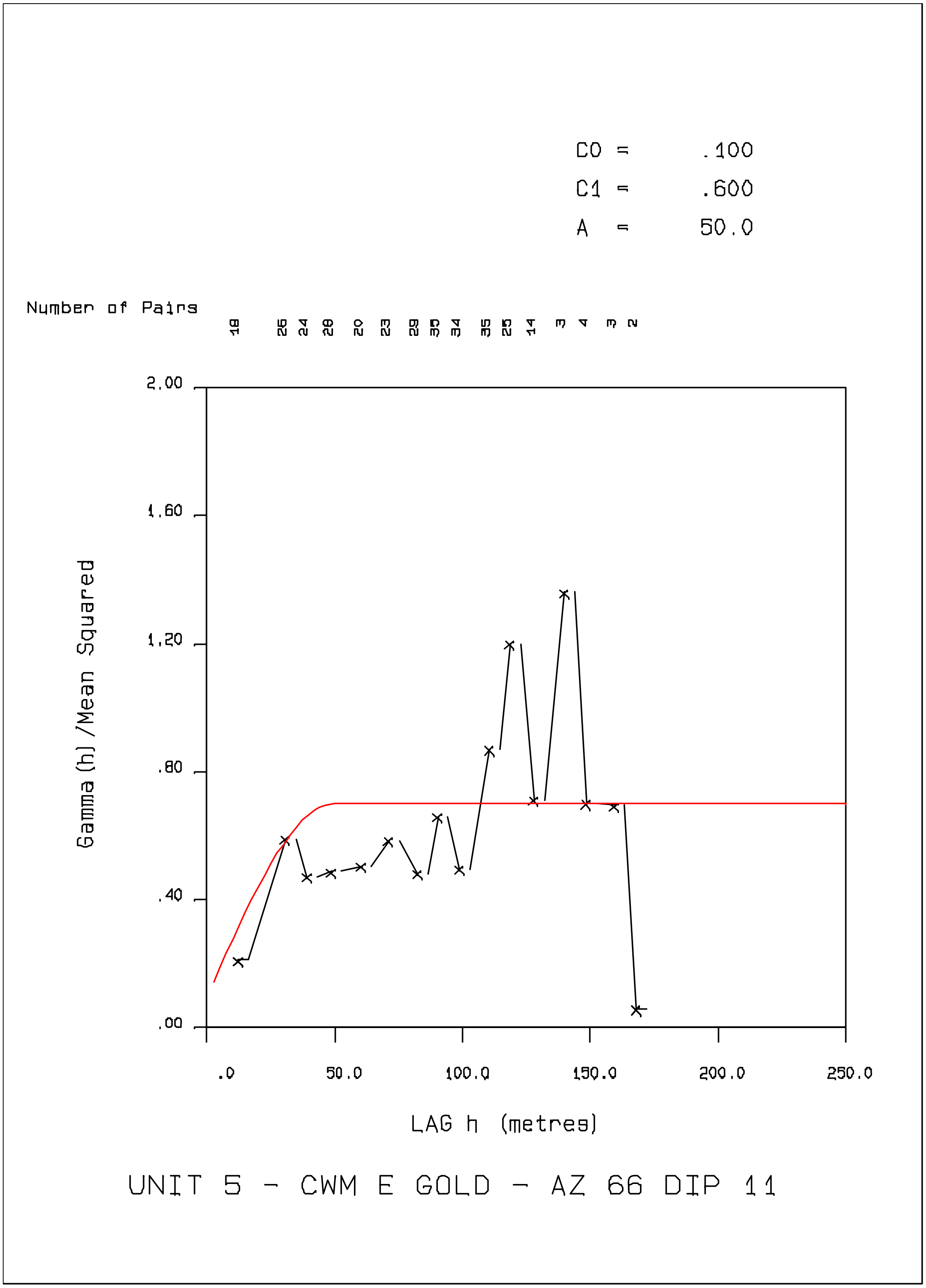
C1 = .900
A = 60.0
Number of Pairs
17 12 11 29 32 48 27 35 31 30 52 27 29 33 33 26 4 35 8 6 13 9
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
UNIT 1 - CHIE GOLD - AZ 157 DIP -48

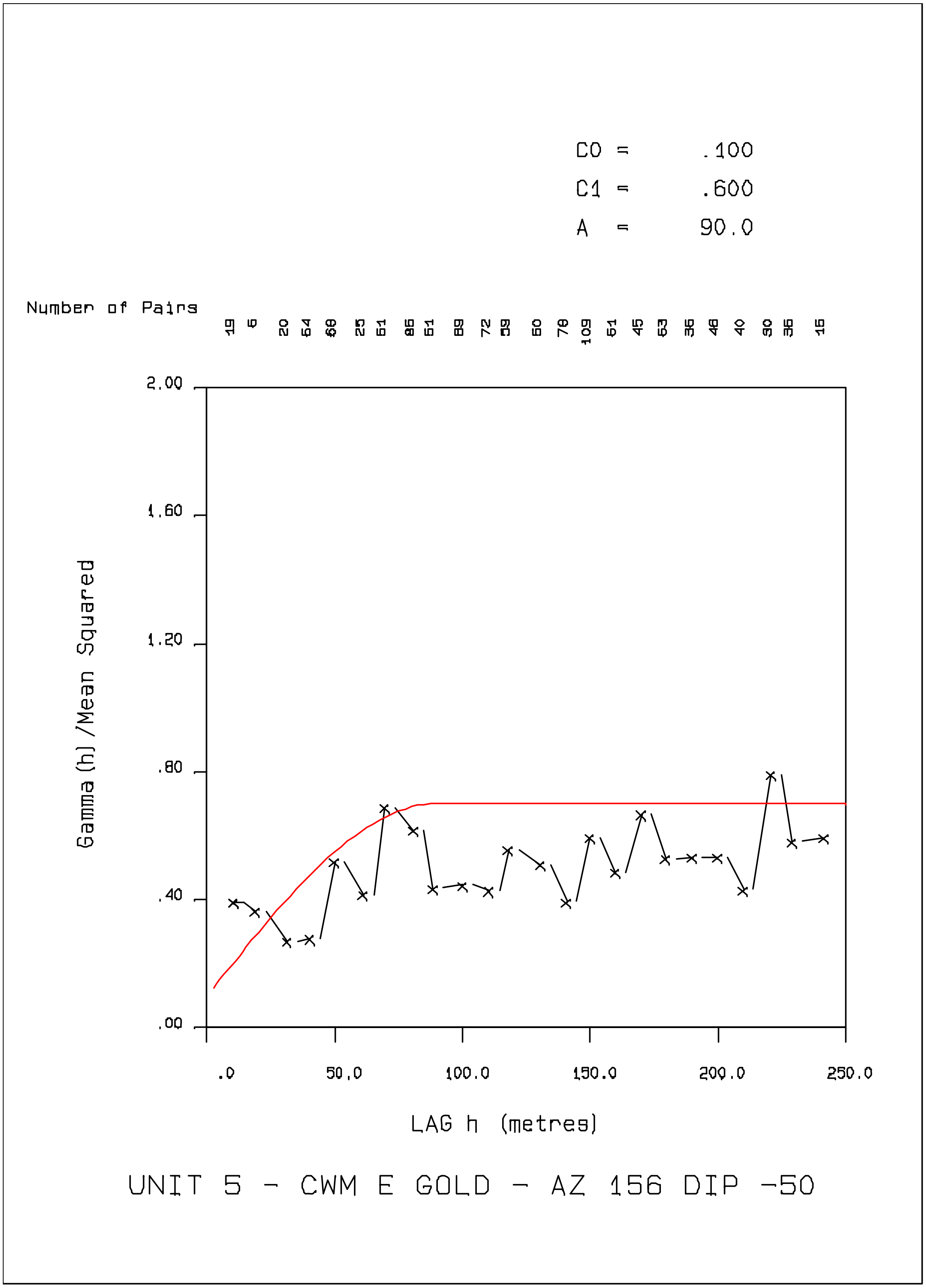


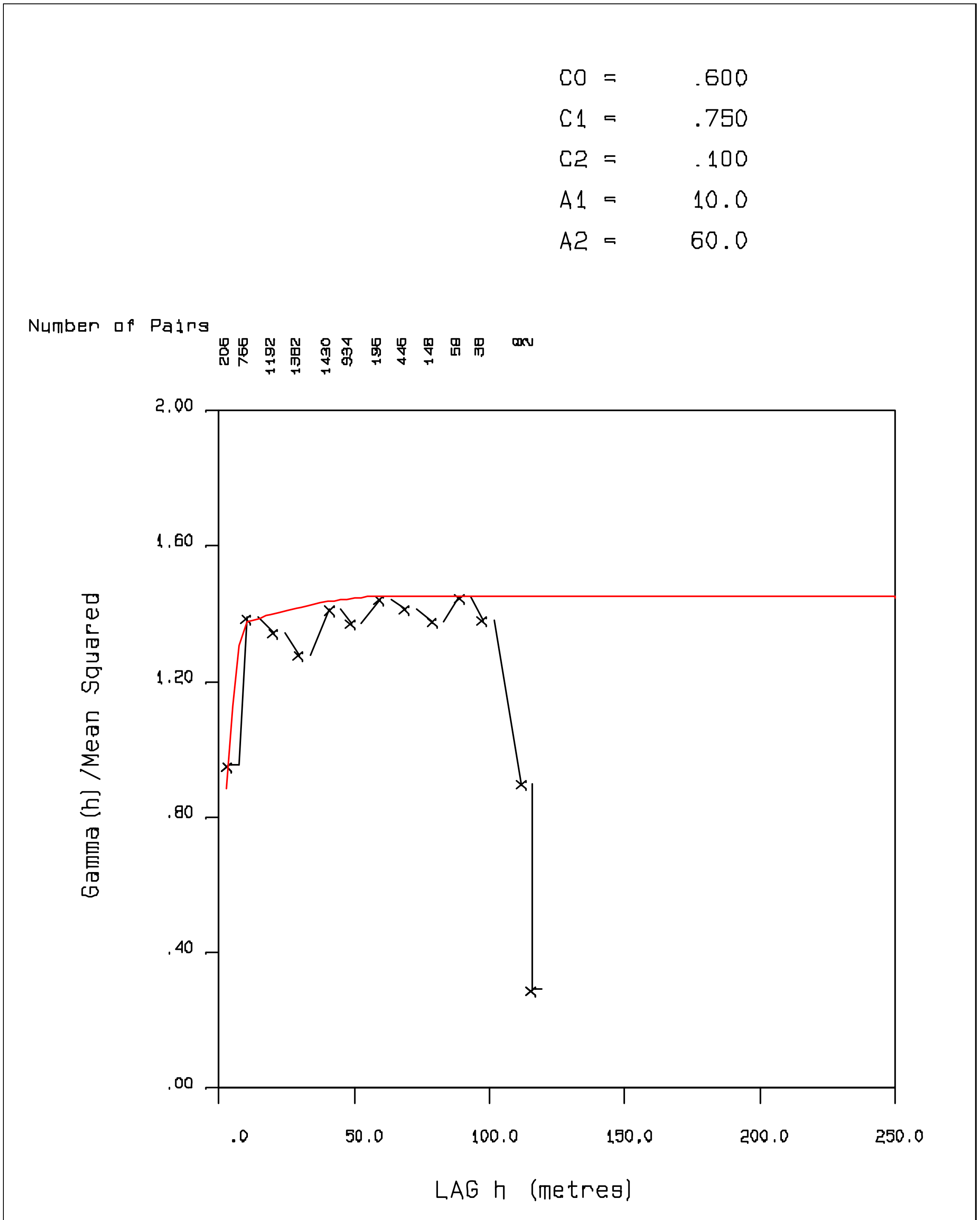
CO = .400
C1 = .550
A = 80.0
Number of Pairs
34 46 41 31 19 7 29 52 44 23 1 11 20 15
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
UNIT 2 - CLOE GOLD - AZ 69 DIP 4


C0 = .400
C1 = .550
A = 60.0
Number of Pairs
17 74 70 82 135 189 102 187 101 95 209 185 173 201 176 66 139 72 177 39 34 62 93 18
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
UNIT 2 - CLOE GOLD - AZ 159 DIP -27

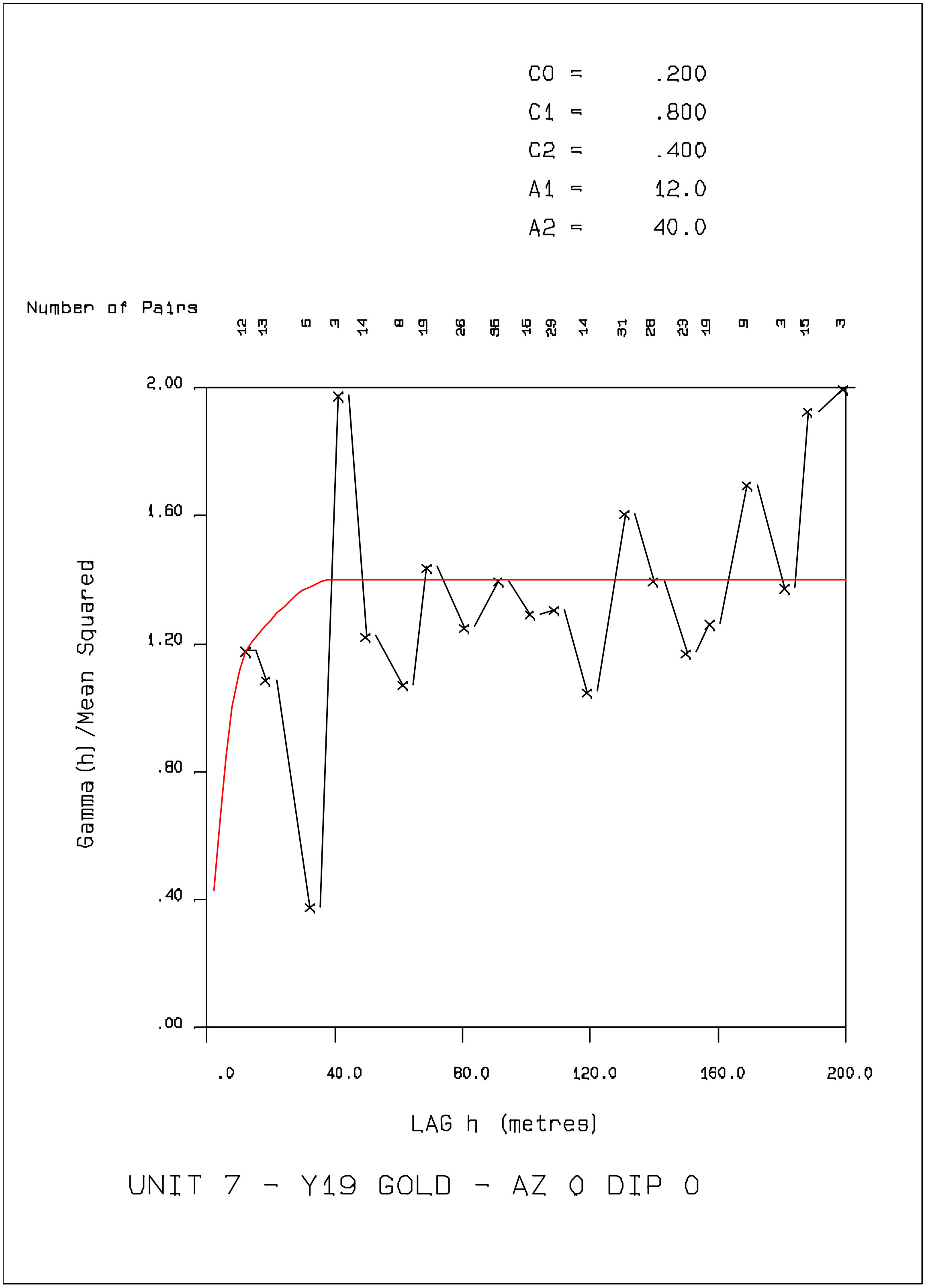

CO = .400
C1 = .550
A = 12.0
Number of Pairs
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
UNIT 2 - CLOE GOLD - AZ 339 DIP -63


C0 = .100
C1 = .600
A = 50.0
Number of Pairs
18 26 24 28 20 23 29 35 34 35 25 14 3 4 3 2
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
UNIT 5 - CWM E GOLD - AZ 66 DIP 11


CO = .100
C1 = .600
A = 90.0
Number of Pairs
19 5 20 54 68 25 61 85 61 89 72 59 50 78 109 61 45 63 36 46 40 30 35 15
Gamma (h) /Mean Squared
2.00
1.60
1.20
.80
.40
.00
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
UNIT 5 - CWM E GOLD - AZ 156 DIP -50



UNIT 6 - WBLOB GOLD - OMNI DIRECTIONAL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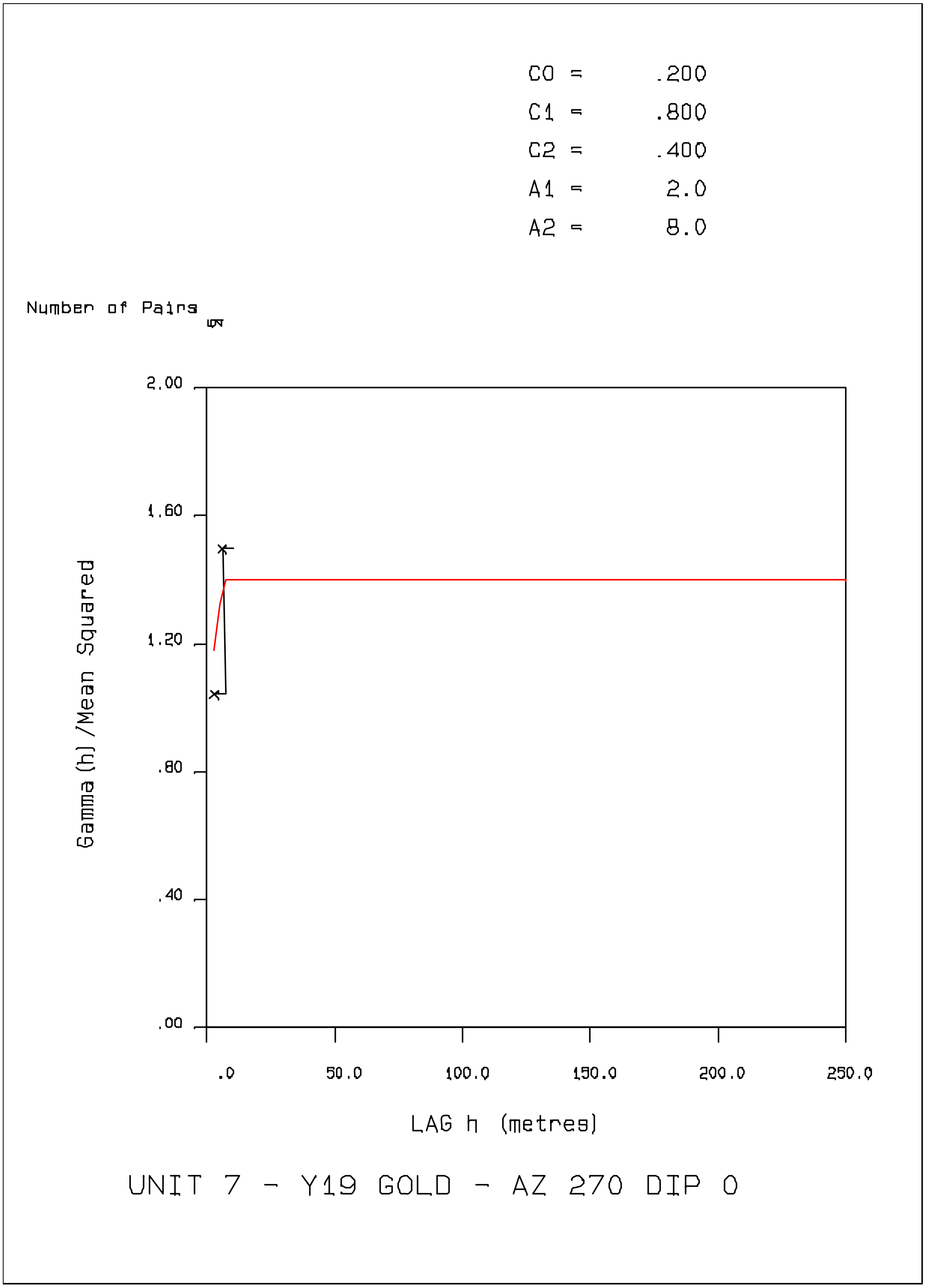
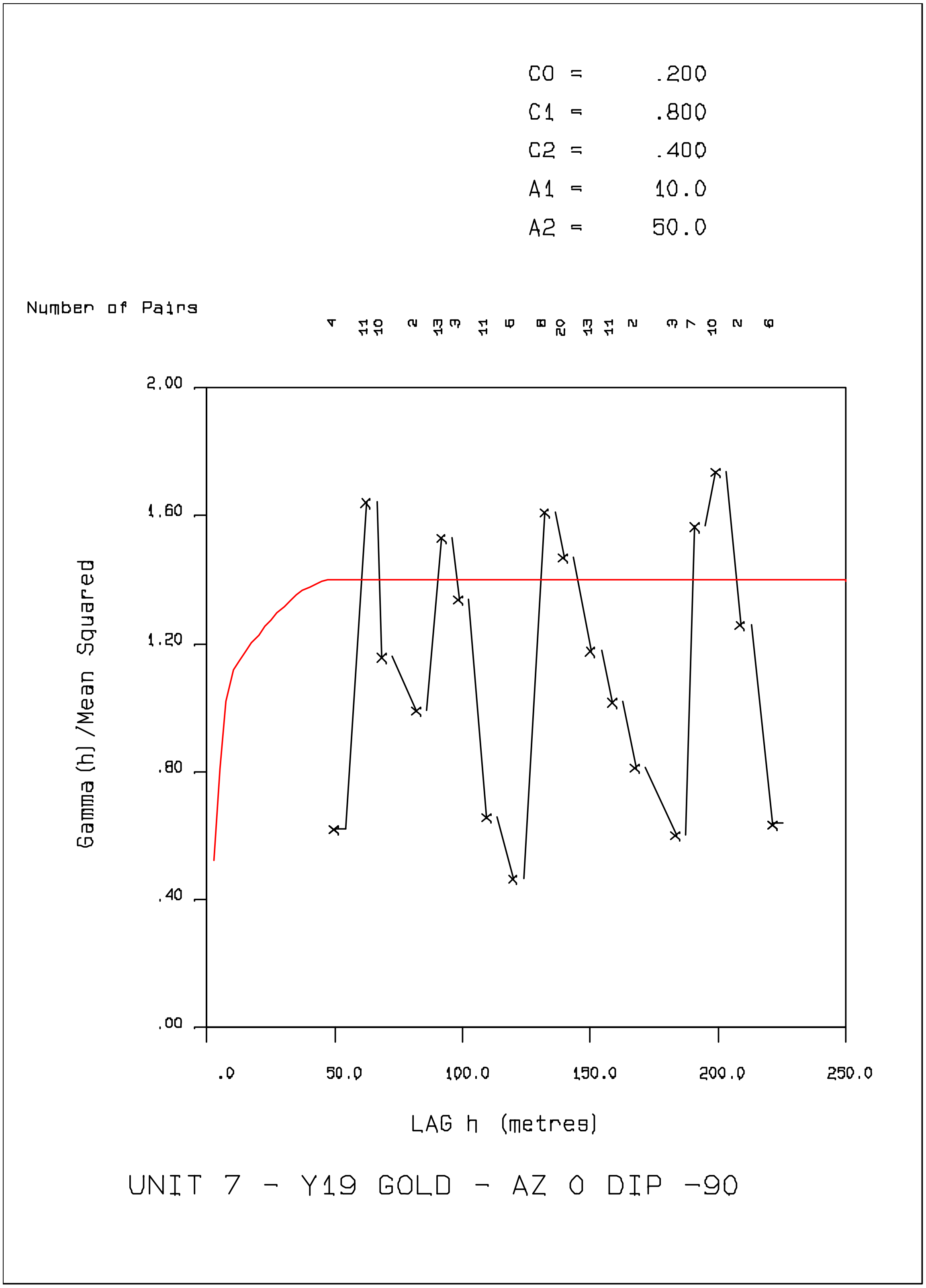
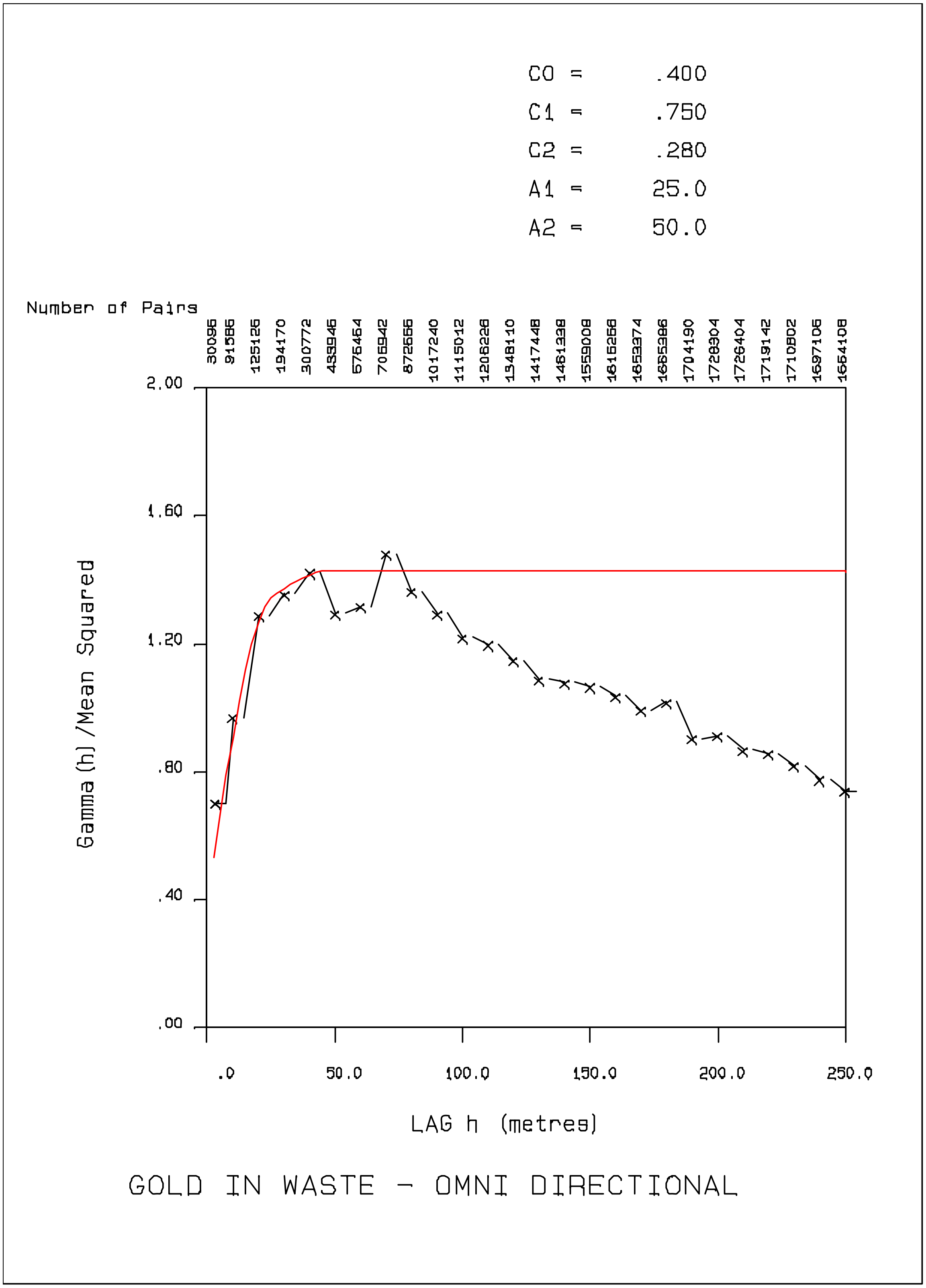
C1 = .800
C2 = .400
A1 = 12.0
A2 = 40.0
Number of Pairs
12 13 5 3 14 8 19 26 35 15 29 14 31 28 23 19 9 3 15 3
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
40.0
80.0
120.0
160.0
200.0
LAG h (metres)
UNIT 7 - Y19 GOLD - AZ 0 DIP 0



C0 = .200
C1 = .800
C2 = .400
A1 = 2.0
A2 = 8.0
Number of Pairs
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
UNIT 7 - Y19 GOLD - AZ 270 DIP 0


C0 = .200
C1 = .800
C2 = .400
A1 = 10.0
A2 = 50.0
Number of Pairs
4 11 10 2 13 3 11 5 8 20 13 11 2 3 7 10 2 6
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
UNIT 7 - Y19 GOLD - AZ 0 DIP -90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C1 = .750
C2 = .280
A1 = 25.0
A2 = 50.0
Number of Pairs
30096
91566
125126
194170
300772
433946
575464
705942
872666
1017240
1115012
1206226
1348110
1417448
1461338
1559008
1615256
1653374
1665386
1704190
1728304
1726404
1719142
1710802
1697106
1664108
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
GOLD IN WASTE - OMNI DIRECTIONAL

APPENDIX B LISTING OF SPECIFIC GRAVITY DETERMINATIONS

HOLE	FROM	TO	GEOL	AU	SG
PT30A	271.70	273.22		27.430	2.88
PT9601	54.07	55.66		12.820	2.84
PT9601	99.40	99.64		7.100	2.83
PT9601	99.64	100.40		7.100	2.83
PT9602	72.85	73.67		50.740	2.84
PT9603	28.29	29.57		17.620	2.91
PT9603	73.97	74.98		11.310	2.87
PT9604	84.09	85.28		17.860	2.84
PT9604	85.28	86.00		8.020	2.92
PT9604	86.00	86.56		8.020	2.92
PT9604	91.62	92.66		10.080	2.89
PT9604	97.05	97.78		15.980	2.91
PT9604	97.78	99.06		8.500	2.94
PT9604	99.06	99.30		8.500	2.94
PT9605	44.47	44.99		11.070	2.93
PT9606	28.50	29.57		9.810	2.82
PT9606	62.33	62.85		8.910	2.80
PT9607	11.70	13.11		11.520	2.88
PT9609	56.85	57.91		49.030	2.87
PT9610	60.90	62.42		14.060	2.83
PT9610	82.45	83.36		8.060	2.90
PT9611	72.54	72.80		7.170	2.88
PT9611	72.80	73.03		7.170	2.88
PT9613	51.79	53.31		8.980	2.90
PT9613	56.36	57.88		9.770	2.91
PT9613	59.41	60.24		7.820	2.98
PT9613	60.24	60.72		7.820	2.98
PT9613	83.79	84.49		44.090	2.93
PT9613	84.49	85.04		44.090	2.93
PT9615	115.85	116.28		13.710	2.91
PT9615	116.28	117.68		10.800	2.89
PT9616	112.29	112.78		8.400	2.81
PT9616	112.78	113.84		8.540	2.83
PT9617	57.24	58.12		12.030	2.98
PT9617	58.12	58.58		12.030	2.98
PT9618	124.54	125.43		13.710	2.90
PT9618	125.43	126.07		13.710	2.90
PT9618	126.07	127.59		10.080	2.90
PT9619	126.46	127.41		20.670	2.85
PT9622	119.54	121.07		23.830	2.85
PT9622	139.29	140.82		7.130	3.96
PT9624	167.58	168.92		10.830	2.90
PT9624	168.92	169.55		36.140	2.93
PT9624	169.55	170.23		36.140	2.93
PT9624	170.23	171.57		14.190	2.90
PT9704	176.05	177.52		27.670	2.88
PT9709	52.33	53.10		9.810	2.87
PT9711	150.02	151.55		19.170	3.00
PT9732	212.54	214.06		10.250	2.81
PT9733	0.76	2.29		11.690	2.94
PT9733	257.83	259.35	7.0	8.980	2.84
PT9734	0.00	1.52		11.250	2.84

PT9734	2.93	4.06		16.940	2.88
PT9734	4.06	4.21		16.940	2.88
PT9734	5.73	6.19		13.270	2.92
PT9734	92.57	94.09		8.670	2.89
PT9734	94.09	94.74		24.100	2.85
PT9734	94.74	95.62		24.100	2.85
PT9735	1.28	3.14		11.450	2.81
PT9735	127.25	128.93		18.380	2.85
PT9735	131.98	133.50		6.960	2.82
PT9736	226.86	228.39		7.200	2.86
PT9743	5.55	7.07		10.350	2.82
PT9743	8.60	9.51		15.460	2.80
PT9743	161.27	162.25		17.970	2.88
PT9743	162.25	162.55		17.970	2.88
PT9743	164.44	165.14		12.510	2.88
PT9743	166.27	166.88		16.940	2.87
PT9744	249.88	250.45		29.730	2.79
PT9744	250.45	251.49		20.980	2.73
PT9744	257.77	259.35	6.0	7.030	2.76
PT9744	259.35	259.69	6.0	19.920	2.88
PT9744	259.69	260.79	6.0	19.920	2.88
PT9744	262.19	263.41	6.0	15.870	2.84
PT9744	264.90	266.27	6.0	22.110	2.82
PT9744	266.61	268.10	6.0	25.270	2.84
PT9744	268.10	268.65	6.0	7.440	2.78
PT9744	269.32	270.45	6.0	17.180	2.79
PT9744	270.45	271.88	6.0	28.830	2.79
PT9744	271.88	273.34	6.0	10.290	2.79
PT9744	273.34	274.93	6.0	24.450	2.83
PT9744	274.93	276.45	6.0	29.790	2.79
PT9744	276.45	277.98	6.0	17.350	2.84
PT9744	277.98	279.11	6.0	20.230	2.82
PT9744	279.11	280.11	6.0	8.300	2.74
PT9744	280.11	281.09	6.0	21.290	2.80
PT9744	281.09	282.82	6.0	43.340	2.82

APPENDIX C DIAMOND DRILL HOLES (1988 TO 2006)

Hole ID	Easting	Northing	Elevation	Length	Dip	Azimuth
03-P01	1569.72	1021.08	17.45	365.46	-70	345
03-P02	1513.33	1021.08	17.98	426.72	-70	330
03-P03	1591.06	1255.78	31.82	768.71	-60	260
04-1707E1	1706.88	873.25	16.55	259.08	-75	359
04-1707E2	1707.44	873.25	16.55	280.11	-83	359
04-1737E1	1737.63	873.16	16.79	287.43	-83	359
04-1737E2	1737.33	845.82	16.72	289.26	-83	320
04-300SW1	1842.61	902.79	17.07	224.03	-66	320
04-330SW2	1789.62	904.06	16.46	237.44	-84	324
04-360SW1	1777.39	884.53	16.48	237.44	-75	318
04-360SW2	1777.39	884.53	16.48	243.84	-83	318
05-1676E1	1676.40	872.49	16.76	210.92	-71	359
05-1676E2	1676.40	872.49	16.76	240.49	-78	359
05-1676E3	1676.40	872.49	16.76	94.49	-83	359
05-300SW4	1875.64	860.29	17.30	289.86	-78	322
05-300SW5	1877.20	858.36	17.32	315.16	-85	318
05-300SW6	1911.06	816.86	17.52	347.17	-85	320
05-330SW3	1835.64	860.95	16.76	277.98	-77	320
05-330SW4	1835.69	860.88	17.10	298.40	-83	320
05-330SW5	1853.22	839.92	16.76	315.16	-83	320
06-1585E8	1560.00	725.00	16.76	396.00	-74	0
06-1585E9	1560.00	725.00	16.76	483.00	-76	353
06-1615E1	1605.07	913.78	15.48	150.00	-63	0
06-1615E2	1605.07	913.78	15.48	165.00	-77	360
06-1615E4	1621.13	827.57	15.73	252.00	-70	349
06-1615E5	1621.13	827.57	15.73	249.00	-76	350
06-1615E6	1621.13	827.57	15.73	274.00	-79	349
06-1615E7.5	1615.44	750.77	15.77	372.00	-74	355
06-1615E8	1615.68	791.00	15.87	377.50	-82	355
06-1615E9	1615.44	750.77	15.77	471.00	-84	355
06-1646E1	1646.51	916.90	15.82	176.50	-75	357
06-1646E2A	1646.17	914.73	15.99	186.00	-83	356
06-1646E4	1647.00	840.50	16.76	231.00	-72	356
06-1646E5	1645.66	825.32	15.52	269.00	-77	355
06-1646E6	1645.66	825.32	15.52	273.00	-79	356
06-1646E7	1645.85	797.57	15.49	317.00	-79	350
06-1646E7A	1645.90	796.00	15.50	303.00	-80	355
06-1646E8	1645.84	795.00	15.57	369.00	-84	355
06-1646E9	1645.91	754.60	15.45	407.00	-82	356
06-1676E4	1677.41	826.88	15.89	297.00	-77	356
06-1676E5B	1673.93	807.58	15.76	285.00	-76	356
06-1676E6	1675.06	785.32	15.65	309.00	-80	356
06-1676E7	1675.06	785.32	15.65	344.00	-85	356

Hole ID	Easting	Northing	Elevation	Length	Dip	Azimuth
06-1676E8A	1676.40	697.00	16.76	405.00	-79	358
06-1685DE1	1681.50	570.00	16.76	642.00	-76	352
06-1707DE1	1711.50	615.50	16.76	462.00	-77	352
06-1707E3	1706.73	823.15	15.60	285.00	-78	356
06-1707E4	1706.73	823.15	15.60	363.00	-84	355
06-1707E6	1711.67	776.02	15.74	129.00	-84	343
06-1707E6A	1711.67	776.02	15.74	360.00	-84	355
06-1707E7	1704.68	729.96	15.47	366.00	-82	356
06-1737E3	1736.73	823.38	15.95	342.00	-82	356
06-1737E4	1736.61	772.69	15.82	318.00	-80	355
06-1737E5	1736.61	772.69	15.82	336.00	-85	355
06-1737E6	1730.37	722.08	15.64	390.00	-82	358
06-1768DE1	1767.84	575.00	16.76	568.00	-76	353
06-1768DE2	1767.84	575.00	16.76	574.00	-88	353
06-1768E1A	1765.16	800.61	15.79	294.60	-75	356
06-1768E2	1765.16	800.61	15.79	291.00	-82	356
06-1768E3	1767.81	744.20	15.92	345.00	-82	356
06-1768E4A	1768.59	689.41	15.78	444.00	-84	360
06-1813E1	1813.23	739.20	16.10	315.00	-73	355
06-1813E2	1813.23	739.20	16.10	351.00	-82	356
06-1813E3	1813.78	682.54	16.25	417.00	-82	355
06-1859E1	1859.28	720.00	16.76	165.00	-73	355
06-1859E1A	1859.28	720.00	16.76	381.00	-81	355
06-1859E2	1859.28	685.00	16.76	461.00	-85	355
06-195SW1	2027.16	841.38	16.59	420.00	-71	316
06-195SW2	2027.16	841.38	16.59	336.00	-84	316
06-195SW3	2059.58	799.53	16.44	363.00	-82	316
06-240SW3	1889.82	940.39	16.61	309.00	-82	316
06-240SW4	1923.13	898.68	16.95	341.00	-77	316
06-240SW4.5	1938.62	880.48	16.81	330.00	-78	316
06-240SW5	1923.13	898.68	16.95	114.00	-83	317
06-240SW5A	1923.13	898.68	16.95	318.00	-83	317
06-240SW7	1950.04	860.89	16.65	401.50	-84	316
06-240SW8	1991.88	813.61	15.95	432.00	-81	317
06-240SW9	2026.26	770.78	15.93	371.00	-82	317
06-270SW2	1880.17	902.95	16.56	240.00	-64	316
06-270SW3	1880.17	902.95	16.56	279.00	-78	318
06-270SW4	1898.98	880.21	16.64	384.00	-77	318
06-270SW5	1898.98	880.21	16.64	378.00	-85	316
06-270SW7	1931.30	836.25	16.61	413.50	-84	316
06-270SW8	1978.38	777.34	16.40	420.00	-81	316
06-300SW4.5	1919.45	806.49	16.58	299.00	-70	316
06-300SW7	1937.96	783.08	16.86	427.50	-83	315
06-300SW8	1975.97	736.58	15.94	365.50	-80	316
06-300SW9A	1978.91	736.12	16.09	397.00	-85	318
06-330SW10	1998.39	664.71	16.30	389.00	-73	316

Hole ID	Easting	Northing	Elevation	Length	Dip	Azimuth
06-330SW11	1998.39	664.71	16.30	405.00	-82	316
06-330SW6	1866.83	819.57	16.31	357.00	-84	316
06-330SW9	1998.39	664.71	16.30	351.00	-66	315
L52	1465.17	1330.15	41.45	248.41	-25	85
L55	1510.89	1361.54	41.45	274.32	-17	40
L56	1498.70	1313.69	41.45	161.85	-25	92
L57	1464.26	1330.45	41.45	215.49	-25	51
L58	1473.40	1278.79	41.45	189.28	-25	112
L59	1477.67	1279.25	41.45	343.20	-25	95
L62	1463.04	1233.83	41.45	306.32	-25	149
L84	1342.64	1237.79	-186.84	229.21	-3	248
P8918A	1819.79	889.97	21.49	367.89	-70	315
P91C01	1888.57	762.30	16.86	322.17	-65	338
P91C02	1888.57	762.30	16.86	331.62	-77	338
P91C03	1937.98	805.62	16.70	320.04	-75	339
P91C04	1937.98	805.62	16.70	324.92	-87	339
P91C05	1827.28	942.78	16.82	199.19	-73	339
P91C06	1672.83	818.27	15.97	263.96	-71	339
P91C07	1907.53	704.18	16.79	320.65	-75	335
P91Y01	1767.03	1369.25	22.25	330.10	-66	270
P91Y02	1769.59	1336.46	17.64	305.10	-64	266
P91Y03	1705.02	1095.33	18.29	274.93	-49	268
P91Y04B	1845.38	1336.03	17.65	444.09	-68	274
P92C08	1810.12	984.99	15.70	199.03	-71	342
P92C09	1889.24	964.20	16.73	210.31	-72	340
P92C10	1917.31	894.37	17.19	243.84	-77	340
P92C11	1978.82	925.86	17.25	251.16	-72	337
P92C12	1870.25	819.00	16.46	282.55	-72	337
P92C13	1635.16	860.39	16.76	223.72	-70	345
P92C14	1614.98	823.90	17.37	262.13	-70	343
P92C15	1702.92	823.57	16.46	258.17	-72	338
P92C16	1816.00	780.29	17.37	254.20	-72	338
P92C17	1730.41	865.11	17.37	264.26	-76	339
P92C18	1606.63	792.60	17.37	350.22	-70	337
P92C19	1694.87	761.73	17.37	381.30	-70	350
P92C20	1549.57	832.74	17.10	267.31	-70	350
P92C20A	1549.57	832.74	17.10	273.71	-70	350
P92C21	1534.49	871.36	17.59	213.97	-71	358
P92C22	1624.58	743.71	17.68	413.61	-71	336
P92C23	1518.98	873.06	17.37	274.62	-72	337
P92C24	1798.02	918.21	16.43	149.66	-61	331
P92Y05	1769.67	1335.82	17.80	346.56	-70	259
P92Y08	1787.35	1367.03	19.20	326.96	-67	274
P92Y09	1745.28	1336.55	20.12	248.11	-61	269
P93C25	1737.36	911.35	16.46	102.41	-66	340
P93C26	1858.98	1036.32	17.07	144.17	-70	295

Hole ID	Easting	Northing	Elevation	Length	Dip	Azimuth
P93C27	1860.13	1034.89	16.79	166.73	-85	295
P93C28	1882.41	1050.04	16.40	165.51	-75	310
P93C29	1906.65	1088.81	16.73	179.83	-75	280
P93C30	1928.77	1144.68	16.46	168.25	-75	280
P93C31	1815.61	901.17	16.76	215.49	-74	335
P94C32	1908.04	962.37	17.07	249.02	-72	303
P94C33	1908.37	961.28	17.05	279.81	-86	284
P94C34	1937.31	1015.90	16.46	258.17	-72	287
P94C35	1937.31	1015.90	16.46	339.55	-83	288
P94C36	1957.12	891.54	16.46	371.86	-80	288
P94C37	1883.66	789.74	16.46	279.50	-74	327
P94C38	1699.87	806.20	16.46	287.73	-82	348
P94C39	1769.36	768.10	16.46	290.17	-77	351
P94N01	1469.75	2124.76	248.72	84.73	-55	85
P94N02	1469.75	2124.76	248.72	84.73	-85	85
P94N03	1488.95	2194.26	251.16	60.35	-55	85
P94N04	1488.95	2194.26	251.16	106.07	-90	0
P94N05	1495.96	2258.26	260.30	118.26	-56	85
P94N06	1495.96	2258.26	260.30	124.36	-87	85
P94N07	1488.34	2332.63	289.26	224.94	-55	85
P94N08	1488.34	2332.63	289.26	273.71	-85	85
P94N09	1490.17	2438.10	300.84	264.57	-50	70
P94N10	1481.33	2438.10	300.84	96.93	-85	70
P94N11	1492.30	2499.36	306.32	75.59	-50	70
P94N12	1492.30	2499.36	309.37	71.02	-85	70
P94N13	1482.85	2384.76	294.13	66.45	-50	70
P94N14	1482.85	2384.76	294.13	66.45	-85	70
P94N15	1187.20	1822.70	236.22	51.21	-60	70
P94N16	1187.20	1822.70	236.22	57.30	-60	100
P94Y10	1663.90	1203.96	17.07	236.83	-63	266
P94Y11	1813.56	1280.16	16.46	251.76	-55	270
P94Y12	1813.56	1280.16	16.46	276.15	-63	265
P94Y13	1929.69	1191.46	16.46	371.86	-62	265
P94Y14	1929.69	1191.46	16.46	400.81	-55	262
P95C40	1965.96	575.77	17.07	786.69	-83	355
P95C41	2029.97	553.21	17.07	769.01	-80	360
P95C42	1861.72	571.50	17.07	673.00	-78	0
P95C43	1861.72	571.50	17.07	751.94	-80	340
P95C44	1920.24	525.78	17.07	792.48	-83	340
P95C44A	1920.24	525.78	17.07	749.81	-83	340
P95N17	1347.22	2122.93	312.12	264.57	-70	85
P95N18	1370.69	2183.89	332.23	270.66	-72	85
P95N19	1350.26	2243.33	354.79	315.47	-68	85
P95N20	1348.74	2302.76	354.48	335.28	-70	85
P95N21	1348.13	2329.59	363.32	215.80	-70	70
P95N22	1353.92	2387.80	375.82	227.99	-72	70

Hole ID	Easting	Northing	Elevation	Length	Dip	Azimuth
P95N23	1338.07	2449.07	393.80	261.52	-68	70
P95N24	1336.24	2449.07	393.80	294.74	-85	70
P95N25	1502.66	2804.16	347.47	255.42	-55	100
P95N26	1325.88	2804.16	480.06	273.71	-80	100
P95N27	1325.88	2804.16	480.06	334.67	-80	80
P-9612	1289.97	1259.92	77.21	174.65	-35	119
P9705A	1055.52	1545.03	80.70	231.34	-21	52
PC25A	1738.27	909.83	16.46	159.41	-68	336
PC8801	1554.16	1178.32	30.51	106.68	-50	310
PC8802	1554.16	1178.32	30.51	38.71	-90	0
PC8803	1496.69	1146.74	41.14	133.20	-60	310
PC8804	1496.21	1141.91	41.12	119.48	-60	280
PC8805	1614.15	1363.59	51.28	149.35	-45	271
PC8806	1614.68	1363.74	51.26	189.28	-60	270
PC8807	1614.14	1364.20	51.39	180.14	-48	290
PC8808	1614.26	1365.03	51.27	110.95	-48	310
PC8901	1573.08	1149.96	23.76	142.34	-90	0
PC8902	1594.63	1088.77	19.11	128.32	-86	275
PC8903	1593.56	1088.51	19.14	167.64	-55	265
PC8904	1546.21	1068.50	19.83	50.90	-52	90
PC8904A	1546.21	1068.50	19.83	230.73	-52	90
PC8905	1546.73	1039.04	19.79	167.94	-60	90
PC8906	1705.42	1363.04	22.96	220.98	-55	268
PC8907	1704.52	1363.20	23.11	236.83	-70	290
PC8908	1704.52	1363.20	23.11	243.23	-50	290
PC8909	1639.71	1341.13	38.47	185.32	-45	270
PC8910	1639.14	1341.19	38.41	126.34	-59	270
PC8911	1639.77	1342.07	38.50	187.45	-53	310
PC8912	1726.56	1440.13	23.96	307.85	-45	270
PC8913	1727.76	1440.48	23.96	244.75	-60	270
PC8914	1726.95	1439.69	23.95	253.59	-56	256
PC8915	1726.18	1163.72	18.70	413.31	-65	270
PC8916	1762.08	1377.37	22.01	233.78	-55	270
PC8917	1676.68	1157.69	18.70	171.30	-45	270
PC8918	1819.79	889.97	21.49	191.11	-70	315
PC8919	1777.26	1103.56	20.20	219.15	-60	270
PC9001	1820.17	822.27	21.49	400.51	-70	340
PC9002	1838.20	767.83	16.61	388.92	-70	340
PC9003	1746.57	837.70	21.34	305.10	-70	340
PC9004	1853.47	897.63	21.34	305.71	-70	340
PC9004A	1853.47	897.63	21.34	108.91	-70	340
PC9005	1675.88	852.91	21.49	265.48	-60	340
PC9006	1762.29	792.22	21.49	404.16	-70	340
PC9007	1605.02	880.79	21.34	242.62	-68	340
PC9008	1778.45	746.71	21.58	441.05	-70	330
PT23A	1317.96	1244.50	43.28	206.65	-32	195

Hole ID	Easting	Northing	Elevation	Length	Dip	Azimuth
PT29A	1317.96	1244.50	43.28	325.53	-31	134
PT30A	1317.96	1244.50	43.28	279.81	-15	90
PT31A	1334.41	1456.94	43.74	406.30	0	20
PT37B	1595.05	1102.00	-45.96	179.53	-52	45
PT38A	1595.32	1097.49	-46.09	260.30	-35	124
PT9601	1444.26	1379.06	79.14	106.07	-36	79
PT9602	1441.21	1379.06	79.14	120.40	-51	79
PT9603	1441.22	1378.72	79.14	102.11	-36	96
PT9604	1441.22	1378.72	79.14	130.76	-51	96
PT9605	1441.09	1379.36	79.14	101.80	-49	68
PT9606	1441.33	1378.29	79.14	115.82	-38	110
PT9607	1436.92	1380.33	79.14	43.59	22	291
PT9608	1436.92	1380.62	79.14	17.98	22	299
PT9609	1437.02	1380.72	79.14	119.48	22	304
PT9610	1437.02	1380.73	79.14	119.48	-5	304
PT9611	1436.91	1380.62	79.14	125.88	-22	299
PT9613	1289.76	1259.71	77.21	149.05	-41	134
PT9614	1289.43	1258.88	77.21	121.01	-52	164
PT9615	1289.76	1259.71	77.21	147.52	-52	134
PT9616	1289.97	1259.92	77.21	176.78	-46	119
PT9617	1288.85	1263.64	77.21	152.10	0	15
PT9618	1055.52	1545.03	80.70	146.00	18	88
PT9619	1055.52	1545.03	80.70	127.41	18	108
PT9620	1055.52	1545.03	80.70	173.74	-10	118
PT9621	1055.52	1545.03	80.70	141.43	-9	80
PT9622	1055.52	1545.03	80.70	152.10	22	108
PT9623	1055.52	1545.03	80.70	167.34	-14	110
PT9624	1055.52	1545.03	80.70	272.19	-10	52
PT9701	1055.52	1545.03	80.70	9.75	-5	52
PT9702	1055.52	1545.03	80.70	287.73	-7	52
PT9703	1055.52	1545.03	80.70	220.07	-13	60
PT9704	1055.52	1545.03	80.70	205.44	-10	60
PT9706	1055.52	1545.03	80.70	255.42	-10	49
PT9707	1055.52	1545.03	80.70	240.18	10	52
PT9708	1049.12	1542.59	80.70	144.48	0	242
PT9709	1217.37	1301.04	77.11	69.80	-22	0
PT9710	1217.98	1301.19	77.11	69.49	-22	340
PT9711	1218.90	1301.04	77.11	163.37	-16	78
PT9712	1218.90	1301.04	77.11	171.30	4	60
PT9713	1222.10	1299.97	78.64	281.33	-15	90
PT9714	1219.20	1297.84	78.33	47.61	-42	0
PT9715	1218.90	1301.04	78.33	59.74	-46	330
PT9716	1217.07	1296.62	78.64	135.33	-15	270
PT9717	1217.07	1296.62	78.33	154.84	-30	270
PT9718	1217.07	1296.62	78.64	118.26	-18	249
PT9719	1290.83	1417.62	43.28	112.47	-5	90

Hole ID	Easting	Northing	Elevation	Length	Dip	Azimuth
PT9720	1290.83	1417.62	43.28	133.81	-35	90
PT9721	1290.52	1417.62	43.28	153.92	-44	71
PT9722	1289.30	1418.23	44.04	219.15	-5	270
PT9723	1317.96	1244.50	43.28	17.98	-32	195
PT9724	1317.96	1244.50	43.28	192.94	-30	217
PT9725	1317.96	1244.50	43.28	244.14	-47	218
PT9726	1317.96	1244.50	43.28	124.66	-10	334
PT9727	1317.96	1244.50	43.28	110.34	-12	351
PT9728	1317.96	1244.50	43.28	228.60	-31	351
PT9731	1334.11	1456.94	43.74	19.20	0	20
PT9732	1594.99	1102.89	-45.90	229.51	-22	27
PT9733	1594.65	1102.28	-46.45	272.49	-52	24
PT9734	1595.23	1101.76	-45.78	155.14	-32	50
PT9735	1594.96	1102.83	-45.54	198.73	-10	26
PT9736	1594.87	1102.95	-46.02	237.13	-29	24
PT9738	1595.35	1097.49	-46.09	14.33	-35	124
PT9739	1594.96	1097.43	-46.39	204.52	-29	136
PT9740	1594.53	1097.25	-46.18	180.75	-34	146
PT9741	1594.93	1097.74	-46.60	155.14	-42	130
PT9742	1593.46	1096.88	-46.54	247.65	-37	188
PT9743	1592.92	1096.85	-46.54	246.58	-36	210
PT9744	1594.71	1097.40	-46.39	290.17	-38	134
PT9744A	1594.71	1097.40	-46.39	136.55	-36	142
PT9745	1593.10	1096.85	-46.18	219.46	-26	193
PY06B	1769.36	1406.96	19.81	295.05	-60	273
PY07A	1772.11	1406.96	19.81	387.10	-67	275
299 Drill Holes			**Total Length =**	**77,642.3 m**		