

**REPORT ON THE
2003 TO 2005
EXPLORATION PROGRAM
ON THE NEW POLARIS
MINE SITE**

North Western British Columbia

NTS: 104 K 12

Latitude: 58°42'N

Longitude: 133°37'W

Atlin Mining Division

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

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Item 2: Table of Contents

	Table of Contents	Page 2
	List of Figures	Page 3
	List of Tables	Page 3
	Appendices	Page 3
Item 3	Summary	Page 4
Item 4	Introduction	Page 5
Item 5	Reliance on Other Experts	Page 5
Item 6	Property Description and Location	Page 5
Item 7	Accessibility, Climate, Local Resources, Infrastructure and Physiograph	Page 10
Item 8	History	Page 11
Item 9	Geological Setting	Page 16
Item 10	Deposit Types	Page 19
Item 11	Mineralization	Page 20
Item 12	Exploration	Page 21
Item 13	Diamond Drilling Program	Page 21
Item 14	Sample Method and Approach	Page 36
Item 15	Sample Preparation, Analyses and Security	Page 37
Item 16	Data Verification	Page 42
Item 17	Adjacent Properties	Page 43
Item 18	Mineral Processing and Metallurgical Testing	Page 43
Item 19	Mineral Resource and Reserve Estimate	Page 45
Item 20	Other Relevant Data and Information	Page 45
Item 21	Interpretation and Conclusions	Page 45
Item 22	Recommendations	Page 45
Item 23	References	Page 46
Item 24	Date	Page 47
	Certificate of Author	Page 47

Illustrations

Figure	Title	Scale
1	Property Location Map	Page 6
2	Claim Map	Page 8
3	Principle Vein Location Map	Page 9
4	Regional Geology Map	Page 17
5	Property Geology Map	Page 18
6	Three Dimensional View of Veins	Page 21
7	Collar Locations	Page 31
8	Inclined Section "C" Vein System	Page 32
9	Cross Section line 2000SW	Page 33
10	Cross Section line 2100SW	Page 34
11	Cross Section line 5500E	Page 35

Tables

No.	Title	
1	List of Claims	Page 7
2	Summary of Exploration Drilling to 2005	Page 12
3	Diamond Drill Holes (1988 to 2005)	Page 22
4	Assay Composites C vein system	Page 27

Appendices

No.	Title	
1	Drill Sections	Page 50
2	Metallurgical Results	Page 60
3	Drill Hole Assay results	Page 194

Item 3: 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 terminated five miles 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 gold 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 northwest striking and southwest dipping, the “Y” veins north striking and dipping steeply east and finally the “C” veins east-west striking and dipping to the south to southeast at 65° 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. The gold values in the veins are remarkable in their continuity and uniformity.

Several consulting groups have estimated a mineral inventory for the property. Gary Giroux of Montgomery Consultants completed the last in 1995 in which he estimated 450,000 tons grading 0.365 oz/ton Au in a probable resource and 2,509,000 tons grading 0.365 oz/ton Au as a possible resource. Under the current guidelines of 43-101, this resource would be categorized as 450,000 tons grading 0.365 oz/ton Au as an indicated resource and 2,509,000 tons grading 0.365 oz/ton Au as an inferred resource. This resource 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. These estimates therefore should not be relied upon.

Canarc carried out initial phases of infill drilling in 2003 through 2005 with in the “C” vein system. The purpose of this drilling was to better define the continuity and grade of the vein systems prior to committing to a major infill drilling campaign.

The 2003 and 2004 drilling confirmed the continuity of both the “C” vein system and provides support and justification to complete the in-fill drilling of the “C” vein system. The objective of the proposed program is to define the grade and tonnes present in the upper portion of the “C” vein. To this end a program of 20,000 metres of drilling in approximately 65 holes is recommended the cost is estimated at \$3.18 million.

Item 4: Introduction

The Author was commissioned in September 2006 by Bradford J. Cooke, Chairman of Canarc Resource Corp to provide a report summarizing the results of the June 2002 to December 31, 2005 exploration on the New Polaris Property. The report will be used to file a revised Annual Information Report (AIF) as required by the provincial securities regulation. The author is familiar with the project having made site visits to the property in May, June, July and August 2006. In total, the Author has spent 7 days on site. During the time on the Polaris property, the Author inspected the underground workings and confirmed the nature of the mineralization. He examined drill core from the 2003 to 2005 drilling. He compared his observations with that of the drill logs and found no discrepancies. In addition, the author has reviews the drill hole database, drill hole information on sections and plans.

The data from this report is based upon the data and reports provided by Canarc Resource Corp. to the Author as well as observations made while at the New Polaris Property. The main sources are internal files and memos primarily assembled and authored by James Moors PGeo., a summary report on work carried out prior to 2002 by Godfrey Walton PGeo. and a reviewed of data on site as well as examining the sections, level plans and digital databases in Canarc Resource Corp.'s office.

The author has not verified the old resource estimated.

Item 5: Reliance on Other Experts

The metallurgical test work was carried out by RDi of Denver Colorado. RDi is reputable firm and the Author has full confidence in the quality of work, results, conclusions and recommendations.

Item 6: Property Description and Location

The New Polaris (formerly the Polaris-Taku mine) property consisting of a group of 61 contiguous crown grants and one modified grid claim totaling 2,956 acres located 60 miles south of Atlin, BC and 40 miles northeast of Juneau, Alaska. Located at approximately 133°37'W Longitude and 58°42'N Latitude, the deposit lies in close proximity to the "Tulesquah Chief" property of Redcorp on the eastern flank of the

Tulsequah River Valley (Figure 1).

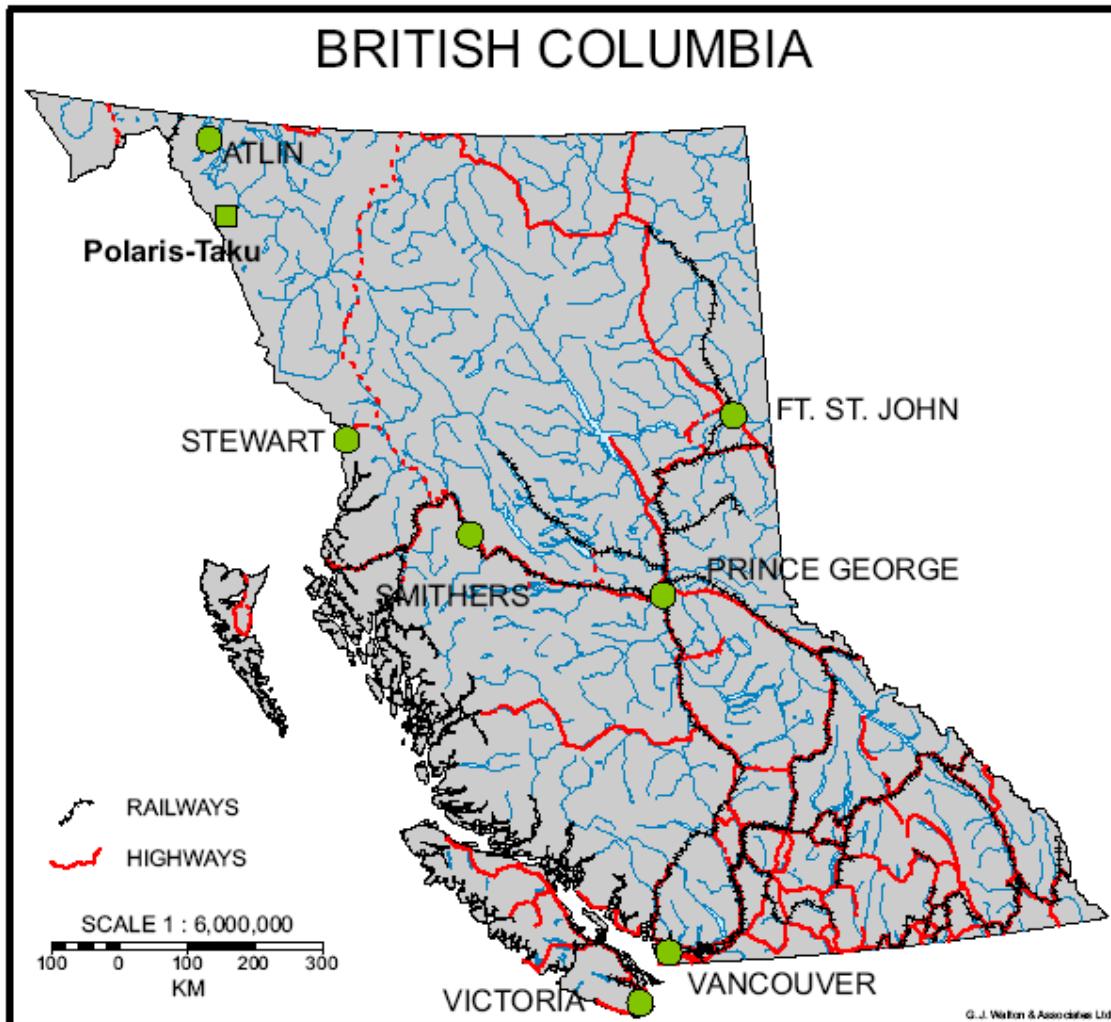


Figure 1 - Location Map

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 2 while Table 1 summarizes the claims shown on Figure 2.

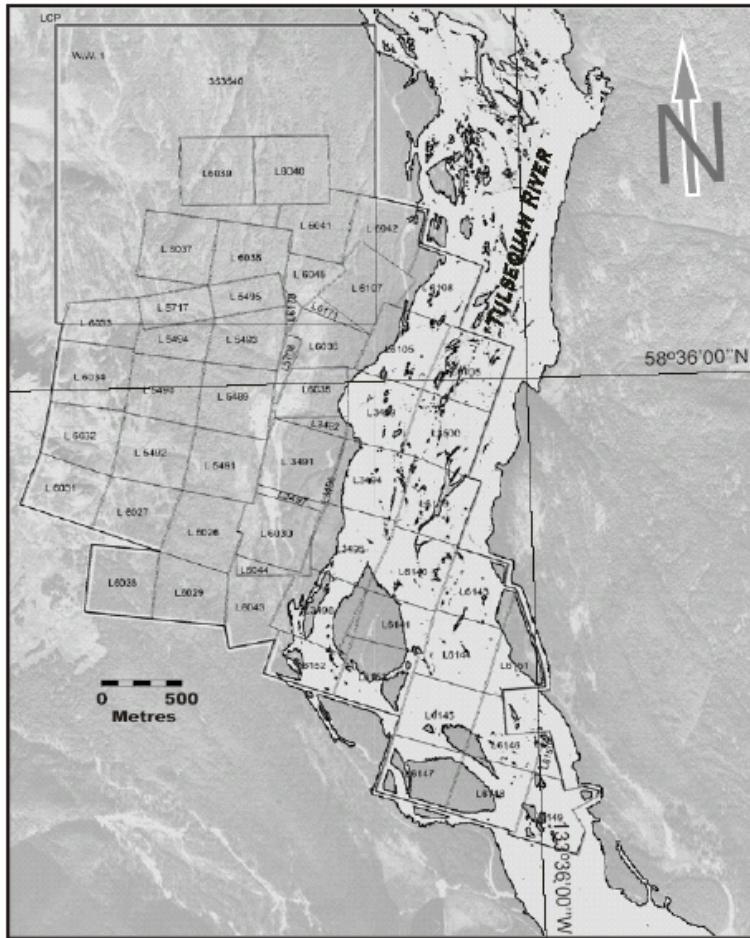
With the exception of the W.W.1 claim, the claims are crown granted and are kept in good standing through annual tax payments. The W.W.1 is a modified grid claim. The claim has sufficient work filed on it to keep it in good standing until February 4, 2015. The crown granted claims were legally surveyed in 1937. The mineralized areas are shown on Figure 3 and 5, which shows the geology of the property on the mineral showings. Notice of work for the program will be required before work on the property can start. 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.

Table 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	4223	Snow No. 2	3495	5088
Polaris No. 3	6141	4223	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	5494
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			

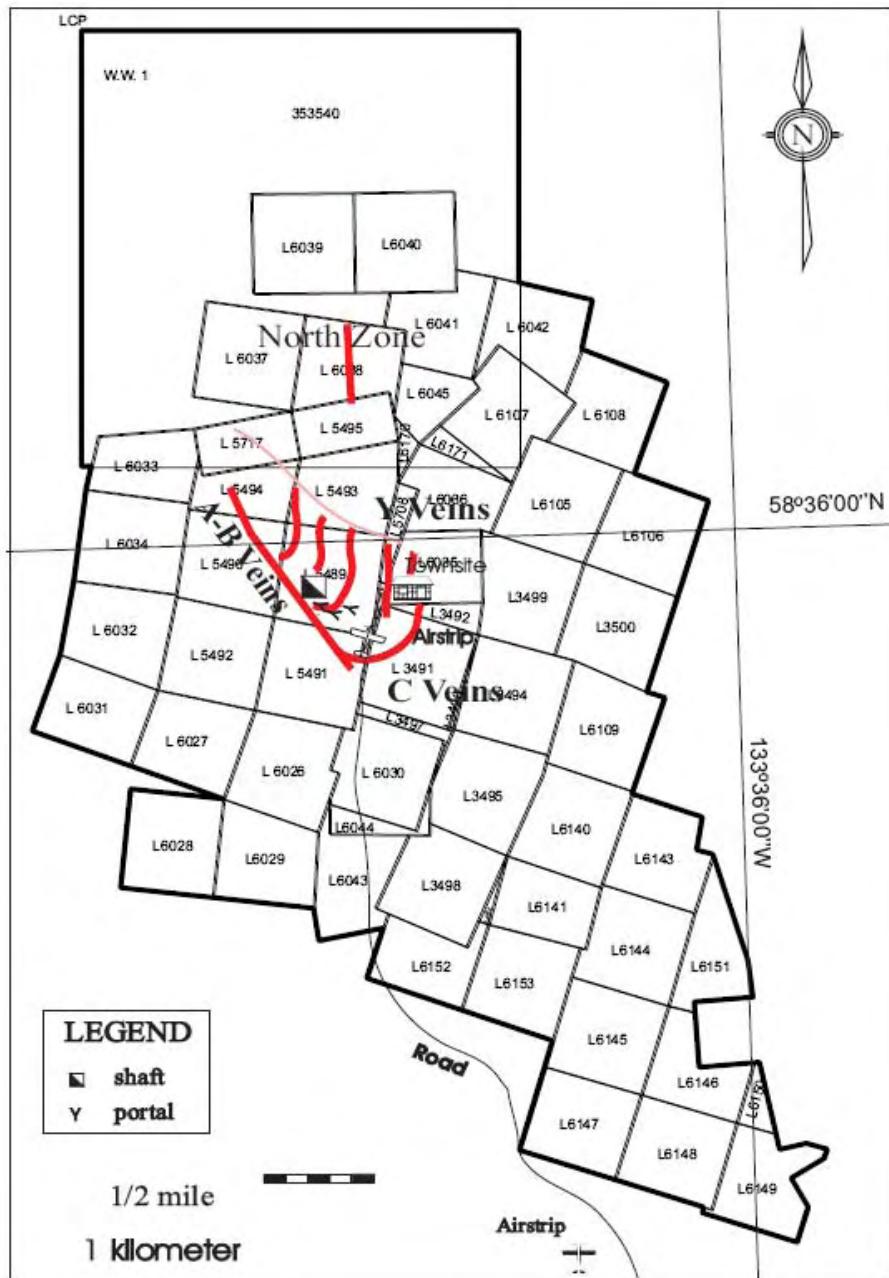
W.W.1 Tenure No. 353540 Issue date February 4, 1997. Expiry date: February 4, 2015.



Canarc Resource Corp
NTS 104K/12
New Polaris Project
Atlin Mining Division
Claim Map

Figure 2 Claim Location Map

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



New Polaris Property
Claim Map with Principle Vein Locations

Mining of the AB Vein system and to a lesser extent on the Y and C veins was carried out during the 1930s to early 1950s. Much of the former infrastructure has been reclaimed. A \$200,000 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 as sampling of water down stream from the site to determine if drainage from 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 Energy and Mines. Work can only commence once approval has been received.

Item 7: Accessibility, Climate, Local Resources, Infrastructure and Physiography

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 (Jakühlhlaup) 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. Redfern has applied to complete a road to their project site, 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 last 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 occur 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 -15°C. Summer temperatures, in July, average 10°C with daytime temperatures reaching the high 20's on occasion. The vegetation is typical of northern temperate rain forest, consisting primarily of fir, hemlock, spruce and cedar forest on the hillsides and aspen and alder groves in the river valley.

Item 8: History

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 30 feet long) was also driven into the side of the hill and Timmins drilled 19 holes for a total of 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 625 feet of drifting 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 775 feet of underground exploration in the "AJ" tunnel and collared 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 labour 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 162,163 feet in 220 holes were drilled on the AB, C and Y vein systems. Individual annual footages are provided in Table 2.

Table 2
Summary of Exploration Drilling to 2006

Year	Zone	No. of Holes	Footage
1988	Y vein	8	3373
1989	Y vein	19	13378
1990	C vein	10	9391
1991	Y & C veins	11	10934
1992	Y& C veins	23	20924
1993	C vein	8	4270
1994	C & Y veins, North Zone	30	17176
1995	North Zone	20	24934
1996	Underground	24	10514
1997	Underground	49	29098
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	5021
2004	C vein	7	5417
2005	C vein	8	7733
Total		220	162,163

A general distribution of this drilling can be seen in figure 6. 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 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 Author has 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 25 feet in the plane of the vein above and below sampled drift sections of minable grade while "possible" ore was projected an additional 25 feet 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 Author has 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 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. 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 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 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 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 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.

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 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.

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 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.

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.

Polaris Takus Geostatistical Resources								
Zone	PROBABLE RESOURCES				POSSIBLE RESOURCES			
	In-Situ		Diluted		In-Situ		Diluted	
	Tons (SDI)	Grade (oz/SDT)	Tons (SDI)	Grade (oz/SDI)	Tons (SDI)	Grade (oz/SDT)	Tons (SDI)	Grade (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

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.

Item 9: Geological Setting

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

Regional

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 underline the Stuhini volcanic rocks. The granodiorite is part of the Coast Plutonic Complex (Figure 4).

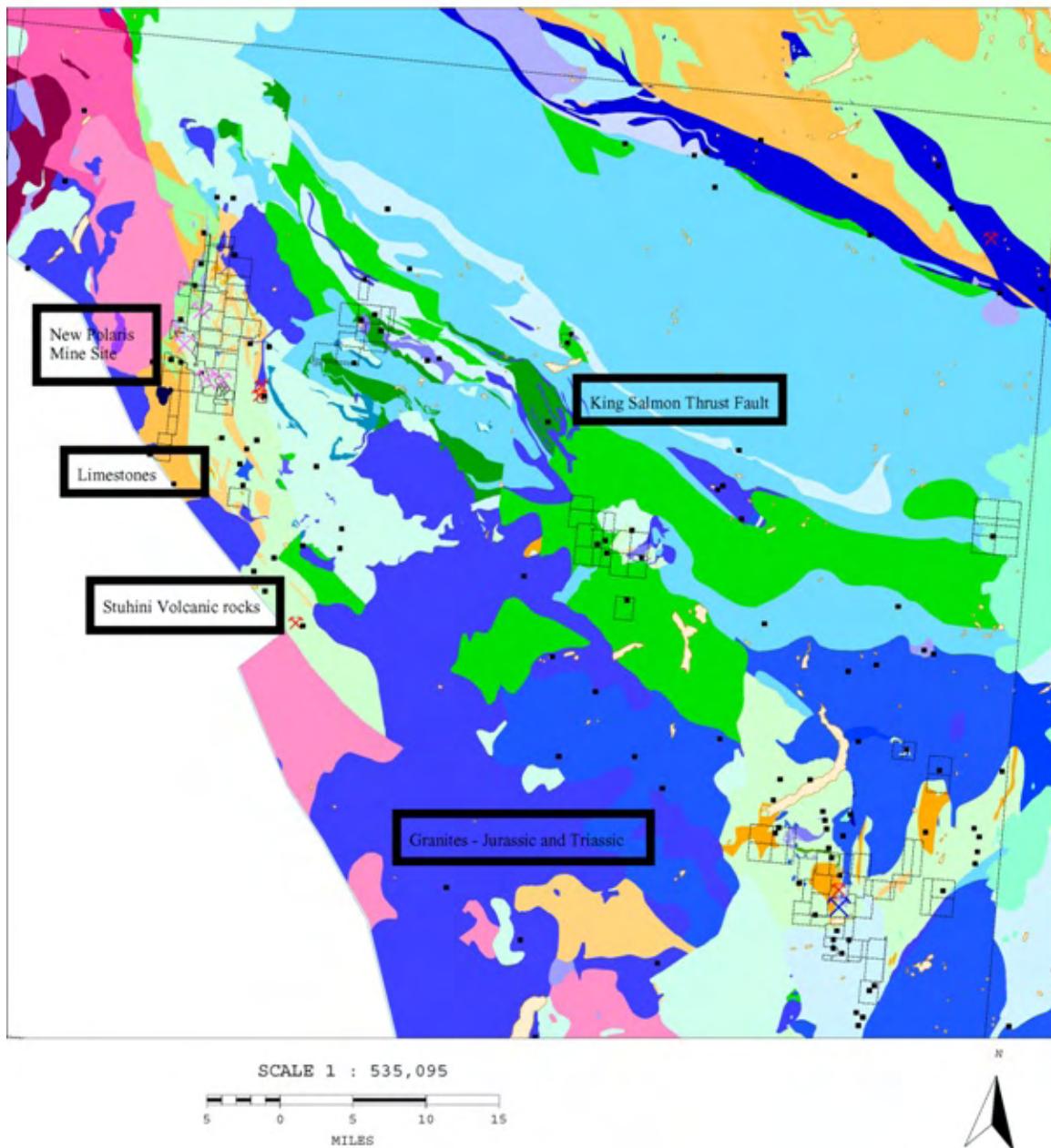


Figure 4 Regional Geology

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.

Property

Canarc has carried extensive mapping of the Polaris Taku property since the early 1990s. The work has been done by a number of employees and contractors and is shown in figure 5. 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 (70° to 80°) to the north based on the orientation of the limestone/basalt interface at the southern portion of the property.

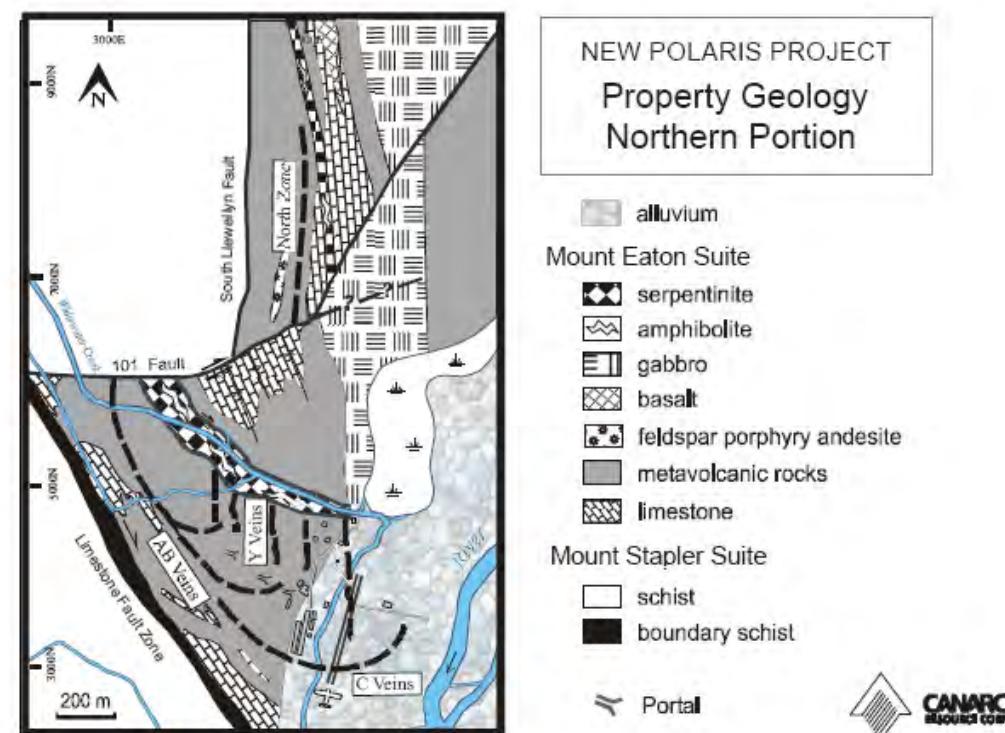


Figure 5 property Geology

A serpentinite unit is located to the northeast, which was identified in recent (996/97) 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 carbonitization 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 a 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 45° 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 zoned, 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.

Item 10: Deposit Types

The exploration target on the New Polaris project is orogenic load 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.

Prior mining and exploration have identified the vein systems so the 2003 to 2005 drill program was designed to test the vein systems between earlier drilled, widely spaced holes.

Item 11: Mineralization

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 than 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 should 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 with weaker control of the vein systems, 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 5 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 non mineralized 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.

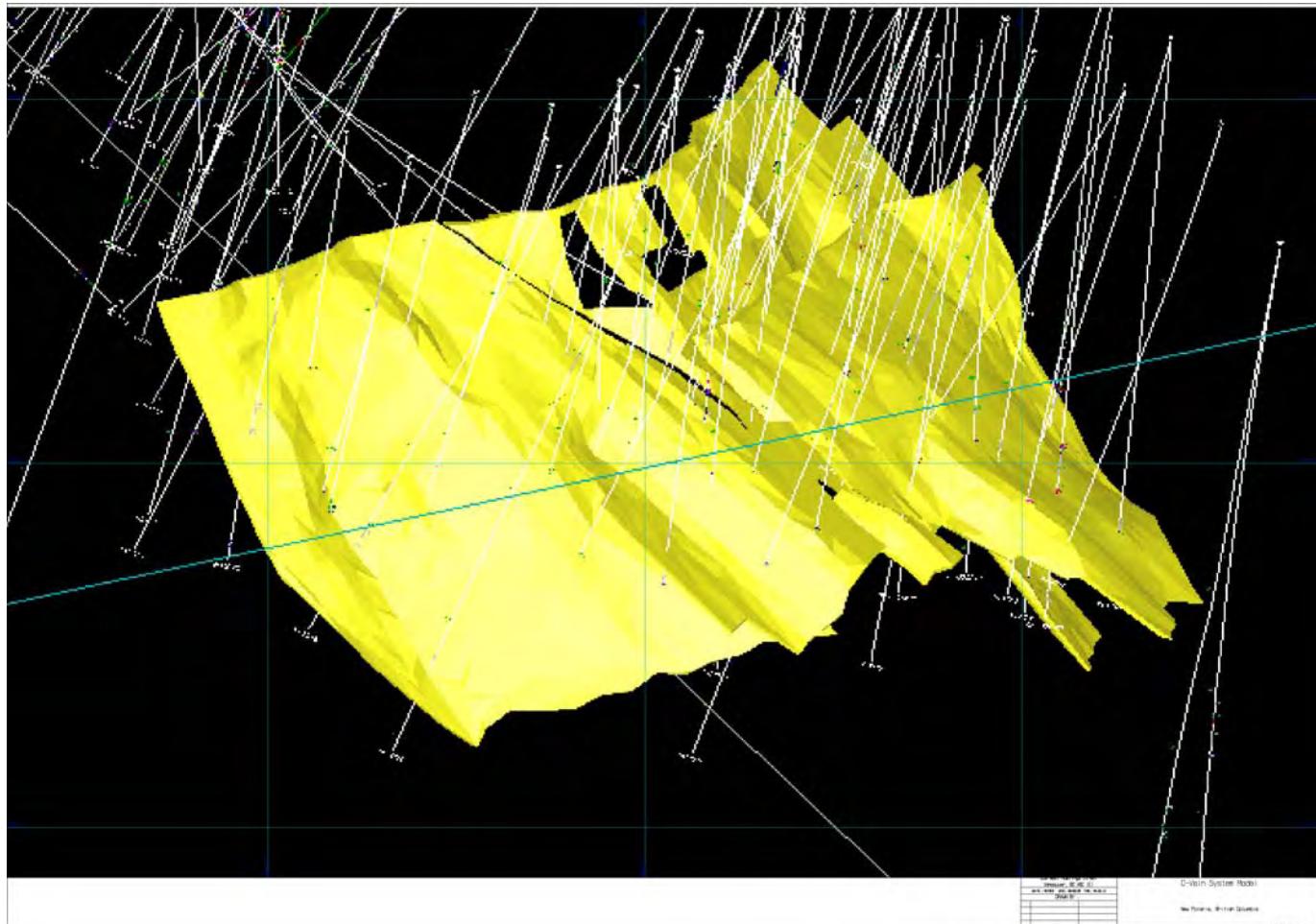


Figure 6 3D Model C Vein

Item 12: Exploration

The exploration program being reported on in the report consisted of 67175.5 metres of core drilling in 23 holes and preliminary metallurgical sampling of vein material from the AB vein system. Results of this work are reported under item 13 and item 18.

Item 13: Diamond Drilling Program

Diamond drill programs have been carried out on the New Polaris Project since the project was reactivated in 1988 (Table 2). 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 10 feet was present. From 1991 to 1993 most of the drill holes tested the C veins with a lesser number 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 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 lead 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 10 feet. The increased interest in the C vein systems 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 cross cuts 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 112 feet grading 0.42 opt). Despite the number of holes drilled during 1996 and 1997, the work did little to expanding the extent of the mineralization in the AB, C or Y vein systems. The work did confirm that the 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..

Table 3: Diamond Drill Holes (1988 to 2005)

HOLE-ID	Easting (m)	Northing (m)	Length (m)	Dip	Azimuth
03-P01	1569.72	1021.08	365.46	-70.0	345.0
03-P02	1513.33	1021.08	426.72	-70.0	330.0
03-P03	1591.06	1255.78	768.71	-60.0	260.0
04-1707E1	1706.88	873.25	259.08	-75.4	358.8
04-1707E2	1707.44	873.25	280.11	-83.0	359.0
04-1737E1	1737.63	873.16	287.43	-83.0	359.0
04-1737E2	1737.33	845.82	289.26	-83.0	320.0

04-300SW1	1842.61	902.79	224.03	-66.1	320.0
04-300SW2	1841.57	901.60	246.89	-75.0	320.0
04-300SW3	1842.70	902.73	248.41	-83.0	320.0
04-330SW1	1785.71	922.54	213.06	-81.7	323.8
04-330SW2	1789.62	904.06	237.44	-83.8	324.0
04-360SW1	1777.39	884.53	237.44	-75.0	318.5
04-360SW2	1777.39	884.53	243.84	-83.0	318.0
05-1676E1	1676.40	872.49	210.92	-71.0	359.0
05-1676E2	1676.40	872.49	240.49	-78.0	359.0
05-1676E3	1676.40	872.49	94.49	-83.0	359.0
05-300SW4	1875.64	860.29	289.86	-77.5	322.0
05-300SW5	1877.20	858.36	315.16	-85.0	318.0
05-300SW6	1911.06	816.86	347.17	-84.5	320.0
05-330SW3	1835.64	860.95	277.98	-77.0	320.0
05-330SW4	1835.69	860.88	298.40	-83.0	320.0
05-330SW5	1853.22	839.92	315.16	-83.0	320.0
L52	1465.17	1330.15	248.41	-25.0	85.0
L55	1510.89	1361.54	274.32	-17.0	40.0
L56	1498.70	1313.69	161.85	-25.0	92.0
L57	1464.26	1330.45	215.49	-25.0	51.0
L58	1473.40	1278.79	189.28	-25.0	112.0
L59	1477.67	1279.25	343.20	-25.0	95.0
L62	1463.04	1233.83	306.32	-25.0	149.0
L84	1342.64	1237.79	229.21	-2.5	247.5
P-9612	1289.97	1259.92	174.65	-35.0	119.0
P8918A	1819.79	889.97	367.89	-70.0	315.0
P91C01	1888.57	762.30	322.17	-65.0	338.0
P91C02	1888.57	762.30	331.62	-77.0	338.0
P91C03	1937.98	805.62	320.04	-75.0	339.0
P91C04	1937.98	805.62	324.92	-87.0	339.0
P91C05	1827.28	942.78	199.19	-73.0	339.0
P91C06	1672.83	818.27	263.96	-71.0	339.0
P91C07	1907.53	704.18	320.65	-75.0	335.0
P91Y01	1767.03	1369.25	330.10	-66.0	270.0
P91Y02	1769.59	1336.46	305.10	-64.0	266.0
P91Y03	1705.02	1095.33	274.93	-49.0	268.0
P91Y04B	1845.38	1336.03	444.09	-68.0	274.0
P92C08	1810.12	984.99	199.03	-71.0	342.0
P92C09	1889.24	964.20	210.31	-72.0	340.0
P92C10	1917.31	894.37	243.84	-76.5	340.0
P92C11	1978.82	925.86	251.16	-71.5	337.0
P92C12	1870.25	819.00	282.55	-71.5	337.0
P92C13	1635.16	860.39	223.72	-70.0	345.0
P92C14	1614.98	823.90	262.13	-70.0	343.0
P92C15	1702.92	823.57	258.17	-71.5	338.0
P92C16	1816.00	780.29	254.20	-71.5	338.0
P92C17	1730.41	865.11	264.26	-76.0	339.0
P92C18	1606.63	792.60	350.22	-70.0	337.0

P92C19	1694.87	761.73	381.30	-70.0	350.0
P92C20	1549.57	832.74	267.31	-70.0	350.0
P92C20A	1549.57	832.74	273.71	-70.0	350.0
P92C21	1534.49	871.36	213.97	-71.0	358.0
P92C22	1624.58	743.71	413.61	-71.0	336.0
P92C23	1518.98	873.06	274.62	-72.0	337.0
P92C24	1798.02	918.21	149.66	-61.0	331.0
P92Y05	1769.67	1335.82	346.56	-70.0	259.0
P92Y08	1787.35	1367.03	326.96	-67.0	274.0
P92Y09	1745.28	1336.55	248.11	-61.0	269.0
P93C25	1737.36	911.35	102.41	-65.5	340.0
P93C26	1858.98	1036.32	144.17	-70.0	295.0
P93C27	1860.13	1034.89	166.73	-85.0	295.0
P93C28	1882.41	1050.04	165.51	-75.0	310.0
P93C29	1906.65	1088.81	179.83	-75.0	280.0
P93C30	1928.77	1144.68	168.25	-75.0	280.0
P93C31	1815.61	901.17	215.49	-74.0	335.0
P94C32	1908.04	962.37	249.02	-72.0	303.0
P94C33	1908.37	961.28	279.81	-86.0	284.0
P94C34	1937.31	1015.90	258.17	-72.0	287.0
P94C35	1937.31	1015.90	339.55	-83.0	288.0
P94C36	1957.12	891.54	371.86	-80.0	288.0
P94C37	1883.66	789.74	279.50	-74.0	327.0
P94C38	1699.87	806.20	287.73	-82.0	348.0
P94C39	1769.36	768.10	290.17	-77.0	351.0
P94N01	1469.75	2124.76	84.73	-55.0	85.0
P94N02	1469.75	2124.76	84.73	-85.0	85.0
P94N03	1488.95	2194.26	60.35	-55.0	85.0
P94N04	1488.95	2194.26	106.07	-90.0	0.0
P94N05	1495.96	2258.26	118.26	-56.0	85.0
P94N06	1495.96	2258.26	124.36	-87.0	85.0
P94N07	1488.34	2332.63	224.94	-55.0	85.0
P94N08	1488.34	2332.63	273.71	-85.0	85.0
P94N09	1490.17	2438.10	264.57	-50.0	70.0
P94N10	1481.33	2438.10	96.93	-85.0	70.0
P94N11	1492.30	2499.36	75.59	-50.0	70.0
P94N12	1492.30	2499.36	71.02	-85.0	70.0
P94N13	1482.85	2384.76	66.45	-50.0	70.0
P94N14	1482.85	2384.76	66.45	-85.0	70.0
P94N15	1187.20	1822.70	51.21	-60.0	70.0
P94N16	1187.20	1822.70	57.30	-60.0	100.0
P94Y10	1663.90	1203.96	236.83	-63.0	266.0
P94Y11	1813.56	1280.16	251.76	-55.0	270.0
P94Y12	1813.56	1280.16	276.15	-63.0	265.0
P94Y13	1929.69	1191.46	371.86	-62.0	265.0
P94Y14	1929.69	1191.46	400.81	-55.0	262.0
P95C40	1965.96	575.77	786.69	-82.5	355.0
P95C41	2029.97	553.21	769.01	-80.0	360.0

P95C42	1861.72	571.50	673.00	-77.5	0.0
P95C43	1861.72	571.50	751.94	-80.0	340.0
P95C44	1920.24	525.78	792.48	-83.0	340.0
P95N17	1347.22	2122.93	264.57	-69.5	85.0
P95N18	1370.69	2183.89	270.66	-72.0	85.0
P95N19	1350.26	2243.33	315.47	-68.0	85.0
P95N20	1348.74	2302.76	335.28	-70.0	85.0
P95N21	1348.13	2329.59	215.80	-70.0	70.0
P95N22	1353.92	2387.80	227.99	-72.0	70.0
P95N23	1338.07	2449.07	261.52	-68.0	70.0
P95N24	1336.24	2449.07	294.74	-85.0	70.0
P95N25	1502.66	2804.16	255.42	-55.0	100.0
P95N26	1325.88	2804.16	273.71	-80.0	100.0
P95N27	1325.88	2804.16	334.67	-80.0	80.0
P9705A	1055.52	1545.03	231.34	-21.0	52.0
PC25A	1738.27	909.83	159.41	-68.0	336.0
PC44A	1920.24	525.78	749.81	-83.0	340.0
PC8801	1554.16	1178.32	106.68	-50.0	310.0
PC8802	1554.16	1178.32	38.71	-90.0	0.0
PC8803	1496.69	1146.74	133.20	-60.0	310.0
PC8804	1496.21	1141.91	119.48	-60.0	280.0
PC8805	1614.15	1363.59	149.35	-45.0	271.0
PC8806	1614.68	1363.74	189.28	-60.0	270.0
PC8807	1614.14	1364.20	180.14	-48.0	290.0
PC8808	1614.26	1365.03	110.95	-48.0	310.0
PC8901	1573.08	1149.96	142.34	-90.0	0.0
PC8902	1594.63	1088.77	128.32	-86.0	275.0
PC8903	1593.56	1088.51	167.64	-55.0	265.0
PC8904	1546.21	1068.50	50.90	-52.0	90.0
PC8904A	1546.21	1068.50	230.73	-52.0	90.0
PC8905	1546.73	1039.04	167.94	-60.0	90.0
PC8906	1705.42	1363.04	220.98	-55.0	268.0
PC8907	1704.52	1363.20	236.83	-70.0	290.0
PC8908	1704.52	1363.20	243.23	-50.0	290.0
PC8909	1639.71	1341.13	185.32	-45.0	270.0
PC8910	1639.14	1341.19	126.34	-59.0	270.0
PC8911	1639.77	1342.07	187.45	-53.0	310.0
PC8912	1726.56	1440.13	307.85	-45.0	270.0
PC8913	1727.76	1440.48	244.75	-60.0	270.0
PC8914	1726.95	1439.69	253.59	-56.0	256.0
PC8915	1726.18	1163.72	413.31	-65.0	270.0
PC8916	1762.08	1377.37	233.78	-55.0	270.0
PC8917	1676.68	1157.69	171.30	-45.0	270.0
PC8918	1819.79	889.97	191.11	-70.0	315.0
PC8919	1777.26	1103.56	219.15	-60.0	270.0
PC9001	1820.17	822.27	400.51	-70.0	340.0
PC9002	1838.20	767.83	388.92	-70.0	340.0
PC9003	1746.57	837.70	305.10	-70.0	340.0

PC9004	1853.47	897.63	305.71	-70.0	340.0
PC9004A	1853.47	897.63	108.91	-70.0	340.0
PC9005	1675.88	852.91	265.48	-60.0	340.0
PC9006	1762.29	792.22	404.16	-70.0	340.0
PC9007	1605.02	880.79	242.62	-68.0	340.0
PC9008	1778.45	746.71	441.05	-70.0	330.0
PT23A	1317.96	1244.50	206.65	-32.0	195.0
PT29A	1317.96	1244.50	325.53	-31.0	134.0
PT30A	1317.96	1244.50	279.81	-15.0	90.0
PT31A	1334.41	1456.94	406.30	0.0	20.0
PT37B	1595.05	1102.00	179.53	-52.0	45.0
PT38A	1595.32	1097.49	260.30	-35.0	124.0
PT9601	1444.26	1379.06	106.07	-36.0	79.0
PT9602	1441.21	1379.06	120.40	-51.0	79.0
PT9603	1441.22	1378.72	102.11	-36.0	96.0
PT9604	1441.22	1378.72	130.76	-51.0	96.0
PT9605	1441.09	1379.36	101.80	-49.0	68.0
PT9606	1441.33	1378.29	115.82	-38.0	110.0
PT9607	1436.92	1380.33	43.59	21.5	291.0
PT9608	1436.92	1380.62	17.98	21.5	299.0
PT9609	1437.02	1380.72	119.48	21.5	304.0
PT9610	1437.02	1380.73	119.48	-4.5	304.0
PT9611	1436.91	1380.62	125.88	-22.0	299.0
PT9613	1289.76	1259.71	149.05	-41.0	134.0
PT9614	1289.43	1258.88	121.01	-52.0	164.0
PT9615	1289.76	1259.71	147.52	-52.0	134.0
PT9616	1289.97	1259.92	176.78	-46.0	119.0
PT9617	1288.85	1263.64	152.10	0.0	15.0
PT9618	1055.52	1545.03	146.00	18.0	88.0
PT9619	1055.52	1545.03	127.41	18.0	108.0
PT9620	1055.52	1545.03	173.74	-10.0	118.0
PT9621	1055.52	1545.03	141.43	-8.5	80.0
PT9622	1055.52	1545.03	152.10	22.0	108.0
PT9623	1055.52	1545.03	167.34	-14.0	110.0
PT9624	1055.52	1545.03	272.19	-10.0	52.0
PT9701	1055.52	1545.03	9.75	-5.0	52.0
PT9702	1055.52	1545.03	287.73	-7.0	52.0
PT9703	1055.52	1545.03	220.07	-13.0	60.0
PT9704	1055.52	1545.03	205.44	-10.0	60.0
PT9706	1055.52	1545.03	255.42	-9.5	49.0
PT9707	1055.52	1545.03	240.18	10.0	52.0
PT9708	1049.12	1542.59	144.48	0.0	242.0
PT9709	1217.37	1301.04	69.80	-22.0	0.0
PT9710	1217.98	1301.19	69.49	-22.0	340.0
PT9711	1218.90	1301.04	163.37	-15.5	78.0
PT9712	1218.90	1301.04	171.30	4.0	60.0
PT9713	1222.10	1299.97	281.33	-15.0	90.0
PT9714	1219.20	1297.84	47.61	-42.0	0.0

PT9715	1218.90	1301.04	59.74	-46.0	330.0
PT9716	1217.07	1296.62	135.33	-15.0	270.0
PT9717	1217.07	1296.62	154.84	-30.0	270.0
PT9718	1217.07	1296.62	118.26	-18.0	249.0
PT9719	1290.83	1417.62	112.47	-5.0	90.0
PT9720	1290.83	1417.62	133.81	-35.0	90.0
PT9721	1290.52	1417.62	153.92	-44.0	71.0
PT9722	1289.30	1418.23	219.15	-5.0	270.0
PT9723	1317.96	1244.50	17.98	-32.0	195.0
PT9724	1317.96	1244.50	192.94	-30.0	217.0
PT9725	1317.96	1244.50	244.14	-47.0	218.0
PT9726	1317.96	1244.50	124.66	-10.0	334.0
PT9727	1317.96	1244.50	110.34	-12.0	351.0
PT9728	1317.96	1244.50	228.60	-31.0	351.0
PT9731	1334.11	1456.94	19.20	0.0	20.0
PT9732	1594.99	1102.89	229.51	-22.0	26.5
PT9733	1594.65	1102.28	272.49	-52.0	23.5
PT9734	1595.23	1101.76	155.14	-32.0	50.0
PT9735	1594.96	1102.83	198.73	-10.0	26.0
PT9736	1594.87	1102.95	237.13	-29.0	24.0
PT9738	1595.35	1097.49	14.33	-35.0	124.0
PT9739	1594.96	1097.43	204.52	-29.0	136.0
PT9740	1594.53	1097.25	180.75	-34.0	146.0
PT9741	1594.93	1097.74	155.14	-42.0	130.0
PT9742	1593.46	1096.88	247.65	-37.0	188.0
PT9743	1592.92	1096.85	246.58	-36.0	210.0
PT9744	1594.71	1097.40	290.17	-38.0	134.0
PT9744A	1594.71	1097.40	136.55	-35.5	141.5
PT9745	1593.10	1096.85	219.46	-26.0	193.0
PY06B	1769.36	1406.96	295.05	-60.0	273.0
PY07A	1772.11	1406.96	387.10	-67.0	275.0

Table 6 Assay Composites C Vein System

HOLE-ID	FROM	TO	LENGTH Metres	True Width	AU_COMP GPT
04-1707E1	201.93	203.03	1.10	C	10.20
04-1707E1	209.46	213.42	3.96	C	5.10
04-1707E2	222.08	224.73	2.65	C	10.10
04-1707E2	246.64	248.72	2.07	C	11.80
04-1737E1	184.25	190.80	7.77	C	17.13
04-1737E1	200.25	207.57	7.32	C	4.00
04-1737E1	231.19	235.37	4.18	C	13.60
04-1737E2	233.66	233.96	0.30	C	30.90
04-1737E2	256.64	259.08	2.44	C	9.20
04-300SW1	180.75	181.72	0.98	C	11.20
04-300SW2	181.20	181.66	0.46	C	5.50
04-300SW2	206.53	208.94	2.41	C	14.80

04-300SW3	196.38	203.61	7.22	C	7.20
04-300SW3	223.88	230.73	6.86	C	14.50
04-330SW1	167.18	168.80	1.62	C	4.00
04-330SW1	172.36	174.49	2.13	C	35.30
04-330SW2	168.86	171.60	2.74	C	5.40
04-330SW2	194.22	202.27	8.05	C	31.90
04-360SW1	162.76	163.67	0.91	C	5.10
04-360SW1	179.10	193.43	14.33	C	11.60
04-360SW2	191.20	195.38	4.18	C	25.70
04-360SW2	198.24	201.87	3.63	C	5.20
05-1676E1	185.62	191.20	5.58	C	12.10
05-1676E2	210.01	213.27	3.26	C	8.90
05-300SW4	200.13	200.86	0.73	C	13.40
05-300SW4	261.34	269.05	7.71	C	17.40
05-300SW5	215.80	218.30	2.50	C	14.10
05-300SW5	241.10	246.74	5.64	C	21.60
05-300SW6	233.54	240.79	7.25	C	16.70
05-300SW6	260.91	266.09	5.18	C	18.20
05-330SW3	213.94	223.11	9.17	C	9.90
05-330SW4	221.89	233.00	11.89	C	19.88
05-330SW5	230.89	246.28	15.39	C	8.57
P8918A	194.43	199.49	5.06	C	37.70
P91C01	251.83	258.17	6.34	C	7.80
P91C01	274.93	281.48	6.55	C	20.20
P91C02	263.35	268.10	4.75	C	25.70
P91C03	226.47	229.64	4.20	C	11.46
P91C03	264.99	269.57	4.57	C	0.70
P91C04	282.55	288.74	6.19	C	4.80
P91C05	156.73	169.10	12.37	C	10.30
P91C06	228.66	231.31	2.65	C	22.40
P91C07	292.06	294.35	2.29	C	18.80
P92C08	124.05	127.35	3.29	C	8.90
P92C09	161.85	163.98	2.13	C	4.20
P92C12	216.26	218.08	1.83	C	4.20
P92C12	258.47	265.27	6.80	C	15.20
P92C13	190.56	192.39	1.83	C	15.00
P92C14	246.10	248.69	2.59	C	6.90
P92C15	231.77	233.17	1.40	C	4.80
P92C15	236.89	238.66	1.77	C	18.70
P92C16	229.36	231.65	2.29	C	4.20
P92C17	209.70	213.48	3.78	C	21.80
P92C18	281.18	283.98	2.80	C	9.20
P92C19	282.64	285.51	2.87	C	23.80
P92C20	243.35	245.97	2.62	C	8.90
P92C20A	233.08	235.82	2.74	C	12.50
P92C21	172.15	178.06	5.91	C	18.70
P92C22	343.27	345.64	2.38	C	1.90
P92C23	184.13	186.14	2.01	C	1.60

P92C24	136.52	139.39	2.87	C	23.70
P93C26	108.02	110.95	2.93	C	26.60
P93C27	148.89	151.49	2.59	C	12.50
P93C28	140.27	142.43	2.16	C	10.50
P93C31	184.80	186.99	2.19	C	8.00
P94C32	167.46	168.37	0.91	C	0.10
P94C32	213.33	215.95	2.62	C	0.10
P94C33	266.43	269.53	3.11	C	6.90
P94C34	176.72	180.14	3.41	C	7.20
P94C36	261.21	262.28	1.07	C	6.40
P94C37	255.24	261.27	6.04	C	22.20
P94C38	255.54	257.13	1.58	C	20.10
P94C38	274.53	279.23	4.69	C	22.90
P94C39	262.01	266.58	4.57	C	28.70
P95C40	493.01	498.93	5.92	C	8.47
P95C40	727.77	735.24	7.47	C	11.31
P95C42	456.47	457.54	1.10	C	13.13
P95C43	482.04	484.51	2.47	C	15.89
P95C44	586.50	589.57	3.07	C	18.76
P95C44	640.08	641.70	1.62	C	12.41
P95C44	692.38	694.15	1.77	C	28.06
PC25A	137.31	141.00	3.69	C	5.60
PC44A	586.40	591.22	4.78	C	16.00
PC44A	726.40	730.60	4.18	C	6.00
PC9001	206.35	213.79	7.44	C	11.80
PC9002	277.31	279.96	2.65	C	29.30
PC9003	215.59	222.50	6.92	C	14.80
PC9003	227.17	232.87	5.70	C	31.20
PC9004	221.25	225.25	3.99	C	16.10
PC9005	189.43	190.96	1.52	C	18.90
PC9006	253.62	257.28	3.66	C	0.60
PC9007	160.90	162.31	1.40	C	15.20
PC9008	300.08	301.35	1.28	C	0.50

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 under took 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 higher confidence level for the mineral 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 7.

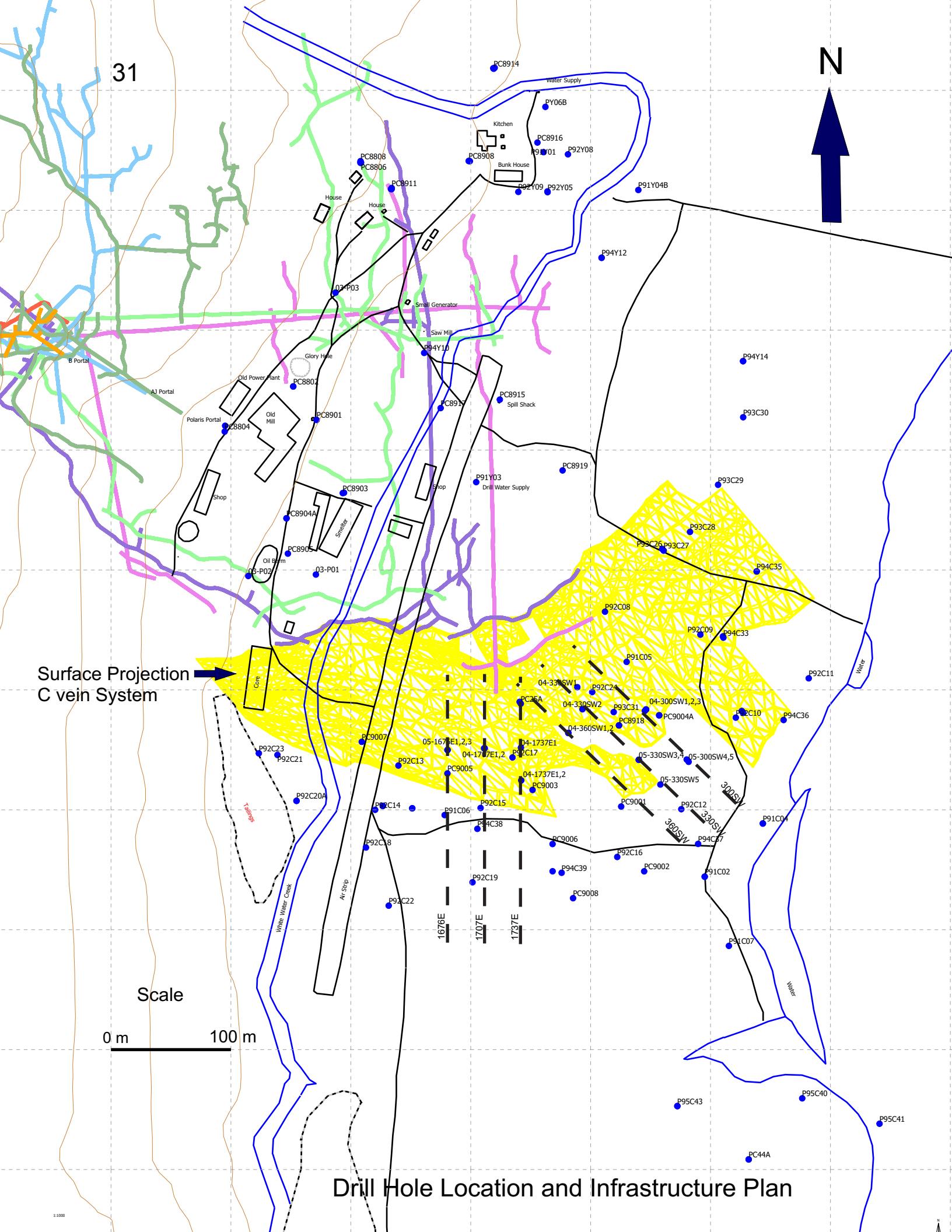
Because there are 220 drill holes on the property it is not possible to plot all of these holes in page size format and still be legible. A more complete plan map is presented in Appendix III. this map is in PDF format and can be expanded to see detail using Adobe Acrobat 7 reader. An inclined section showing the pierce points of drill holes and grades of the 2003 to 2005 drilling is presented in figure 8. 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 9 through 11.

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 2 or more branches. The dip of the vein system is to the south and southeast and has an average dip of about 50°, although east of the No.1 fault the vein appears to flatten and thicken in a sigmoid-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 just over a foot to a maximum of 50 feet. The thicker parts of the vein occur to the east of the Number 1 Fault where the dip of the vein flattens due to an apparent folding of the vein.

Table 6 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 13 feet and the average grade is 0.4 opt gold. The estimated average true thickness of the vein is 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 50 to 260 feet of overburden followed by inter layered ash and lapilli tuff, volcanic wacke, and foliated andesite. The C vein system cross cuts the strike of the volcanic and volcaniclastic rocks at steep angles.



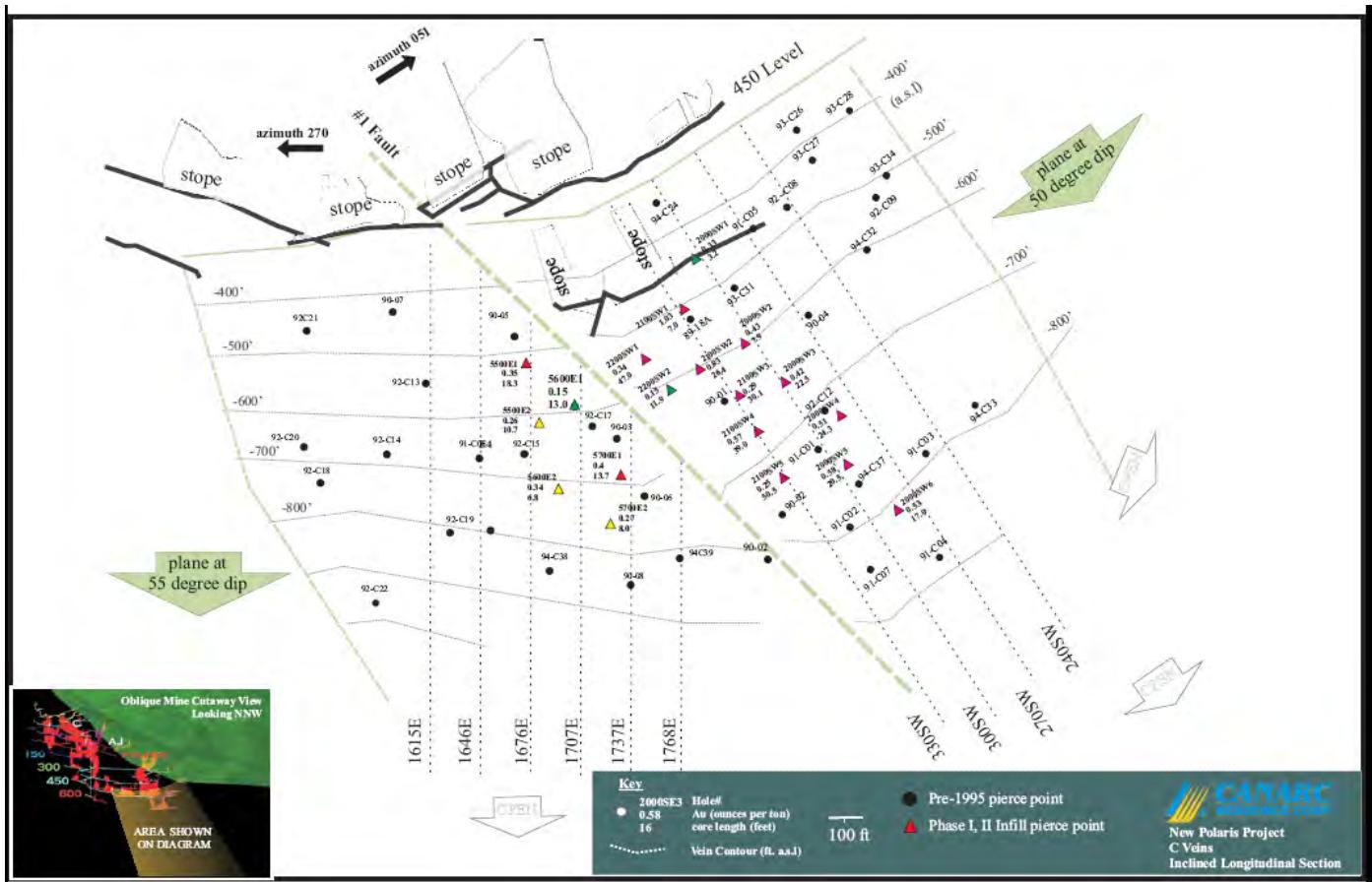


Figure 8 Inclined Longitudinal Section

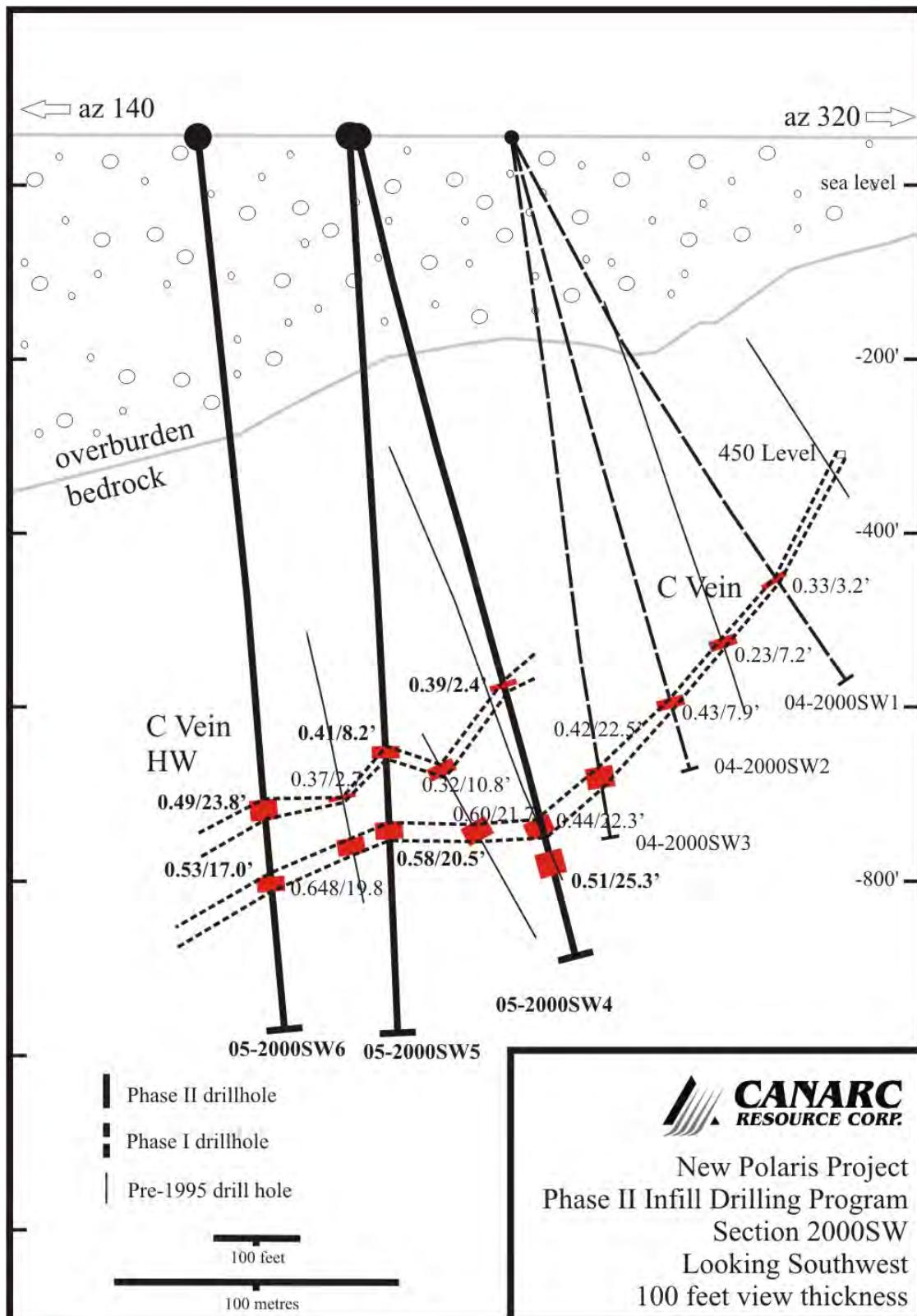


Figure 9 Section 2000SW

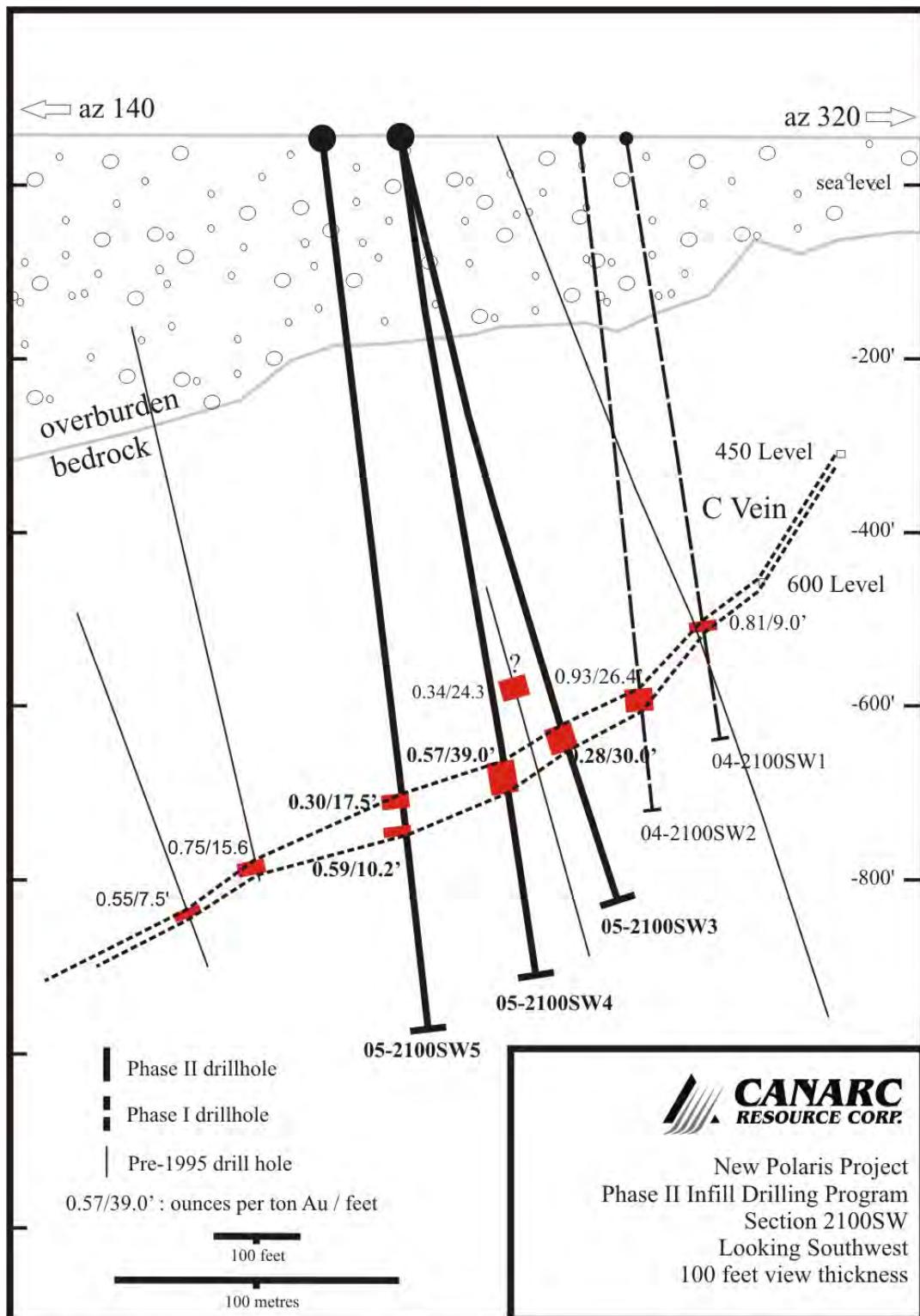


Figure 10 Section 2100 SW

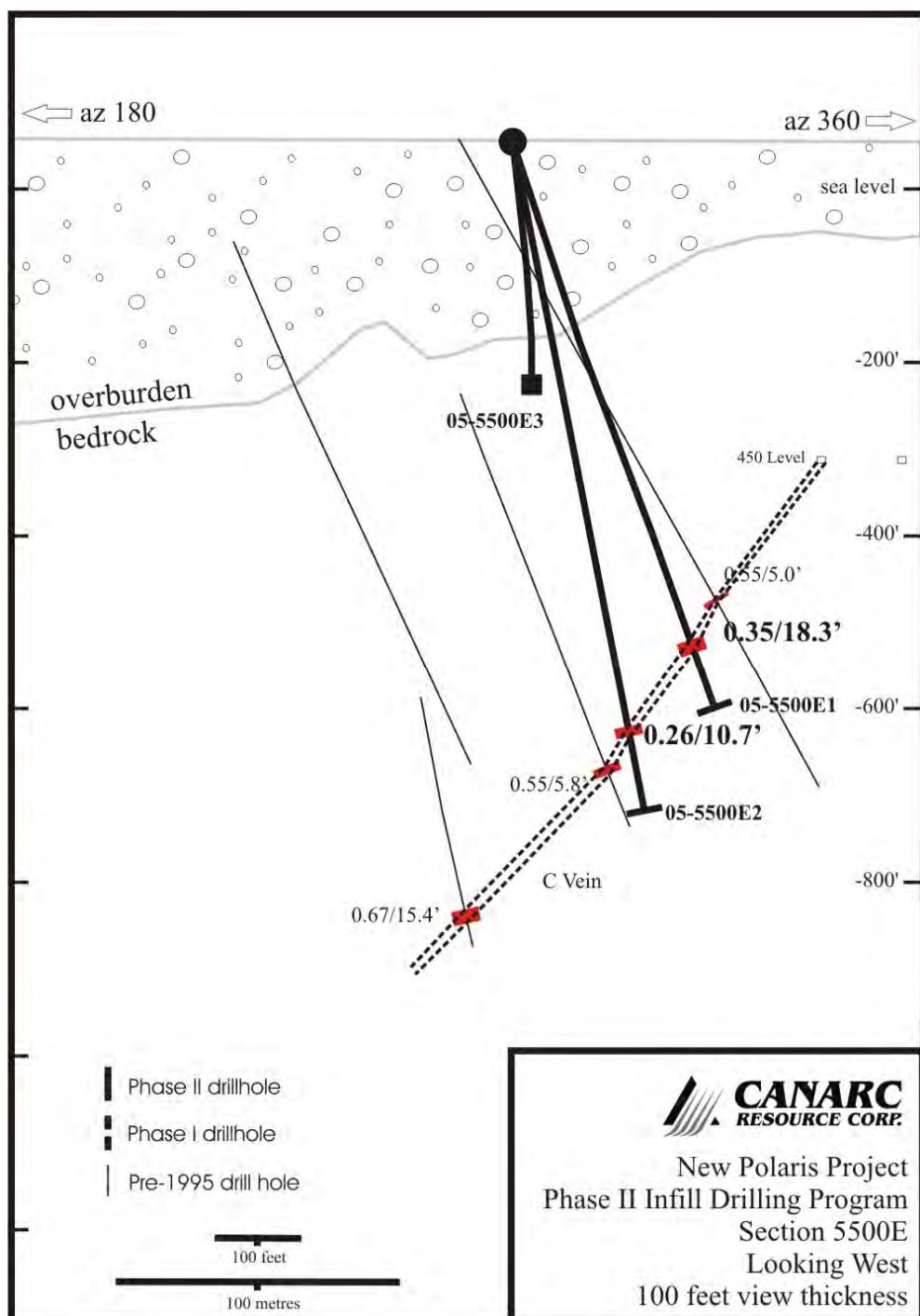


Figure 11 Section 5500E

Item 14: Sample Method and Approach

Sampling of the vein was done by a 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 down hole measurements of azimuth and dip were not done on the three holes drilled in 2003 and prior. In 2004, three different down hole survey systems were tried before settling on a Reflex system. The Reflex system was also used in 2005. The down hole 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 non mineralized 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 4700 samples with greater than .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 not been done in calculating composites or 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. Selections of core intervals for sampling were based in the presences of veining and sulphide mineralization particularly arsenopyrite. Within the defined vein structure sample interval ranged from 1 foot to 5 feet. Divisions were based on intensity of mineralization and veining. Sampling of the core for 10s 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 set up 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.

Composite assay results for the C vein are provided in Table 4 and assay results for the holes drilled from 2003 through 2005 are listed in Appendix I.

Item 15: Sample Preparation, Analyses and Security

Core splitting and packaging for shipment was done on site by employees of Canarc. Core was placed in plastic bags sealed with locking ties. Up to 6 samples were packaged in rice bags that were also sealed with locking ties. The rice bags were addressed and shipped to ASL Chemex's laboratory at 212 Brooksbank North Vancouver. A sample shipment notice was sent separately to ASL to advise it of the quantity, samples numbers and analytical instructions included in the shipment. A paper copy of the same information was included with each shipment.

The samples were shipped from the property via fixed wing aircraft operated by Atlin Air. The sample shipment was received in Atlin by an expeditor from TME Expediting. The expeditor arranged for ground transportation from Atlin to Whitehorse where they were delivered to Air North for shipment to ASL Chemex. ASL Chemex was instructed to notify Canarc in the event that any of the sample bags were missing or if the bags had been opened en route. There were no incidents of tampering with samples.

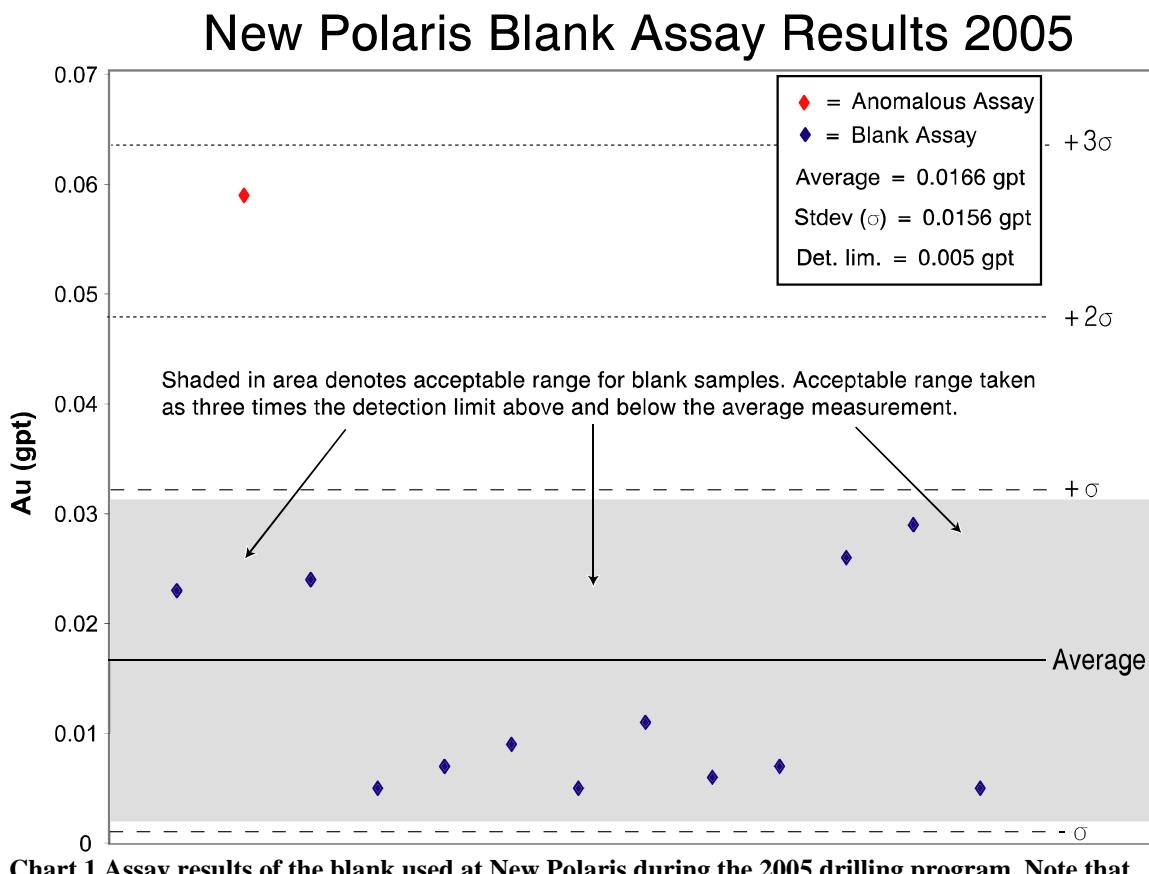
ALS Chemex laboratories in North America are registered to ISO 9001:2000 for the provision of assay and geochemical analytical services by QMI Quality Registrars. In addition, ALS Chemex's main North American laboratory in North Vancouver, BC, Canada, is accredited by the Standards Council of Canada (SCC) for specific tests listed in our Scope of Accreditation No. 579 which is available at http://palcan.scc.ca/specs/pdf/677_e.pdf. This accreditation is based on international standards (**ISO 17025**) and involves extensive site audits and ongoing performance evaluations.

Samples received by ASL Chemex were dried, crushed, split, and a thirty gram sub sample taken for assay. Gold content was by fire assay with a gravimetric finish on samples containing greater than 1 gpt. Other elements including arsenic and antimony were analyzed by atomic absorption.

Three types of controls are employed to assure the quality of the assays. Blanks, standards and duplicates are inserted in with the samples sent to the assay laboratory.

Blanks are simply a texturally and chemically uniform rock which contains no, or an insignificant amount of Au. Blanks are used to test for contamination during the assaying process. Standards are a rock material with a known amount of gold within. These are used to test the laboratory's accuracy in determining the amount of gold in a given sample. Finally, duplicates are simply a second quantity of a real sample. These are submitted to test the reproducibility of the assay results. The way core is split has the most affect on the reproducibility a sample, especially in heterogeneously mineralized rocks. One of each, a standard, a blank and a duplicate, were submitted with roughly every 20 total samples (including the standard, etc.).

Blanks should return values less than or equal to three times the detection limit of Au for the assaying method reported by the lab (reported as 0.005 ppm or gpt). Chart 1 shows the results of 13 blank analyses. Measurements which fall within the grey shaded area are measurements which fall within three times the detection limit of the average measurement. These samples are considered to contain no contamination. One sample however, does fall outside the acceptable range. Rather than contamination, it is likely the result of a minor 'nugget effect' of gold within the blank material. Regardless, the 0.06 gpt measurement, whether it is from contamination or a minor nugget effect, is insignificant and would not affect a resource calculation.



Two standards were used and inserted in with the core samples to be assayed, PM 911 and PM 913. The results of both standards and the statistics of each are shown in Charts 2 and 3. For PM 911, 5 measurements fall outside the 95% confidence level, or within two times the accepted standard deviation above or below the accepted value of the standard. In general, the measurements for PM 911 are somewhat lower than the accepted values stated within the datasheet accompanying the standards. This is likely the result of a different lab technician and/or “drift” in the equipment used. The 5 batches of samples which contain the failed SRM samples will have to be re-assayed at some point. All measurements of PM 913 fall within the range.

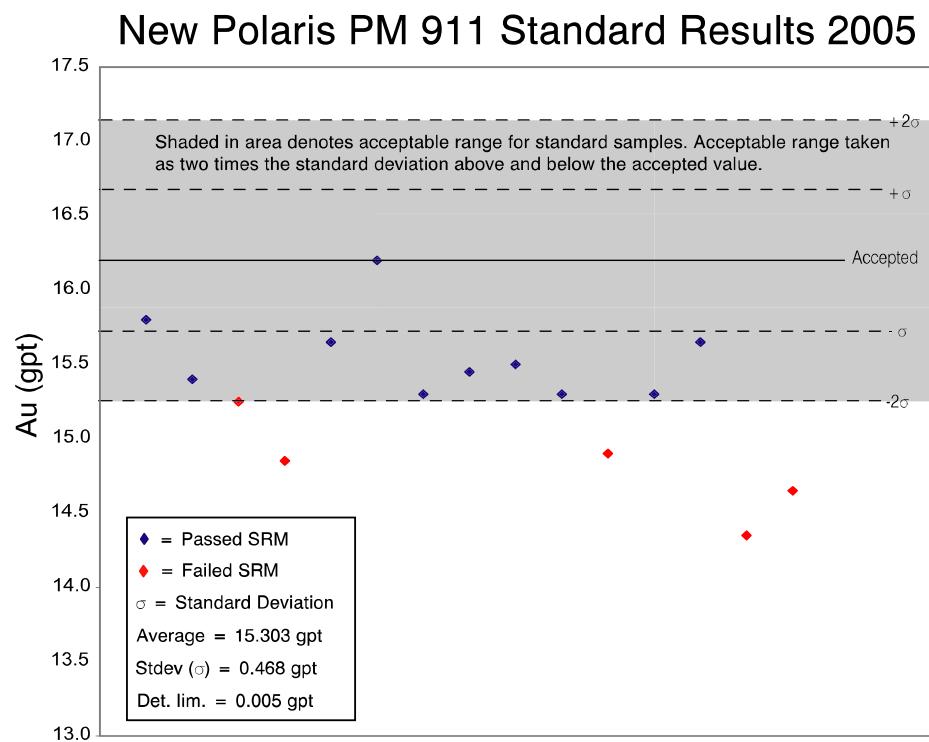


Chart 1: Assay results for standard PM 911 from the 2005 drilling program. X axis is simply for separating samples.

New Polaris PM 913 Standard Results 2005

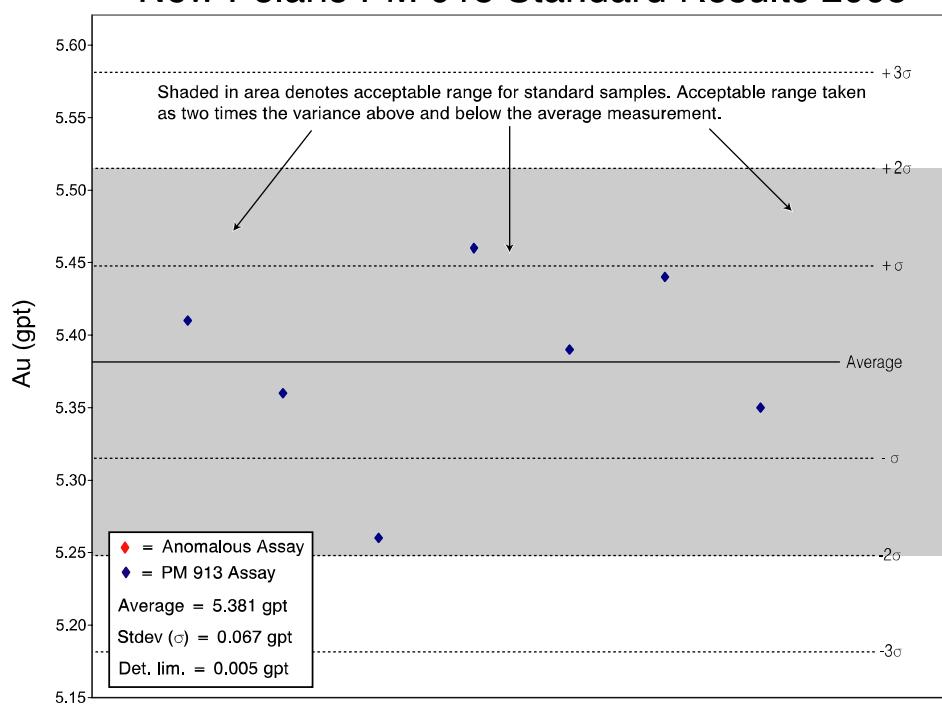


Chart 2: Assay results for standard PM 913 from the 2005 drilling program. X axis is simply for separating samples.

Results for the analysis of the duplicate assay results are shown in Charts 4 and 5. An gold content plot of samples verses their duplicates (Chart. 4) produces a linear regression line who's slope is comparable to that of a $y = x$ plot. This is indicating that there is not a systematic bias during either the core splitting process or the assaying itself.

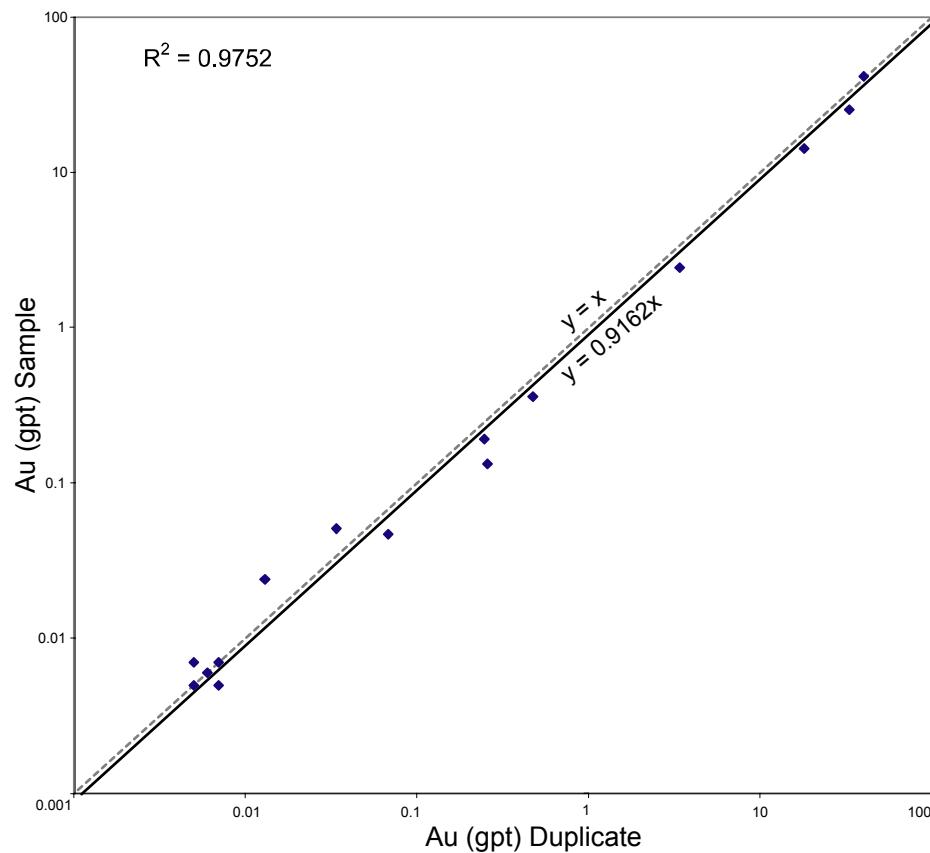


Figure 3: Au content of sample verses the duplicate.

Chart 5 shows the relative percent difference the duplicate assay is from the sample the duplicate was made from. In general, the gold assay results from most duplicates within are within 30% the value determined for the sample. This indicates that in general, the core splitting process was in general carried out properly. In other words, either heterogeneously mineralized core was split such that both had an even gold content, or the core was homogeneous. Duplicates which are significantly different could be caused by heterogeneously mineralized core. Close attention must be paid to the core splitting process to minimize this effect.

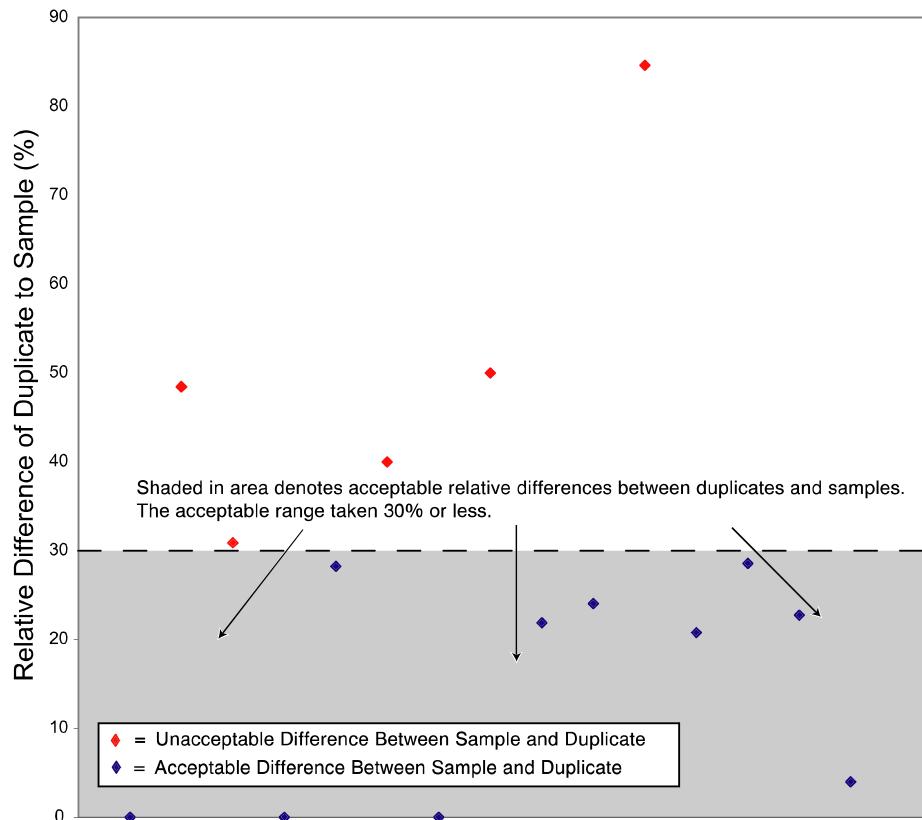


Chart 4: Relative difference percent difference between samples and duplicates. X axis is simply for separating samples.

In the author's opinion the sample preparation, security and analytical procedures were adequate.

Item 16: Data Verification

The Author has spent a total of 7 days on the New Polaris project. 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 core from key holes drilled from the early 1990s to the end of 2005 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, the Author has reviewed assay sheets and compared these with the database. These were accurately transposed into the database. The core-logging sheets have been compared to the digital logs to check to accuracy. All sampling in the core was carefully laid out using geological parameters. The author discussed procedures with James Moors, P.Geo., who supervised the drilling program and was responsible for all of the work completed at the New Polaris property in the 2003 to 2005 period. The Author is satisfied with the data verification procedures used by Canarc on this project.

The Author reviewed the quality control instituted. The core logging facility was clean and orderly. The system of check assaying is adequate. The only issue that the Author has with the system is the use of quartered core for the duplicate samples. The sample size difference between the quartered and half core may account in part for the high relative difference between the original sample and the duplicate. In future, resubmission of pulps on a blind basis should be carried to help separate variance cause by analysis from that due to sample size or bias cutting of the core.

The quality control systems in place prior to the 2003 program are poorly documented and seem to follow the norms of that time period. Of particular concerned is the manner in which the collar locations of drill holes were determined. Most of the holes were located using brunton compass and chaining. Also, the down hole surveying was not consistently done. As a result, the exact location of the vein intersections is not as certain as those from the 2003 to 2005 program. Some re drilling of prior holes is required especially where there are discrepancies with respect to the vein location between the recently drilled holes and those drilled in the 1990s.

Item 17: Adjacent Properties

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

Item 18: Mineral Processing and Metallurgical Testing

As part of the 2003 to 2005 exploration program a 200kg sample of vein mineralization was collected from a muck pile on the “AJ” mine working level. The mineralization sampled was a single boulder from the AB vein that had been exposed to air for over 40 years. For the following reason this sample may not be representative of the “C” and “Y” vein systems:

- Although care was taken to remove the oxidized material from the surface of the boulder, there is no guarantee that the oxidation was not present along internal fractures.
- The sample is of AB material and not from the “C” vein system that is the focus of the current work program.
- The sample is from a single point in the vein and may not be representative of other parts of the vein systems.

Two metallurgical studies were carried out on the 200kg sample. The initial test by Resource Development Incorporated (RDi) had the primary objective of determining the deportation of gold in various minerals, reconfirm and/or optimize the flotation process conditions and produce a flotation concentrate containing maximum gold recovery and

minimum amount of carbonate for bio-oxidation studies. A copy of the RDI work can be found in Appendix II.

A second metallurgical study on the amenability of the concentrate to bio-oxidation was carried out by Mintek on the concentrate produced by RDI. This report is also available in Appendix II.

The RDI study highlighted the following:

- The composited sample received by RDI assayed 19.41gpt, 2.835% As, 122ppm Sb, 1% S and less than 1.7gpt Ag.
- The major minerals in the host rock were dolomite (32 to 35%), micalillite (32%), and quartz (23%).
- Only 9% of the gold was free milling and therefore the ore is refractory. Approximately 4.5% of the gold was associated with stibnite and 68% of the gold was associated with arsenopyrite and pyrite. About 20% of the gold is associated with quartz, possibly due to find grained arsenopyrite encapsulated in quartz.
- Base-case flotation test using potassium anylxanthate (PAX) and methyl isobut carbonal (MIBC) recovered 83.6% of the gold and 83.5% are arsenic in a concentrate assaying 73.63gpt gold. The concentrate weight recovery was 20.7%
- Modified flotation test using Na_2S in the grind and CuSO_4 in the flotation along with base-case reagents improved the flotation recovery and concentrate grade to 93.9% to 96.1% and ± 94 gpt gold respectively.
- Cleaner flotation tests indicate that significant amount of carbonates in the concentrate could be rejected in the first cleaner flotation (78.6%) while loosing only 3% of the gold. The carbonate content was reduced from 15% in the rougher concentrate to less and 1% in the first cleaner concentrate. The concentrate weight was also reduced from 18% to 15.27%.
- A second cleaner did not reject sufficient gangue material, but reduced gold recovery by an additional 5%, therefore only one stage of cleaning is justified.
- Gravity concentrate did not preferentially upgrade rougher concentrate or recover additional gold from the rougher tailings.
- Leaching of rougher tailings resulted in 2% additional gold recovery. However it is not economically justified due to high NaCN consumption.

In summary, a flotation process consisting of roughers and one stage of cleaners can produce a concentrate assaying ± 110 gpt gold and $\pm 15.7\%$ arsenic with the weight and gold recoveries of 15.3% and 95.9% respectively. The concentrate will contain less than 1% carbonate.

The bio-oxidation study carried out by Mintek used a 12 kg sample of rougher concentrate obtained from RDI. This work showed that high levels of iron and arsenic extraction were achieved, however, there may be inhibitory substance associated with the concentrate. Further bio-oxidation studies were carried out by Oxidor Corporation who

concluded that bio-oxidation of the New Polaris concentrate appears facile and economically attractive. Gold recovery by bio-oxidation was later estimated by RDi to be 92%.

RDi also undertook pressure oxidation tests on the sample. This test concluded that the average gold extraction was 98% in 24 hours with NaCN consumption ranging from 0.8 to 1.4 Kg/T. It concluded that pressure oxidation was more attractive than bio-oxidation.

Item 19: Mineral Resource and Mineral Reserve Estimates

No independent NI 43-101 compliant resource estimate has been calculated for the New Polaris Deposit.

Item 20: Other Relevant Data and Information

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

Item 21: Interpretation and Conclusions

The drilling programs conducted since 2002 have 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.

The grades and thickness of the “C” is encouraging and are in the range of other gold producing mines of similar deposit type. However, more drilling is required to confirm the vein continuity and increase the confidence level to the point where a resource can be calculated. Additional geological information is required to better understand the structural complexity of the “C” veins, particularly to the northeast of the No.1 fault.

Metallurgical 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 also yielded positive results with 96 % and 93 % recoveries respectively. The reader is cautioned that the sample used in these tests came from a single boulder and may have contained oxidized material. For these reasons, the sample may not be representative of the mineralization in the “C” vein system.

Item 22: Recommendations

The drilling programs completed from 2003 to 2005 were successful in demonstrating the continuity and grade of the “C” vein system. Therefore, the phase one work program recommended by Walton is completed and his recommended second phase should be undertaken. To this end, it is recommended that 20,000m of infill drilling in the upper

portion of the “C” and vein system be carried out in 2006. The plan that would see about 60 holes drilled in order to reduce the drill hole spacing to approximately 30 metres.

The author is of the opinion that 30-metre spacing will allow a resource at the indicated and measured category to be calculated. This calculation will be determined if the grade and tonnage of the mineralization is sufficient to warrant proceeding to a next phase of preliminary economic evaluation of the resource.

Additional metallurgical testing is also recommended for the 2006 program. This testing of the reject material would use drill reject material from the 2006 drilling in order to provide a fresh representative samples. As well as confirmatory flotation testing, the 2006 metallurgical work should investigating improving the concentrate grade.

The estimated cost of the proposed next phase work is as follows:

Metallurgical Testing	\$ 30,000
Mineral Estimation	\$ 50,000
<u>Surface Drilling</u>	<u>\$ 3,100,000</u>
Total	\$ 3,180,000

Item 23: References

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Montgomery Consultants Ltd. 1991 Geostatistical Study of the Geological Resource contained within the Polaris-Taku Gold Deposit by GH Giroux, P.Eng.

1995 Update of Resource Estimate for Polaris-Taku
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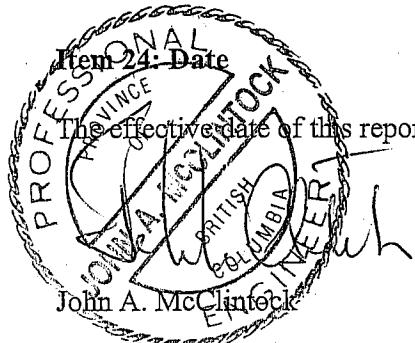
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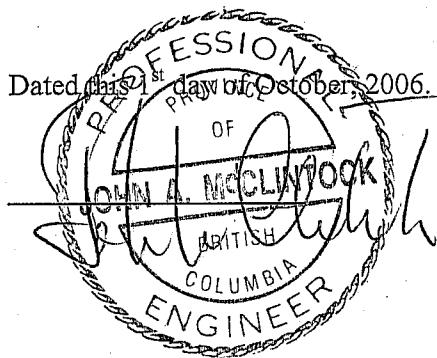


CERTIFICATE OF AUTHOR

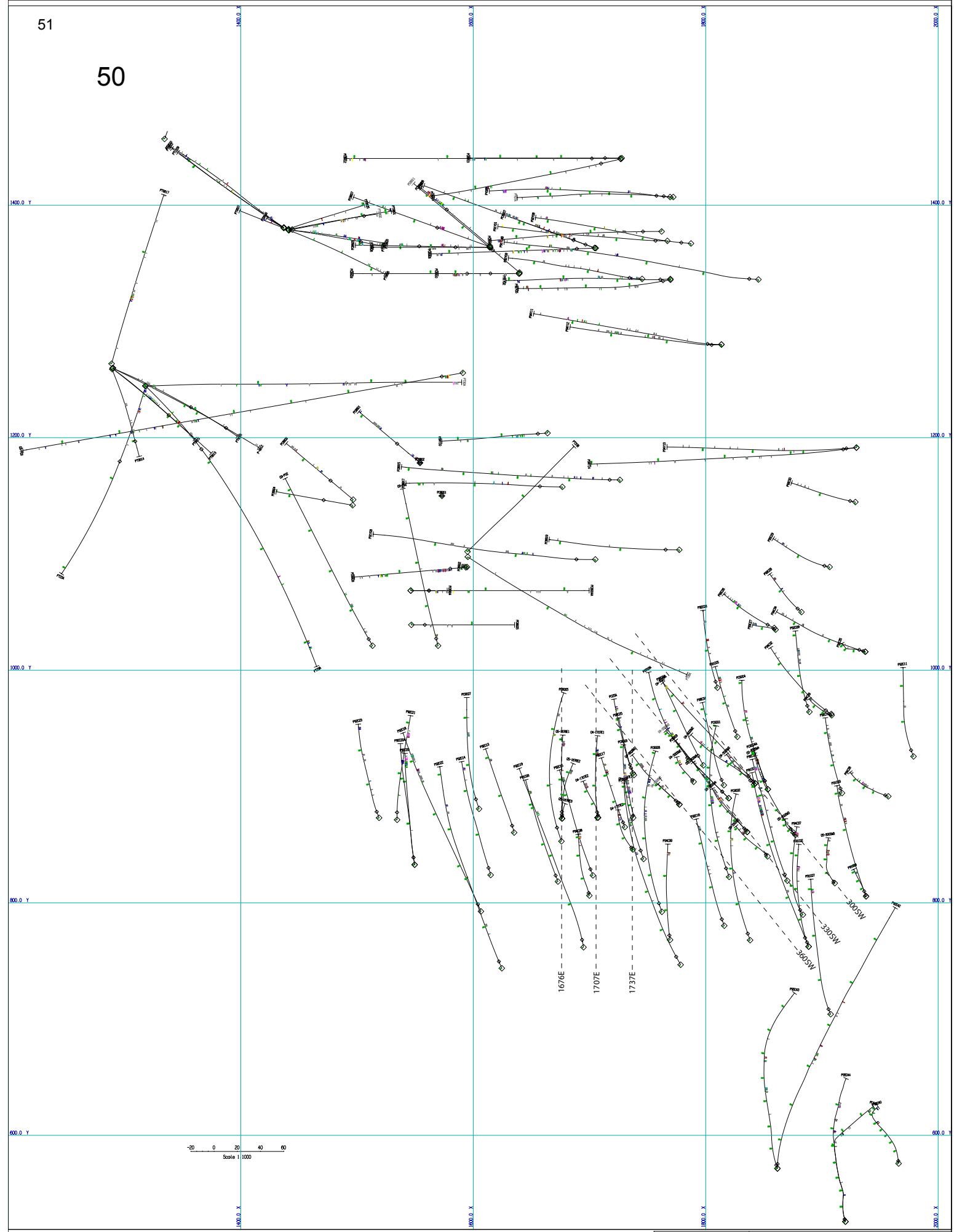
I, John A. McClintock do hereby certify that:

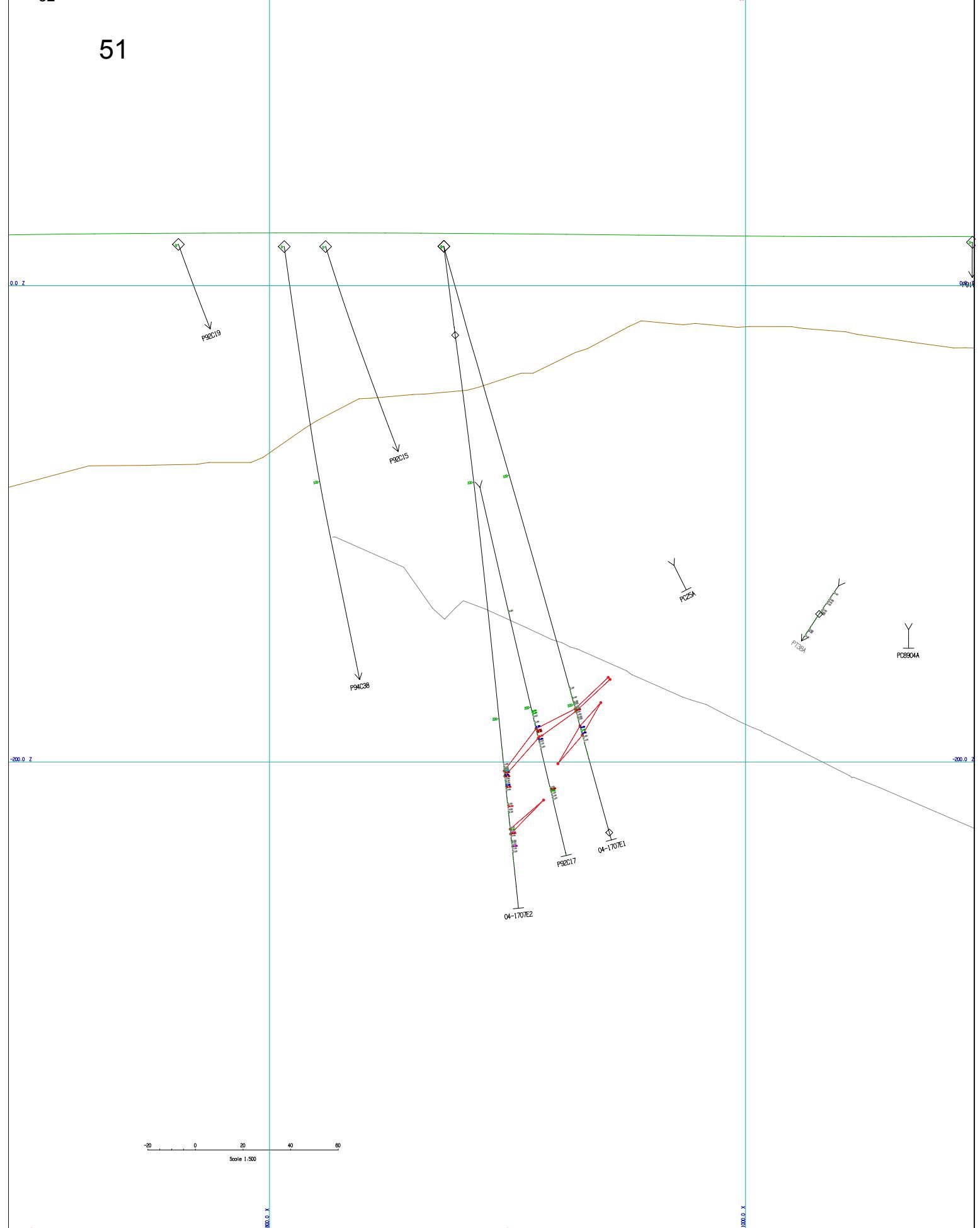
1. I am currently employed as President by:
Canarc Resource Corp.
Suite 800-850 West Hastings Street
Vancouver, BC
V6C 1E1

2. I graduated with a degree in BSc Hons from the University of British Columbia in 1973.
3. I am a member in good standing of the Association of Professional Engineers of British Columbia (No. 12078)
4. I have practiced my profession continuously for over 33 years and have examined and reported on numerous precious metal deposits throughout the world including northern British Columbia and Yukon..
5. I have read the definition of a "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirement to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of the technical report titled "Report on the 2003 to 2005 Exploration Program on the New Polaris Mine Site", dated October 1, 2006 (the "Technical Report"). The information contained in this Technical Report was obtained from reports provided by Canarc Resource Corp., various public documents and my visits to the property. I have visited the property on several occasions since May 2006, and since that time have spent an accumulative 7 days on site examining under ground workings and drill core.
7. I have had no prior involvement with the New Polaris Property.
8. As of the date of this certificate, to the best of the writer's knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am not independent of Canarc Resource Corp. as defined by National Instrument 43-101 as I am an insider of that corporation.
10. I have read National Instrument 43-101 and Form 43-101 F 1, and the technical report has been prepared in compliance with that instrument.



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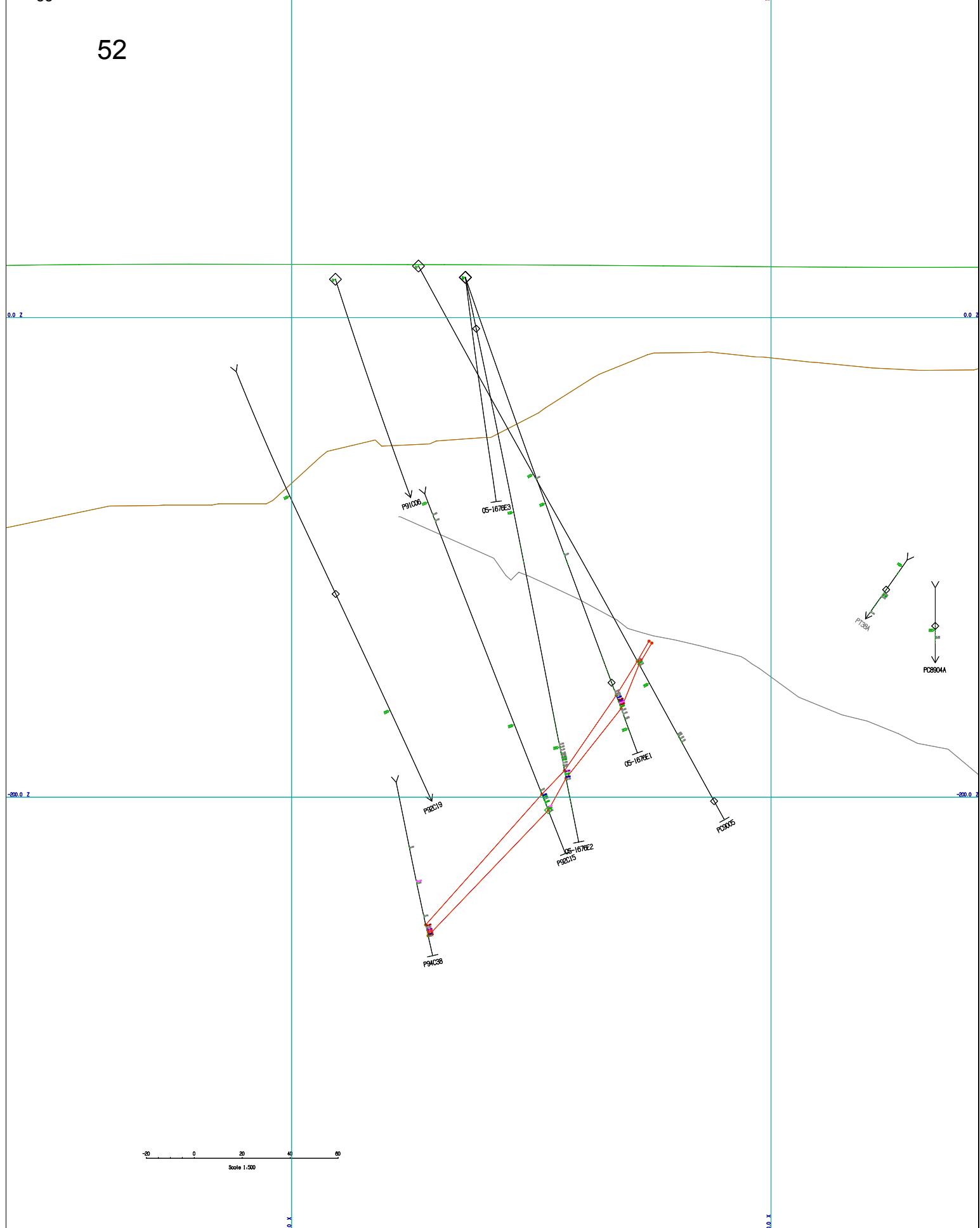
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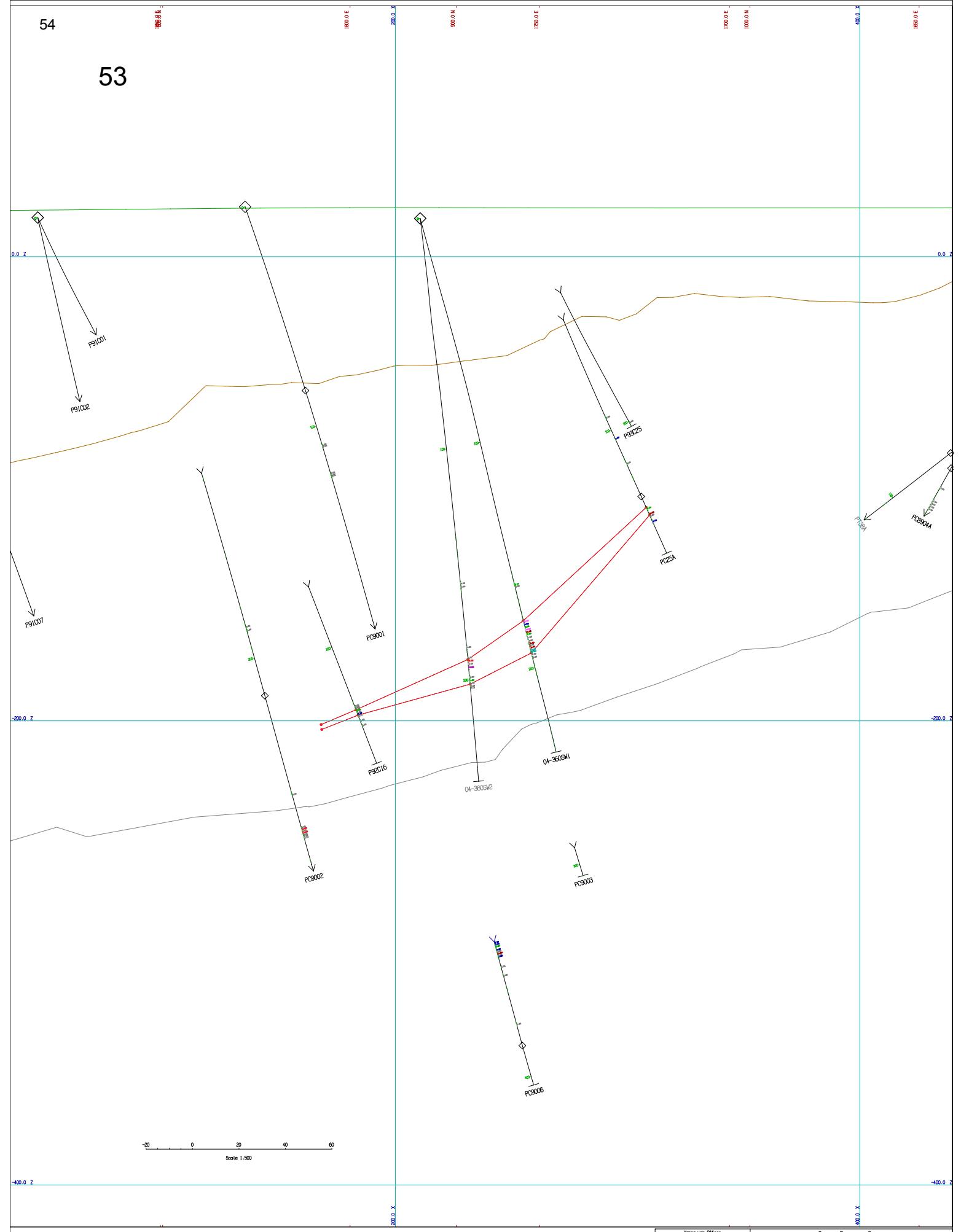
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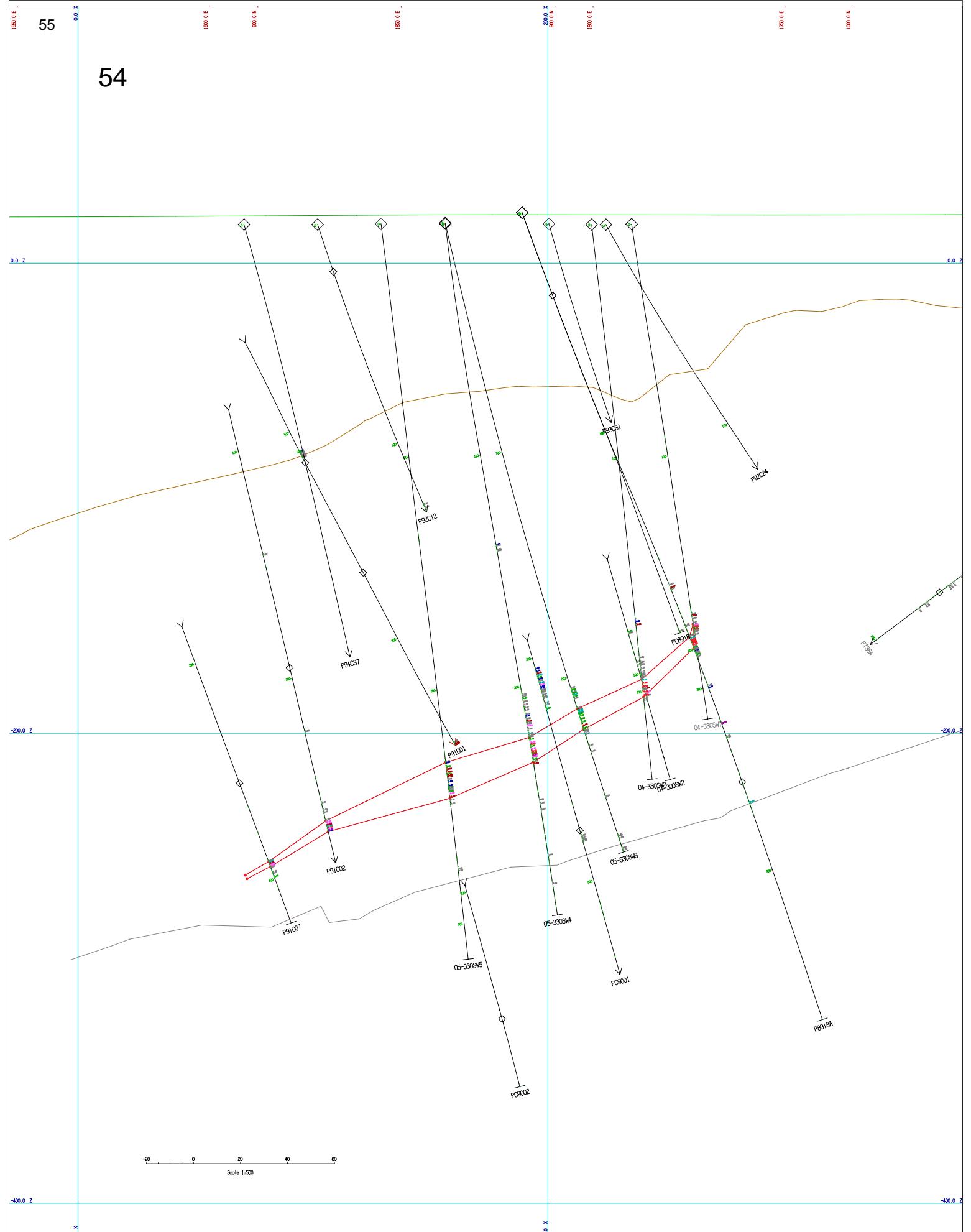
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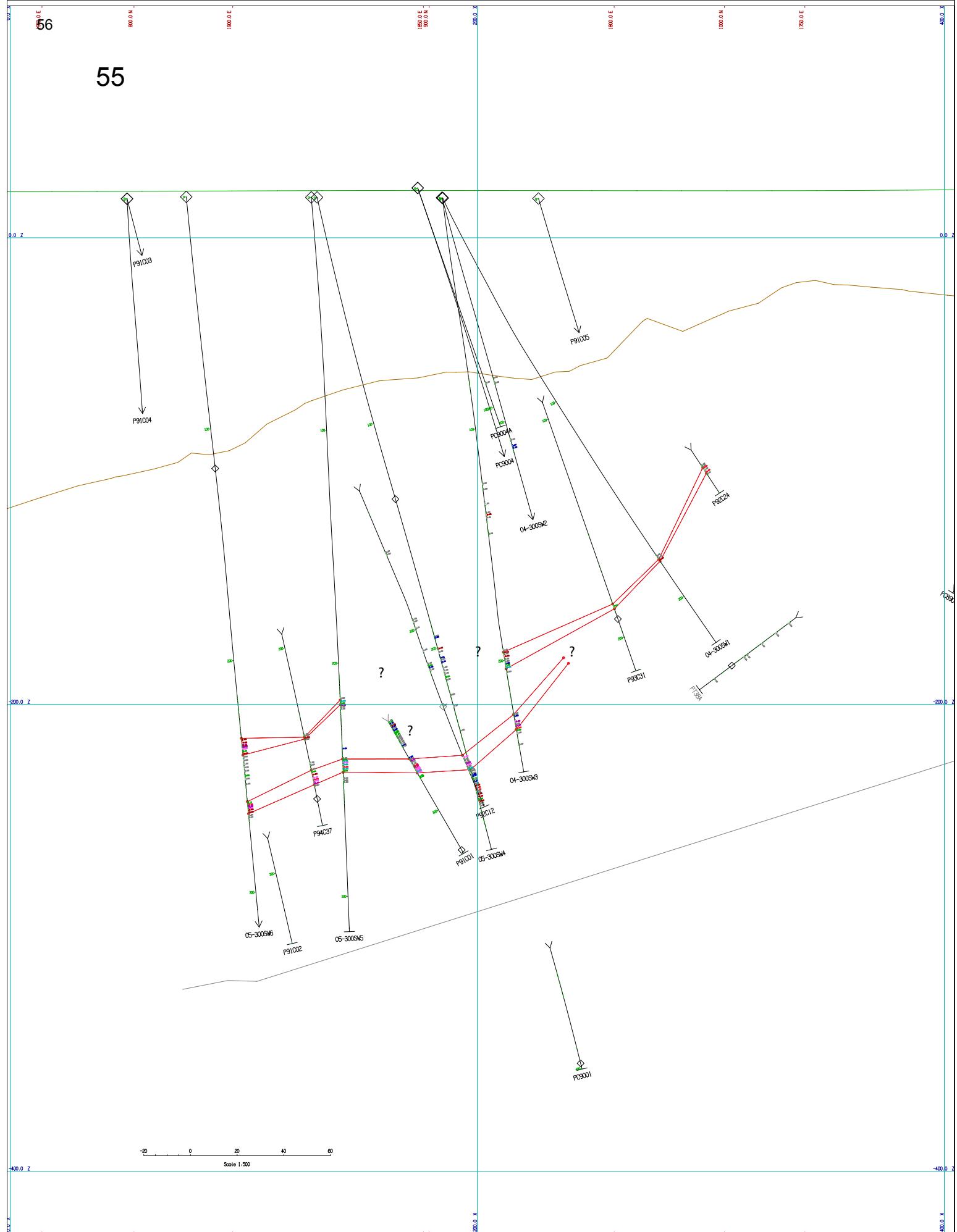
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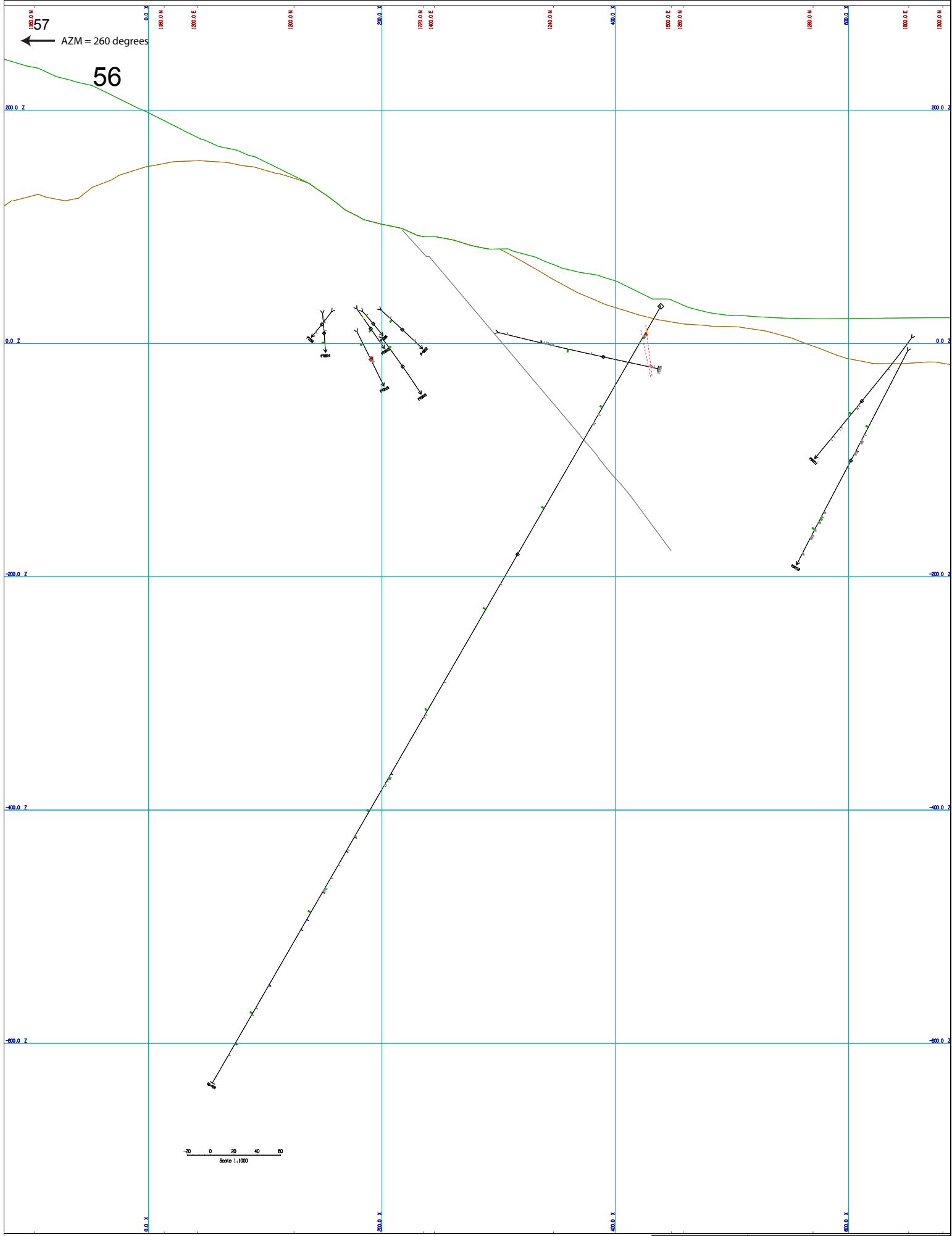






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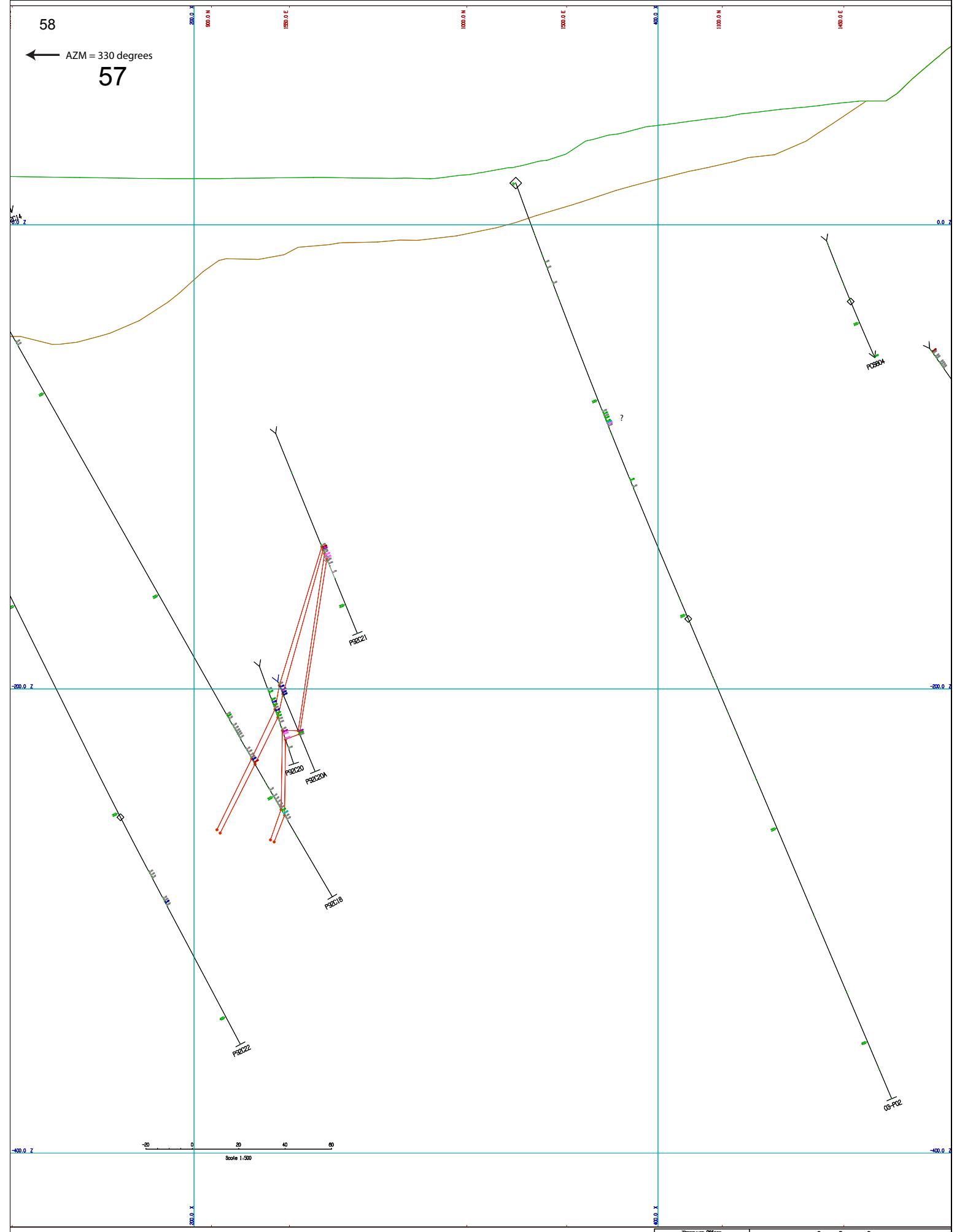


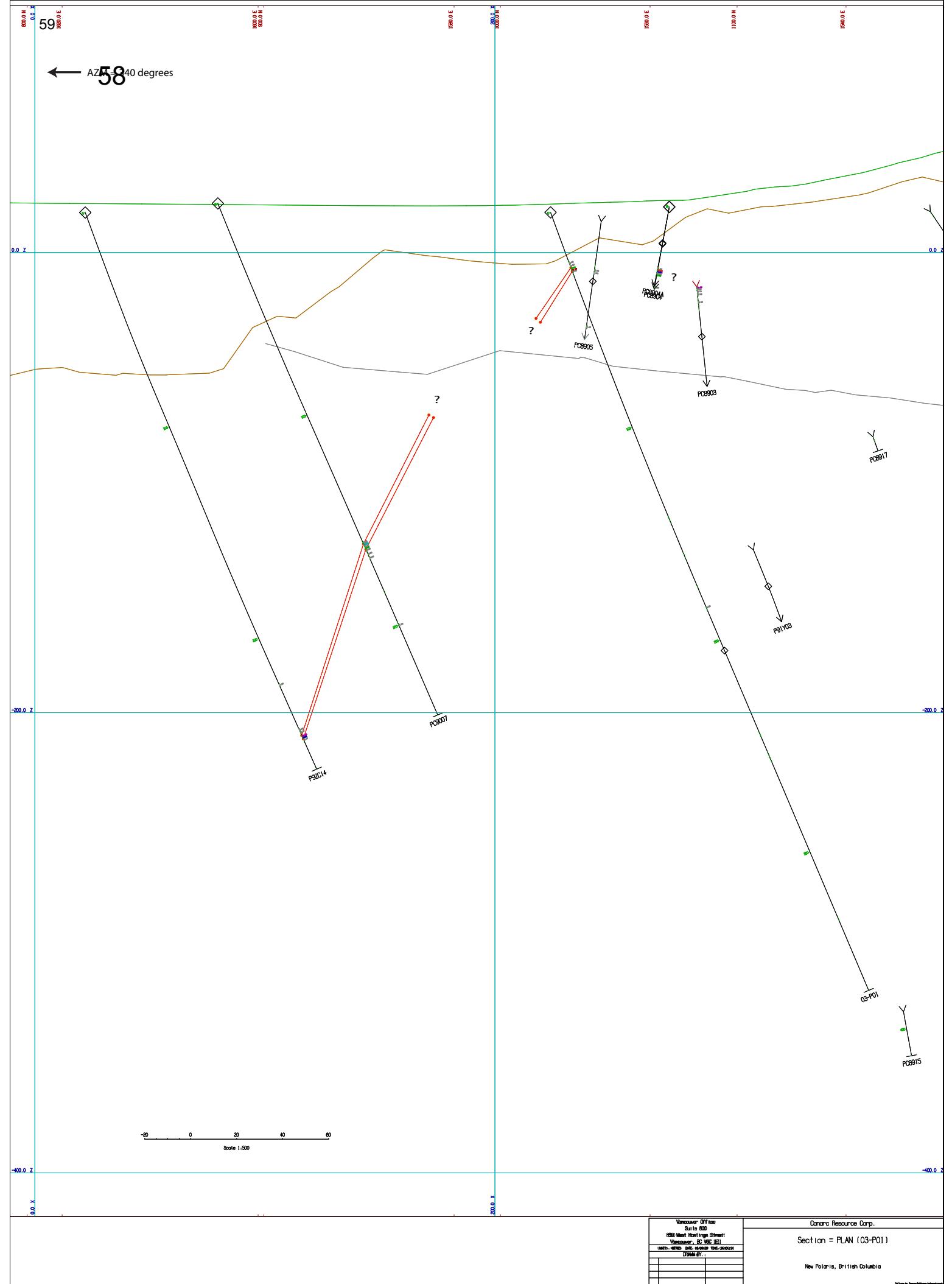


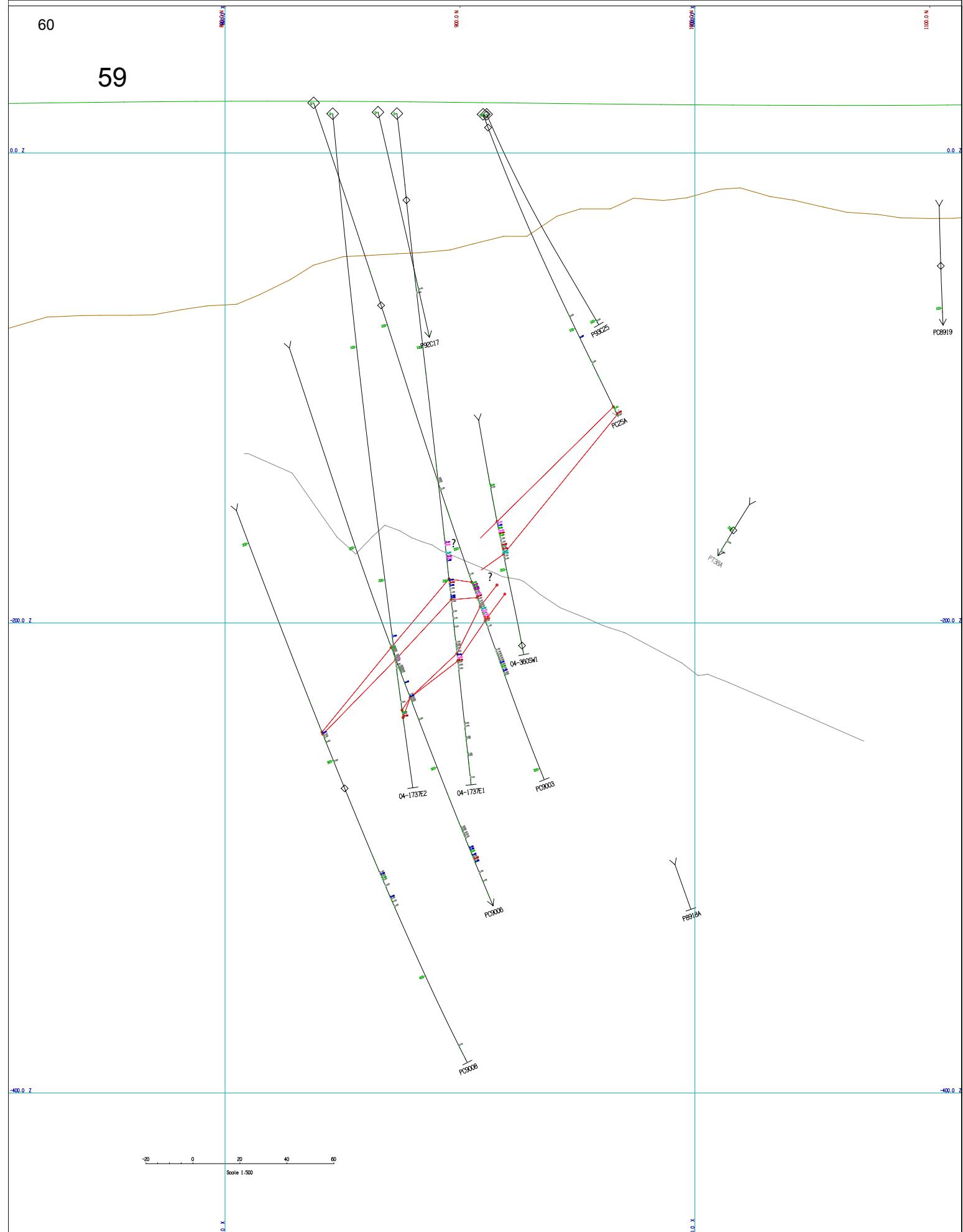
Vancouver Office Suite 600 600 West Georgia Street Vancouver, BC V6C 1E1 1-800-665-2222 (toll-free) 604-681-0777 (local)	Concorde Resource Corp. Section = Dynamic (O3-P03) New Polaris, British Columbia
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CANARC RESOURCE CORP.

METALLURGICAL TESTING OF
POLARIS TAKU SAMPLE



RESOURCE
DEVELOPMENT
INCORPORATED



Resource Development Inc.

January 14, 2003

Mr. Bradford J. Cooke
President and CEO
Canarc Resource Corp.
Suite 800-850 West Hastings Street
Vancouver, B.C. Canada V6C 1E1

RE: METALLURGICAL TESTING OF POLARIS TAKU PROJECT SAMPLES

Dear Mr. Cooke:

Resource Development Inc. (RDI) has completed the testwork outlined on an IOM dated August 30, 2002. This memorandum summarizes the test results and provides the cost estimate to generate concentrate for bio-oxidation studies at Bac Tech.

The results of diagnostic leach and preliminary flotation tests were conveyed in an IOM dated December 17, 2002. Additional cleaner flotation studies and leaching of rougher flotation tailing were undertaken.

The test results indicate the following:

- The rougher flotation process recovered 18.1% of weight, 93.8% of gold and 93.9% of arsenic in a concentrate assaying 94.24 g/T Au.
- The rougher plus one stage of cleaner flotation reduced the concentrate weight recovery to 11.4% and the gold recovery to 88.6%. The concentrate grade improved to 125 g/T Au.
- The rougher plus two stages of cleaner flotation reduced the overall gold recovery to 80.7% with slight reduction in concentrate weight to 10%. The concentrate grade did not improve at all (i.e., 125.9 g/T Au).
- Approximately 80% of the gold in the rougher flotation tailing was leached with sodium cyanide.
- There was insufficient cleaner 1 tailing sample available for leach test.
- We are waiting for carbonate analyses on products.

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These results indicate that it may be worthwhile to run one cleaner flotation if the concentrate is to be shipped from site. The cleaner tailings can be combined with rougher tailings and leached at site if economical to do so.

FUTURE WORK

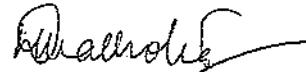
We plan to run large-scale tests to treat \pm 70 Kgs of ore to produce \pm 10 Kgs of rougher concentrate for bio-oxidation studies at Bactech/Mintek in South Africa. The samples will be shipped next week. The cost for this work will be \$5,500 plus cost of shipping concentrate to South Africa.

In addition we will run one large scale test to generate rougher concentrate for one cleaner flotation test. The cleaner concentrate will be analyzed by ICP for concentrate characterization. The cleaner tailing and rougher tailing will be leached with NaCN to determine additional gold extraction. The cost of this study will be \$2,200.

We can write the report of the work that has been completed to date or wait till the above work is completed. Please advise.

Looking forward to discussing the project with Leonard Harris and you on Thursday.

Sincerely,



Deepak Malhotra

cc: Leonard Harris



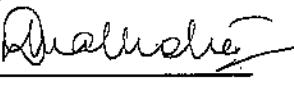
Resource Development Inc.

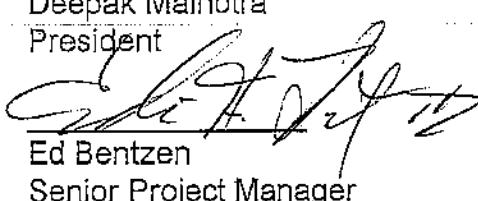
Prepared for:

Canarc Resource Corp.
Suite 800-850, West Hastings Street
Vancouver, B.C., Canada V6C 1E1

March 3, 2003

By:


Deepak Malhotra
President


Ed Bentzen
Senior Project Manager

Approved By:

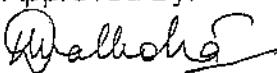

Dr. Deepak Malhotra
President

TABLE OF CONTENTS

	Page
INTRODUCTION	1
SUMMARY	2
RESULTS AND DISCUSSION	4
SAMPLE PREPARATION AND CHARACTERIZATION	4
DIAGNOSTIC LEACH	6
ROUGHER FLOTATION	9
ROUGHER CONCENTRATE PRODUCTION FOR UPGRADING STUDIES	10
CLEANER FLOTATION TESTS	11
GRAVITY CONCENTRATION	14
LEACH TESTS	15
ROUGHER CONCENTRATE PRODUCTION FOR BIO-OXIDATION STUDY	16

APPENDICES

- Appendix A: Feed Analyses
- Appendix B: Diagnostic Leach Data
- Appendix C: Grind Study Data
- Appendix D: Rougher Flotation Tests Data
- Appendix E: Cleaner Flotation Tests Data
- Appendix F: Gravity Concentration Tests Data
- Appendix G: Leach Tests Data

INTRODUCTION

Resource Development Inc. (RDI) undertook a metallurgical testwork on the bulk sample from Polaris Taku project at the request of Bradford J. Cooke, President and CEO of Canarc Resource Corporation, Canada. The property is located in British Columbia, Canada.

The metallurgical testwork was designed based on the review of the historical testwork done on the ore dating back to 1935. The metallurgical review indicated that the gold may be associated with arsenopyrite and was refractory to direct cyanidation process for gold extraction. Fine grinding of ore followed by flotation using simple reagent suite consisting of potassium amyl xanthate and pine oil recovered \pm 88% of gold in a concentrate having an upgrading ratio of \pm 4. The concentrate contained \pm 22% carbonates. The economics of pressure oxidation or bio-oxidation process could be improved by reducing the carbonate content in the flotation concentrate.

RDI received \pm 200 Kgs of bulk ore sample from Canarc Resource Corp. for the metallurgical testwork. The primary objectives of the study were to determine the deportation of gold in various minerals, reconfirm the flotation process conditions and produce a flotation concentrate containing maximum gold recovery and minimum amount of carbonates for bio-oxidation studies.

This report summarizes the test procedures and results obtained in the study.

¹"Review of Historical Metallurgy Data for Polaris Taku Project", RDI IOM dated August 30, 2002.

SUMMARY

Resource Development Inc. (RDI) undertook a metallurgical test program on the bulk sample from Polaris Taku deposit in British Columbia, Canada, for Canarc Resources Corporation. The primary objective of the study was to determine the deportation of gold in various minerals, reconfirm and/or optimize the flotation process conditions and produce a flotation concentrate containing maximum gold recovery and minimum amount of carbonates for bio-oxidation studies.

The highlights of the test program indicate the following:

- The composite sample assayed 19.41 g/T Au, 2.835% As, 127 ppm Sb, 1.92% S and <1.7 g/T Ag.
- The major minerals in the host rock were dolomite (32% to 35%), mica/illite (32%) and quartz (23%).
- Diagnostic leach test indicated that only 9% of the gold was free milling. Hence the ore is refractory. Approximately 4.5% of the gold was associated with stibnite and 66.8% of the gold was associated with arsenopyrite and pyrite. Since predominant sulfide bearing mineral is arsenopyrite and gold recovery follows arsenic recovery in flotation, it is reasonable to conclude that most of the gold is associated with arsenopyrite. About 20% of the gold was associated with quartz. Again, it could be due to fine gold containing arsenopyrite encapsulated in quartz.
- Base-case flotation test using potassium amyl xanthate (PAX) and methyl isobutyl carbonal (MIBC) recovered 83.6% of gold and 83.5% of arsenic in a concentrate assaying 73.63 g/T Au. The concentrate weight recovery was 20.7%.
- Modified flotation conditions using Na_2S in the grind and CuSO_4 in the flotation along with base-case reagents improved the flotation recovery and concentrate grade to 93.9% to 96.1% and ± 94 g/T Au, respectively.
- Cleaner flotation tests indicated that significant amount of carbonates in the rougher concentrate could be rejected in the first-cleaner flotation (78.6%) while losing only 3% of the gold. The carbonate content was reduced from 15% in the rougher concentrate to less than 1% in the first-cleaner concentrate. The concentrate weight recovery was also reduced from 18% to 15.27%.
- The second-cleaner concentrate did not reject sufficient gangue material but reduced gold recovery by additional 5%. Hence only one stage of cleaning is justified.

The gravity concentration did not preferentially upgrade rougher concentrate or recover additional gold values from rougher tailings.

- Leaching of rougher tailings resulted in 2% additional gold recovery. However, it is not economically justified because of high NaCN consumption (i.e., 3.88 Kg/T),
- Approximately 12 Kgs of rougher concentrate was shipped to Mintek for bio-oxidation studies. The gold recovery in the concentrate, assaying 91.98 g/T Au and 12.8% As, was 96.6%.

In summary, a flotation process consisting of roughers and one stage of cleaners can produce a concentrate assaying \pm 110 g/T Au and \pm 15.7% As with the weight and gold recoveries of 15.3% and 95.9%, respectively. The concentrate will contain less than 1% carbonate.

RESULTS AND DISCUSSION

Resource Development Inc. (RDI) received approximately 200 Kgs of bulk sample from Canarc Resource Corporation on October 9, 2002 for the testwork. The sample was packaged inside eight steel canisters.

The following sections discuss the sample preparation and characterization and the diagnostic and flotation testwork.

SAMPLE PREPARATION AND CHARACTERIZATION

The sample in the eight canisters was blended together and jaw crushed to pass nominal 3/8 inch. The crushed material was riffle split in two halves. One half was put in five gallon pails and stored for latter use. The other half was blended and crushed to pass 6 mesh. The minus 6-mesh material was split into 1-Kg charges using rotary splitter, bagged and stored in the freezer.

A 1-Kg sample was pulverized to 150 mesh, blended and a representative split was submitted for chemical analyses. The feed analyses data are given in Appendix A and the results are summarized in Tables 1 to 3. The highlights of the analytical results indicate the following:

- The sample assayed 19.41 g/T Au, 2.835% As and 127 ppm Sb. It contained less than 1.71 g/T Ag.
- The major minerals in the host rock are dolomite (32%), mica/illite (32%) and quartz (23%).
- The sample contained some pyrite besides arsenopyrite. The total sulfur content in the sample was 1.92%.

Table 1. Head Analyses of Polaris Taku Sample

Element	Assay		
	1	2	Average
Au, g/T	19.48	19.34	19.41
Ag, g/T	<1.71	<1.71	<1.71
As, %	2.860	2.810	2.835
Sb, ppm	125	128	127

Table 2. XRF Analyses of Polaris Taku Sample

Element	Assay, %	Element	Assay, ppm
Na ₂ O	<0.05	V	280
MgO	9.79	Cr	286
Al ₂ O ₃	13.7	Co	<50
SiO ₂	45.7	Ni	65
P ₂ O ₅	0.21	W	<50
S	1.92	Cu	169
Cl	<0.02	Zn	63
K ₂ O	2.78	As	250,000
CaO	14.1	Sn	61
TiO ₂	0.63	Pb	11
MnO	0.18	Mo	<10
Fe ₂ O ₃	9.80	Sr	392
BaO	0.02	U	<50
		Th	<50
		Nb	<50
		Zr	<50
		Rb	61
		Y	<50
		Sb	300

Table 3. XRD Analyses of Polaris Taku Sample

Mineral	Approximate Weight %
Dolomite	32
Mica/Illite	32
Quartz	23
Arsenopyrite	5
Pyrite	<5
K-feldspar	<3 (?)
Unidentified	<5

DIAGNOSTIC LEACH

The primary objective of the diagnostic leach is to determine the deportation of gold in various minerals. The process consists of a series of sequential cyanide leaches with intermediate acid leaches or roasting to preferentially breakdown specific minerals. The test procedure is given in Figure 1 and briefly discussed below:

- Cyanide leach of ore ground to 100% passing 150 mesh. This determined the free milling gold in the ore.
- Hydrochloric acid leach of residue to dissociate stibnite followed by cyanidation to determine gold associated with stibnite.
- Reducing roast at 425°C to break down arsenopyrite followed by cyanidation to determine gold associated with arsenopyrite.
- Oxidizing roast at 625°C to break down pyrite and remaining arsenopyrite followed by cyanidation to determine gold associated with pyrite/arsenopyrite.
- The gold remaining in the residue is associated with quartz.

The test data are given in Appendix B and the results are summarized in Table 4. The test results indicate the following:

- Approximately 9% of the gold is free milling in the sample.
- Approximately 4.5% of the gold is associated with stibnite.
- Approximately 66% to 67% of the gold is associated with arsenopyrite and pyrite. Since arsenopyrite is the predominant of the two minerals, it is reasonable to assume that most of the gold is associated with it rather than pyrite.
- A portion of the gold (~20%) is associated with quartz or with arsenopyrite which is encapsulated in quartz.

These results indicate that the major gold-bearing mineral is arsenopyrite. Some gold may be associated with pyrite, stibnite and quartz.

Table 4. Deportation of Gold in Polaris Taku Sample

Step	Gold Association	Extraction, %
Direct Cyanidation	Free milling	9.0
HCl Leach/Cyanidation	Stibnite	4.5
Reducing Roast/Cyanidation	Arsenopyrite	27.2
Oxide Roast/Cyanidation	Arsenopyrite/Pyrite	39.2
Residue	Quartz	20.1

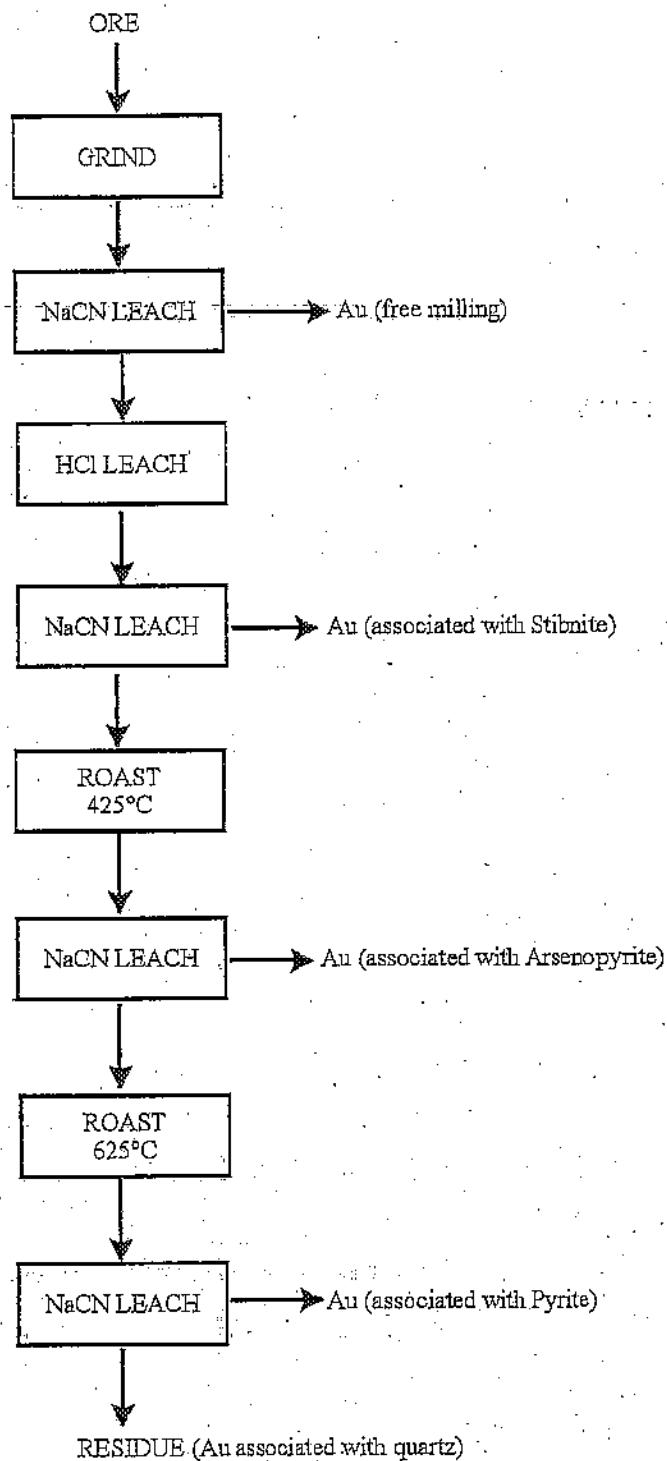


FIGURE 1. DIAGNOSTIC LEACH PROCEDURE

ROUGHER FLOTATION

Previous studies had indicated that fine grinding of the ore to 100% passing 200 mesh followed by flotation using simple reagent suite consisting of potassium amyl xanthate and pine oil recovered \pm 88% of the gold in a concentrate having an upgrading ratio of \pm 4.

The objectives of the present study were to reconfirm the process conditions, reduce carbonate content in the flotation concentrate and produce a flotation concentrate for bio-oxidation studies.

A grind study was undertaken to determine the relationship between grind time-grind size in the laboratory rod mill. A series of grind tests with varying time were performed with 1-Kg ore at 50% solids. The ground pulp was wet and dry screened to determine particle size distribution. The test data are given in Appendix C and the results are summarized in Table 5. The data indicated that 60 minutes of grind time was required to produce a product approximately 100% passing 200 mesh.

Table 5. Laboratory Grind Time - Particle Size Data

Grind Time, min	P ₈₀ , μm	% Passing 200 Mesh
20	262	49.1
30	106	62.5
40	63	88.0
50	51	96.8
60	45	98.9

A bench-scale flotation test was performed at a grind size of nominal 200 mesh using PAX and MIBC as flotation collector and frother, respectively. The flotation was performed at a natural pH of 8. Total flotation time was 15 minutes (Test No. 1). The test data are given in Appendix D and the results are summarized in Table 6. The flotation concentrate recovered 20.7% of the weight, 83.6% of gold and 83.5% of arsenic. The concentrate assayed 73.9 g/T Au and 9.10% As.

The reagent suite was modified in Test No. 2. Sodium sulfide (500 g/T) was added in the grind to sulfidize the ore and copper sulfate (250 g/T) was added after 10 minutes of flotation (Test No. 2). The test data, given in Appendix D and summarized in Table 6, indicated that flotation concentrate, assaying 94.24 g/T Au and 11.88% As, recovered 18.1% of weight, 93.8% of gold and 93.9% of arsenic.

Table 6. Summary of Rougher Flotation Test Results
(Flotation Time: 15 minutes; Grind: 100% 200 mesh)

Test No.	Process Conditions	Recovery, %			Concentrate Grade, g/T Au
		Wt.	Au	As	
1.	PAX/MIBC	20.7	83.6	83.5	73.63
2.	Na ₂ S/PAX/MIBC/CuSO ₄	18.1	93.8	93.9	94.24
3.	Lower pH/PAX/MIBC/CuSO ₄	21.0	87.0	86.1	78.08

A third rougher flotation test was performed where the flotation pulp was acidified with sulfuric acid (pH=5.8) and sulfide minerals floated with PAX and MIBC. Copper sulfate was added after 10 minutes of flotation time (Test No. 3). The test data, given in Appendix D and summarized in Table 6, indicated that flotation concentrate assaying 78.08 g/T Au and 10.05% As, recovered 21% of the weight, 87% of gold and 86.1% of arsenic. The results were not as good as obtained in Test No. 2.

ROUGHER CONCENTRATE PRODUCTION FOR UPGRADE STUDIES

Five bench-scale flotation tests were performed to generate rougher concentrate for upgrading studies using cleaner flotation and/or gravity concentrator. The objective of upgrading studies was to reject carbonates which negatively impact the oxidation of the concentrate by using more sulfuric acid.

The test procedure was slightly different from the previous three rougher tests. Initially the sulfide minerals were floated with PAX and MIBC. Sodium sulfide (285 g/T) was used after 10 minutes of flotation time. The test procedure and data are given in Appendix D. The test was designated as Test No. 4.

The rougher concentrate recovered 17.6% of weight, 90.1% of gold and 89.1 % of arsenic. The concentrate assayed 81.96 g/T Au, 10.85% As and 7.47% CO₂ (~15% carbonates).

The flotation gold recovery was slightly lower than Test No. 2 where Na₂S was added to mill and CuSO₄ was added after 10 minutes. However, the gold recovery was still significantly higher than the base-case test (No. 1) of 83.6%.

76
76

CLEANER FLOTATION TESTS

The rougher concentrate produced in Test No. 4 was split into three fractions. Two parts were used for cleaner flotation tests and one part for gravity concentration test.

In the first test, the rougher concentrate was cleaned for 10 minutes and the first cleaner concentrate was refloated for a total of 9 minutes (Test No. 5). PAX and MIBC were added in both first-and second-cleaner flotation. The test data are given in Appendix E and the results are summarized in Table 7. The test results indicate the following:

- The first-cleaner concentrate recovered 61.1% of weight, 93% of gold and 93% of arsenic. The concentrate assayed 123.3 g/T Au.
- When two stages of cleaner flotation were employed, the concentrate recovered 55.2% of weight, 85.9% of gold and 85.6% of arsenic. The concentrate assayed 125.9 g/T Au.
- The first stage of cleaner flotation rejected 84.4% of the carbonate in the rougher concentrate. The second-cleaner flotation rejected only 5.2% of additional carbonate. Hence the concentrate upgrading in second cleaner flotation was negligible (i.e., 125.9 versus 123.3 g/T Au).

Table 7. Summary of Two Stages of Cleaner Flotation Test Results
(Test No. 5)

Product	Flotation Time, min	Assay			Distribution, %			
		Au, g/T	% As	% Co ₂	Wt.	Au	As	Co ₂
Cleaner 2 Con 1	5	118.00	15.00	0.54	43.4	63.2	62.1	3.1
Cleaner 2 Con 2	4	155.14	20.80		11.8	22.7	23.5	7.3
Cleaner 2 Tail	—	98.68	13.10	6.56	5.9	7.1	7.4	5.2
Cal. Cleaner 1 Conc	10	123.3	15.94		61.1	93.0	93.0	15.6
Cleaner 1 Tail	—	14.54	1.89	16.20	38.9	7.0	7.0	84.4
Cal. Rougher Conc	—	81.00	10.47	7.47	100.0	100.0	100.0	100.0

An additional cleaner flotation test was performed and timed concentrates were collected (Test No. 6). The results, given in Appendix E and summarized in Table 8, confirm that significant amount of carbonate can be rejected (78.6%) with a small loss of gold to the cleaner tails (5.4%). Approximately 37.3% of the concentrate weight was rejected to the cleaner tailing.

**Table 8. Summary of One Stage of Cleaner Flotation Test Results
(Test No. 6)**

Product	Flotation Time, min	Assay			Distribution, %		
		Au, g/T	% As	% CO ₂	Wt.	Au	As
Cleaner 1 Con 1	1.0	122.02	16.10	1.10	42.1	62.0	63.2
Cleaner 1 Con 2	9.0	130.80	16.40	5.20	20.6	32.5	31.5
Cleaner 1 Tail	—	12.07	1.52	15.10	37.3	5.5	5.3
Cal. Rougher Conc	—	82.79	10.72	7.17	100.0	100.0	100.0

A cubic-foot rougher flotation test was undertaken to generate concentrate for cleaner flotation tests with and without regrind (Test No. 7, 8, 9). The test data are given in Appendix E and summarized in Tables 9 to 11. The rougher and cleaner tails were also leached. The results are discussed in section titled "LEACH TESTS".

Table 9. Summary of One Cubic Foot Rougher Flotation Test (Test No. 7)

Product	Flotation Time, min	Assay		Distribution, %		
		Au, g/T	% As	Wt.	Au	As
Rougher Conc	24	93.27	13.08	18.9	96.1	97.4
Rougher Tails	—	0.89	0.08	81.1	3.9	2.6
Cal. Feed	—	18.31	2.53	100.0	100.0	100.0

Table 10. Summary of Cleaner Flotation Test Without Regrind of Rougher Concentrate (Test No. 8)

Product	Flotation Time, min	Assay		Distribution, %		
		Au, g/T	% As	Wt.	Au	As
Cleaner Conc	10	108.78	15.7	80.8	97.2	97.9
Cleaner Tails	—	13.17	1.44	19.2	2.8	2.1
Cal. Rougher Conc	—	90.39	12.96	100.0	100.0	100.0

Table 11. Summary of Cleaner Flotation Test with Regrind of Rougher Concentrate (Test No. 9)

Product	Flotation Time, min.	Assay		Distribution, %		
		Au, g/T	% As	Wt.	Au	As
Cleaner Conc	8	116.26	15.60	46.7	56.5	55.2
Cleaner Tail	—	78.46	11.10	53.3	43.5	44.8
Cal. Rougher Conc		96.10	13.20	100.0	100.0	100.0

The highlights of the test results indicate the following:

- The rougher concentrate, assaying 93.27 g/T Au and 13.08% As, recovered 18.9% of weight, 96.1% of gold and 97.4% of arsenic.
- The rougher concentrate can be upgraded in a cleaner flotation without regrind to 108.78 g/T Au and 15.7% As. The concentrate weight recovery was 80.8% of rougher concentrate (15.27% of feed), 97.2% of gold in rougher concentrate (93.4% in feed) and 98.5% of arsenic in rougher concentrate (95.9% in feed).
- Regrinding of rougher concentrate is not beneficial (Tables 8 and 9).
- The first cleaner concentrate assayed 0.72% carbonate.
- The first cleaner concentrate analyses are given in Table 12.

Table 12. First Cleaner Concentrate Analyses

Element	Assay
Au, g/T	114.0
Ag, g/T	5.59
As, %	16.15
Cu, ppm	871
Fe, %	29.6
Hg, ppm	0.20
Pb, ppm	194
Zn, ppm	350
S _{Total}	25.88
S _{Sulfide}	25.63
Carbonate, %	0.72
Sb, ppm	750

78
79

GRAVITY CONCENTRATION

Rougher flotation concentrate and tailing were processed separately in a laboratory Knelson concentrator. The objective of the test was to determine the amenability of recovering additional gold values from rougher flotation tailing and the potential of gravity concentrator to upgrade the rougher concentrate.

The test data are given in Appendix F and the results are summarized in Tables 13 and 14. The test results indicate the following:

- Gravity concentrator did not recover additional gold values preferentially from rougher flotation tailing.
- Gravity concentrator also could not upgrade the concentrate.

Table 13. Gravity Concentration Test Data for Rougher Flotation Tailing

Product	Assay		Distribution, %		
	Au, g/T	% CO ₂	Wt.	Au	CO ₂
Knelson Concentrate	3.77	7.18	8.5	11.4	3.5
Knelson Tails	2.71	18.10	91.5	88.6	96.5
Cal. Feed	2.80	17.18	100.0	100.0	100.0

Table 14. Gravity Concentration Test Data for Rougher Flotation Concentrate

Product	Assay	Distribution, %	
	Au, g/T	Wt.	Au
Knelson Concentrate	100.34	9.7	12.0
Knelson Tails	79.22	90.3	88.0
Cal. Feed	81.27	100.0	100.0

LEACH TESTS

The rougher flotation tailing from Test No. 4 contained 9.9% of the gold in the feed and assayed 1.92 g/T Au. Bottle roll cyanide leach test was run for 48 hours at \pm 40% solids, pH 11 and 5 g/L NaCN.

The test data, given in Appendix G indicated that 23.3% of the gold in the tailing was leached. This is equivalent to 2% of additional gold. However, the NaCN consumption was high at 3.88 Kg/T. Hence it may not be economical to leach the rougher tailing.

Additional leach tests were performed on rougher tailing from Test No. 7 and Cleaner No. 1 tailing from Test No. 8. The test data are given in Appendix F and the results are summarized in Table 15. The test results confirm that it is not economical to leach the rougher or cleaner tailings because the gold extractions are low and the NaCN consumption is high.

Table 15. Leach Test Results for Rougher and Cleaner Tailing Samples

Sample	Cal. Head Grade, g/T Au	Extraction % Au	Residue Assay, g/T Au	NaCN Consumption Kg/T
Rougher Tailing	0.89	29.5	0.65	4.139
Cleaner Tailing	13.15	17.9	10.80	4.696

ROUGHER CONCENTRATE PRODUCTION FOR BIO-OXIDATION STUDY

Six one-cubic foot tests (10 Kg charges) were performed to generate rougher concentrate for bio-oxidation study (Test 10A to F). The test conditions are given in Appendix E and the overall recovery and concentrate weight were calculated based on feed, concentrate and tailing assays.

Rougher concentrate weighing over 12 Kgs was shipped to Mintek for bio-oxidation testwork.

The overall recovery and concentrate grade are given in Table 16.

Table 16. Large-Scale Flotation Test Results

Product	Assay			Distribution, %			
	Au, g/T	% As	% S	Wt.	Au	As	S
Feed	18.58	2.84	4.19	100.0	100.0	100.0	100.0
Concentrate	91.98	12.80	21.1	19.5	96.6	88.1	98.1
Tailing	0.78	(0.42)	0.10	80.5	3.4	11.9	1.9

⁸¹
82

APPENDIX A
FEED ANALYSES

82
83

Appendix A

Sample Description and Preparation:

Project No: 02-020
Date: 09-Oct-02

RDI Sample No: 1
Client's Identification: Composite
Date Received: 09-Oct-02
Sample Weight: Approximately 150-Kilograms
Sample Container: The samples were packaged inside eight steel cans.
The samples were shipped via CF Freightways, prepaid.
Sample Description: Rock samples ranging in size from 10-inches to fines.
Method of Preparation: The samples in each can blended together and jaw crushed to pass 3/8-inch. The crushed material was riffle split in-half. One half was bagged and set aside. The other half was blended and crushed to pass 6-mesh. The minus 6-mesh material was split into 1-kg samples and stored in plastic bags. A representative portion was further split, pulverized, split and a portion of the pulverized material submitted for chemical analyses.

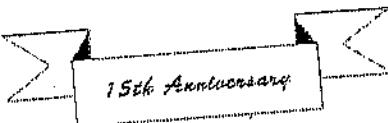
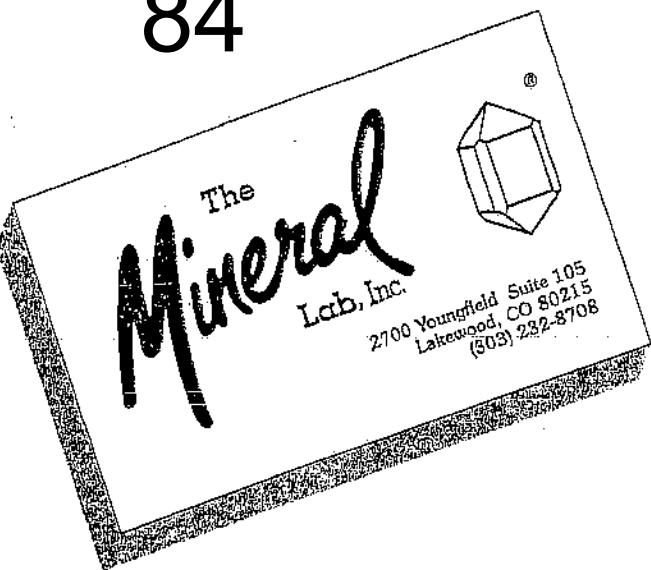
Chemical Analysis:

Au ¹ g/int	Ag ¹ g/int	As ppm	Sb ppm			
19.48	< 1.71	28,350	127			
SiO ₂ %	Fe ₂ O ₃ %	S %	As ppm	Cu ppm	Zn ppm	Pb ppm
(XRF) 45.1	9.8	1.90	24,500	169	63	11

1 Fire Assay

Gold, silver, total arsenic and antimony analyses by Florin Analytical Laboratory, Reno, Nevada.
XRF analyses by The Mineral Laboratory, Lakewood, Colorado.

83
84



October 31, 2002
Lab no. 202785

Mr. E. H. Bentzen III
Resource Development, Inc.
11475 West I-70 Frontage Road North
Wheat Ridge, Colorado 80033

Dear Mr. Bentzen:

Enclosed are the x-ray fluorescence (XRF) and the x-ray diffraction (XRD) results for sample, "Head Canarc" received last week. This report will be mailed and faxed to you, as usual.

A representative portion of the sample was ground to approximately -400 mesh in a steel swing mill and then analyzed by our "XRF-powder" procedure for the standard 31 major, minor and trace elements plus Sb. The relative precision/accuracy for this procedure is ~5-10% for major-minor elements and ~10-15% for trace elements (those elements listed in ppm) at levels greater than twice the detection limit in samples of average geologic composition. Detection limits are higher and analytical accuracy is poorer for this high As sample. A replicate sample and a standard reference material ("SY3", a CANMET standard rock) were analyzed with the sample to demonstrate analytical reproducibility for your sample and analytical accuracy for a geologic standard, respectively. The accepted ("known") values for the quality control standard are listed with the XRF results.

A representative portion of the ground sample was packed into a well-type plastic holder and then scanned with the diffractometer over the range, 3-61° 2θ using Cu-Kα radiation. The results of the scan are summarized as approximate mineral weight percents on the enclosed table. Estimates of mineral concentrations were made using our XRF-determined elemental composition and the relative peak heights/areas on the XRD scan. The detection limit for an average mineral in this sample is ~1-3% and the analytical reproducibility is approximately equal to the square root of the amount. "Unidentified" accounts for that portion of the XRD scan which could not be resolved and a "?" indicates doubt in both mineral identification and amount.

Thank you for the opportunity to be of continuing service to RDI.

Sincerely,

Peggy Dalheim
Peggy Dalheim

84
85

OCTOBER 31, 2002
LAB NO. 202785

SOURCE DEVELOPMENT, INC
XRF-POWDER RESULTS FOR SAMPLE, "CANARC" RECEIVED FROM ED BENTZEN

IDENT	Na2O	MgO	Al2O3	SiO2	P2O5	S	WT %	C1	K2O	CaO	TiO2	MnO	Fe2O3	BaO
CANARC	<0.05	9.79	13.7	45.7	0.21	1.92	<0.02	2.78	14.1	0.63	0.18	9.80	0.02	
QUALITY CONTROL - REPLICATE (R)	SAMPLE AND STANDARD REFERENCE MATERIAL (SY3) ANALYZED WITH SAMPLE													
CANARC (R)	<0.05	9.63	13.6	44.5	0.20	1.87	<0.02	2.75	13.9	0.62	0.18	9.72	0.01	
SY3-XRF	4.19	2.64	12.6	61.1	0.78	<0.05	<0.02	4.33	7.99	0.13	0.28	6.21	0.05	
SY3-known	4.15	2.67	11.8	59.7	0.54	0.05	0.014?	4.20	8.26	0.15	0.32	6.45	0.05	

IDENT	V	Cr	Co	Ni	W	Cu	PPM	Zn	As	Sn	Pb	Mo	Si	U
CANARC	280	286	<50	65	<50	169	63	25000	61	11	<10	392	<50	
QUALITY CONTROL	CANARC (R)													
CANARC (R)	278	281	<50	65	<50	168	63	24000	81	10	<10	384	<50	
SY3-XRF	48	<10	11	12	40	14	251	<20	<50	144	<10	307	621	
SY3-known	51	10	12	11	--	16	250	20	--	130	--	306	650	

IDENT	Th	Nb	Zr	Rb	Y	Sb	PPM							
CANARC	<50	<50	<50	61	<50	300								
QUALITY CONTROL	CANARC (R)													
CANARC (R)	<50	<50	<50	62	<50	300								
SY3-XRF	920	179	327	204	695	<50								
SY3-known	990	145	320	208	740	--								

ANALYSIS PERFORMED BY THE MINERAL LAB, INC

85
86

Resource Development, Inc.
XRD Results for "Head Canarc"

October 31, 2002
Lab no. 202785

Mineral Name	Chemical Formula	Approx. Wt %
Dolomite	$\text{Ca}(\text{Mg},\text{Fe})(\text{CO}_3)_2$	32
Mica/Illite	$(\text{K},\text{Na},\text{Ca})(\text{Al},\text{Mg},\text{Fe})_2(\text{Si},\text{Al})_4\text{O}_{10}(\text{OH},\text{F})_2$	32
Quartz	SiO_2	23
Arsenopyrite	FeAsS	5
Pyrite	FeS_2	<5
K-feldspar	KAISi_3O_8	<3?
"Unidentified"	?	<5

Analysis performed by The Mineral Lab, Inc

86
87

Florin Analytical Services, LLC

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Laboratory No.: 021560

Submitted By: Resource Development, Inc

Client Number: F174

Date: 11/14/02

Attention: Edwin H. Bentzen III

Method: 1 AT Fire assay, gravimetric finish

Lab code: 4002

Check

Check

Element:

Gold

Gold

Silver

Detection Limit (@ 1 AT):

0.10

0.10

1.71

Units:

g/t

g/t

g/t

Head Can Arc

19.48

<1.71

19.34

<1.71

Richard A. Grondin, QC Manager

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

87
88

Florin Analytical Services, LLC

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc

Laboratory No.: 021560

Client Number: F174

Date: 11/14/02

Attention: Edwin H. Bentzen III

Method: 2 Acid digestion, ICP analysis.

Lab code:	7002	7002	7047	7047
Element:	Arsenic	Arsenic QC	Antimony	Antimony QC
Detection Limit:	1	1	0.5	0.5
Units:	mg/kg	mg/kg	mg/kg	mg/kg

Head Can Arc 28600 28100 125 128

Richard A. Grondin, QC Manager

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

⁸⁸
89

APPENDIX B
DIAGNOSTIC LEACH DATA

Appendix B

Leaching Test 1

800

Permit:

Sample:

RDI Project:
Date: 02-02-82
23-Oct-82

To determine the distribution of leachable gold by diagnostic leaching procedures.

Approximately 1,000 grams of RDI Composite Sample 1.

Procedure:

A one kilogram sample of material was weighed in a rod mill to 80% passing 200-mesh and transferred to a rolling bottle container. The ground sample slurry was then adjusted to 40% solids and the pH adjusted to approximately 11 and leached with sodium cyanide at a level of approximately 2.0 g/l for 48 hours (Leach A). The residue from the first cyanide leaching was filtered, washed, re-filtered, dried and a thief sample removed for assay. The dried residue was slurried with water and hydrochloric acid was added until the pH stabilized at 2. Leaching continued for four hours after which the residue was filtered, washed, re-filtered, dried and a thief sample removed for assay. The dried residue from the hydrochloric acid leach was slurried to approximately 40% solids and the pH adjusted to approximately 11 and leached with sodium cyanide at a level of approximately 1.0 g/l for 48 hours (Leach C). The residue from the second cyanide leaching was filtered, washed, re-filtered, dried and a thief sample removed for assay. The dried residue was slurried to approximately 40% solids and the pH adjusted to approximately 11 and leached with sodium cyanide at a level of approximately 1.0 g/l for 48 hours (Leach D). The residue from the third cyanide leaching was filtered, washed, re-filtered, dried and a thief sample removed for assay. The dried residue was rosted in a muffle furnace at 625°C for 4 hours, cooled, and a thief sample removed for assay. The rosted material was slurried to approximately 40% solids and the pH adjusted to approximately 11 and leached with sodium cyanide at a level of approximately 1.0 g/l for 48 hours (Leach E). The residue from the fourth cyanide leaching was filtered, washed, re-filtered, dried and a thief sample removed for assay.

Conditions:	Grind	Leach Time	Leachant	% Solids	
	30% minus 200 M	48 hours	1.9 g/l NaCN	40% Solids	
		4 hours	1.9 g/l HCl	19% Solids	
		48 hours	0.99 g/l NaCN	40% Solids	
		4 hours	0.99 g/l NaCN	40% Solids	
		48 hours	1.00 g/l NaCN	40% Solids	

Summary of Results:

Parameter	NaCN #A	HCl #B	NaCN #C	NaCN #E	NaCN #G	Cumulative Extraction	
						Au	Au
Au Extraction, % (1)	9.0	—	4.7	31.5	66.1	82.2%	
Calculated Head, Au/gmt	18.20	16.57	27.20	25.18	26.98		
Astrial Head, Au/gmt	19.48	16.41	25.87	25.44	24.35		
Final Tail Assay, Au/gmt	16.41	25.87	25.67	20.36	7.13		
Cyanide Consumption	4,090	—	0.481	1,685	1,777	kg NaCN/metric ton feed	
Lime Added	1,164	—	1,425	2,627	2,392	kg CaO/metric ton feed	
Hydrochloric Acid Added	—	307.66	—	—	—	kg HCl/metric ton of feed	

Detailed Results:

A. Processing Conditions

Time hrs	Net Pulp Weight g	Net Sols Volume ml	Reagents Added, g			Residual Reagents NaCN p/l	pH	Temp °C
			NaCN	Ca(OH) ₂	HCl			
NaCN Leach, A	2505	1505	3.00	1.67		0.37 1.00 1.76	5.1 11.7 11.9	11.1
	2505	1505	2.22	0.00				
	2505	1505	1.50	0.00				
	2504	1494						
Acid Leach, B	2426	1460			307.66	— —	9.3 2.1	3.0
	3191	2574						
NaCN Leach, C	1426	857	0.85	124		0.60 1.04 1.08	5.5 4.42 3.80	11.1 11.0 10.6
	1426	857	0.34	0.64				
	1425	856	0.00	0.00				
	1425	849						
Roast 425, D	539					—	9.3	425
	528							
NaCN Leach, E	1237	738	0.73	2.21		0.20 0.68 1.00	4.0 9.2 10.0	11.1 11.1 11.1
	1227	738	0.59	0.71				
	1226	736	0.24	0.56				
	1225	737						
Roast 625, F	446					—	9.3	625
	421							
NaCN Leach, G	963	578	0.58	2.03		0.36 0.90 0.64	3.9 9.6 9.6	11.0 11.2 11.1
	962	578	0.37	0.61				
	953	608	0.12	0.52				
	991	608						
			10.54	10.18	307.66			
			(0.7)	CaO Equivalent				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/t	Thief g/ml
Feed (unanalyzed)	1,000		19.46	
Feed (computed)			18.20	
NaCN Leach, A Solution		1,494		
Residue (intermediate)	1,010		16.41	1.08
Acid Leach, B Solution	960.5	2574	0.62	48.3
Residue (intermediate)	617.0		25.87	
NaCN Leach, C Solution	568.7	849	0.84	30
Residue (intermediate)	574.4		25.87	
Roast 425, D Erod	539		25.44	51.5
Residue	527.9		25.44	
NaCN Leach, E Solution	489.1	737	3.33	30
Residue (intermediate)	488.6		20.36	48.3
Roast 625, F Feed	447.6		21.44	48.9
Residue	420.7			
NaCN Leach, G Solution	384.4	608	6.24	30
Residue (final)	383.1		7.13	

(1) Based on calculated head assays.

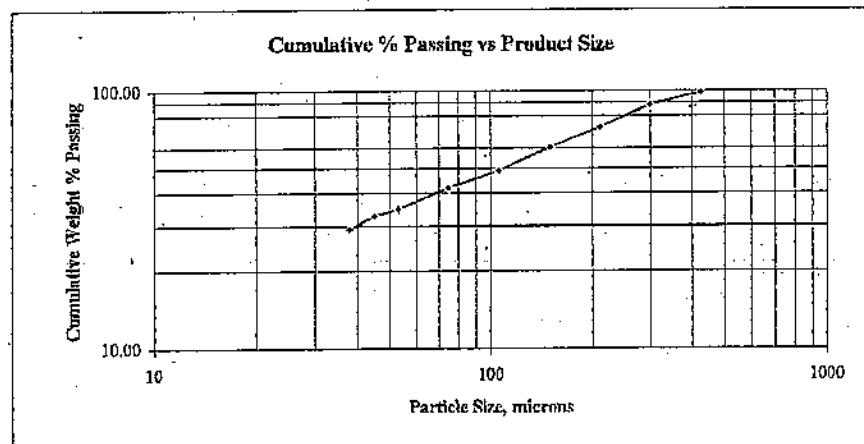
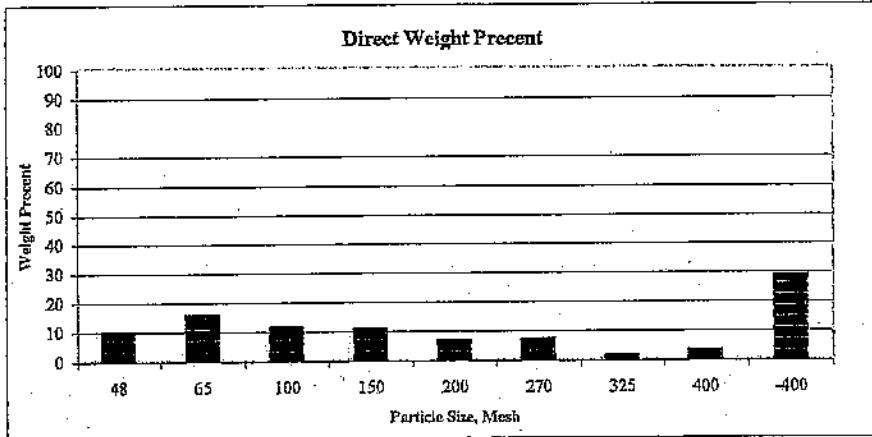
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91

APPENDIX C
GRIND STUDY DATA

Project: 02-020
Date: 16-Oct-02**Purpose:** To determine the particle size distribution of the sample after rod mill grinding.**Sample:** Approximately 1,000 g of RDI Composite Sample 1.**Procedure:** The sample was rod mill ground at 50% solids for 20-minutes. The slurry was then wet screened at 400-mesh and-dried. The plus 400-mesh fraction was dry screened (Ro-Tap 20-min) at the screen sizes shown below. The screen fractions were weighed and the particle size distribution calculated.**Results:**

Mesh (Retained)	Microns	Weight			Cumulative Retained
		Grams	%	Passing	
Feed (Calculated)		1001.1	100.0		
35	425	14.8	1.48	98.52	1.48
48	300	103.4	10.33	88.19	11.81
65	212	160.1	15.99	72.20	27.80
100	150	118.6	11.85	60.35	39.65
150	106	112.7	11.26	49.10	50.90
200	75	70.3	7.02	42.07	57.93
270	53	74.0	7.39	34.68	65.32
325	45	20.3	2.03	32.65	67.35
400	38	34.0	3.40	29.26	70.74
-400	-38	292.9	29.26	0.00	100.00

Product Size Passing 80% (F80): = 262 Microns



92
93

Appendix C

Particle Size Analysis 2

Project: 02-020
 Date: 16-Oct-02

Purpose: To determine the particle size distribution of the sample after rod mill grinding.

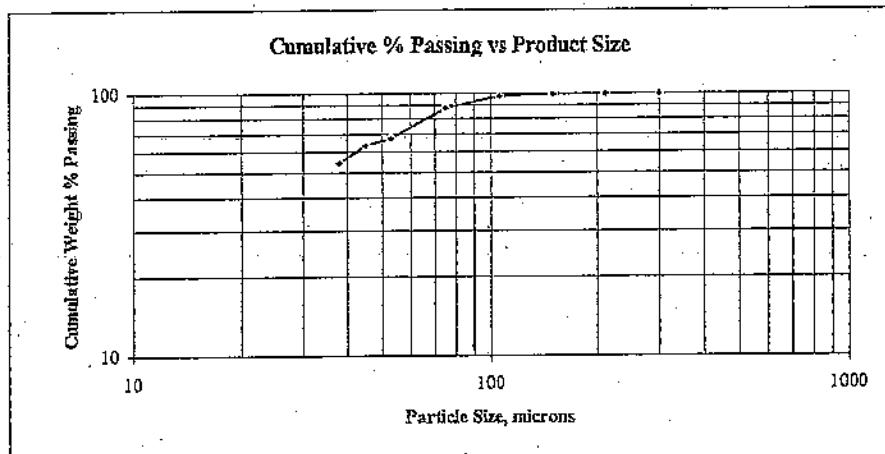
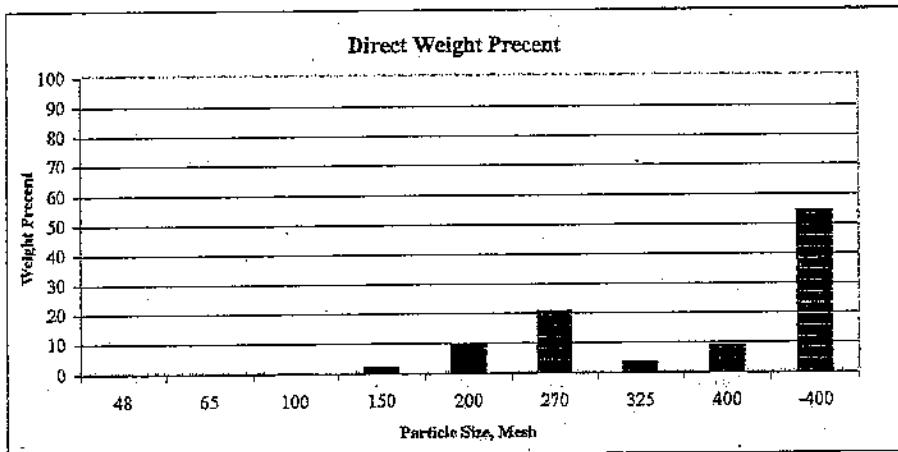
Sample: Approximately 1,000 g of RDI Composite Sample 1.

Procedure: The sample was rod mill ground at 50% solids for 40-minutes. The slurry was then wet screened at 400-mesh and dried. The plus 400-mesh fraction was dry screened (Ro-Tap 20-min) at the screen sizes shown below. The screen fractions were weighed and the particle size distribution calculated.

Results:

Product Mesh (Retained)	Microns	Weight			Cumulative Retained
		Grams	%	Passing	
Feed (Calculated)		1002.1	100.0		
48	300	0.1	0.01	99.99	0.01
65	212	0.4	0.04	99.95	0.05
100	150	2.0	0.20	99.75	0.25
150	106	21.5	2.15	97.61	2.39
200	75	96.2	9.60	88.01	11.99
270	53	207.7	20.73	67.28	32.72
325	45	38.3	3.82	63.46	36.54
400	38	89.6	8.94	54.52	45.48
-400	-38	546.3	54.52	0.00	100.00

Product Size Passing 80% (F80): = 63 Microns



93
94

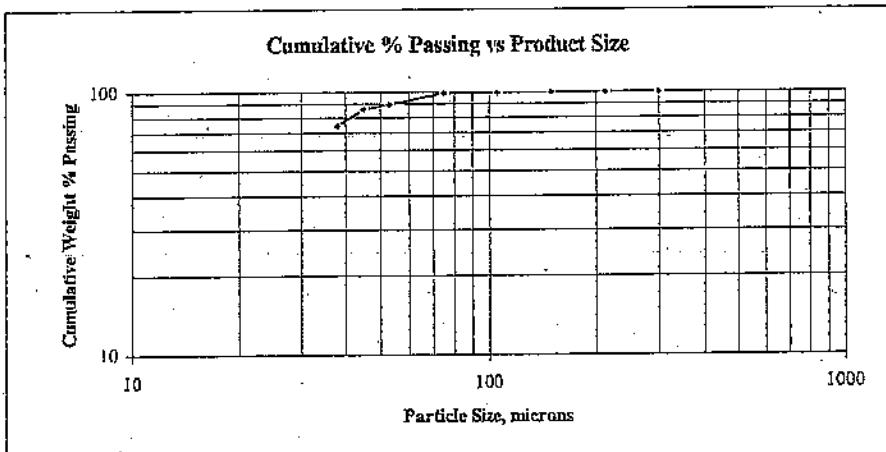
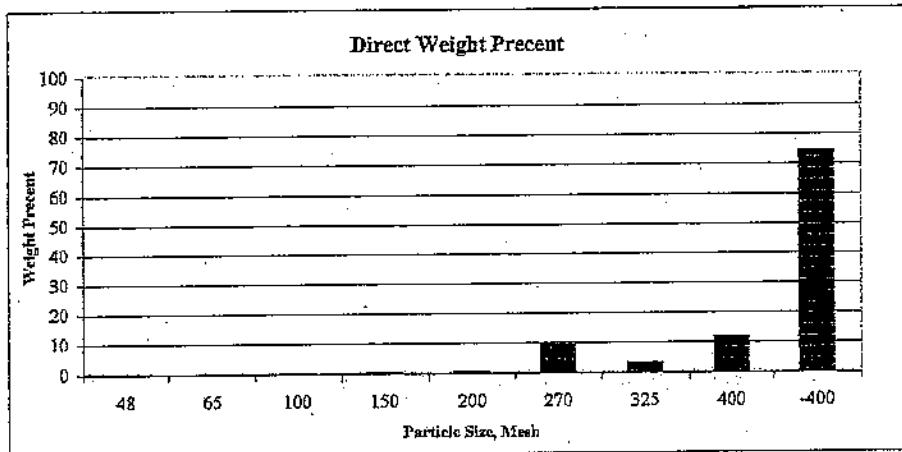
Appendix C

Particle Size Analysis 3

Project:
Date:02-020
16-Oct-02**Purpose:** To determine the particle size distribution of the sample after rod mill grinding.**Sample:** Approximately 1,000 g of RDI Composite Sample 1.**Procedure:** The sample was rod mill ground at 50% solids for 60-minutes. The slurry was then wet screened at 400-mesh and dried. The plus 400-mesh fraction was dry screened (Ro-Tap 20-min) at the screen sizes shown below. The screen fractions were weighed and the particle size distribution calculated.**Results:**

Mesh (Retained)	Microns	Weight			Cumulative Retained
		Grams	%	Passing	
Feed (Calculated)		1003.7	100.0		
48	300	0.0	0.00	100.00	0.00
65	212	0.3	0.03	99.97	0.03
100	150	0.9	0.09	99.88	0.12
150	106	3.2	0.32	99.56	0.44
200	75	7.1	0.71	98.85	1.15
270	53	93.0	9.27	89.59	10.41
325	45	34.0	3.39	86.20	13.80
400	38	118.0	11.76	74.44	25.56
-400	-38	747.2	74.44		

Product Size Passing 80% (F80): = 45 Microns



94
95Appendix C
Particle Size Analysis 4Project:
Date:

02-020

Purpose: To determine the particle size distribution of the sample after rod mill grinding.

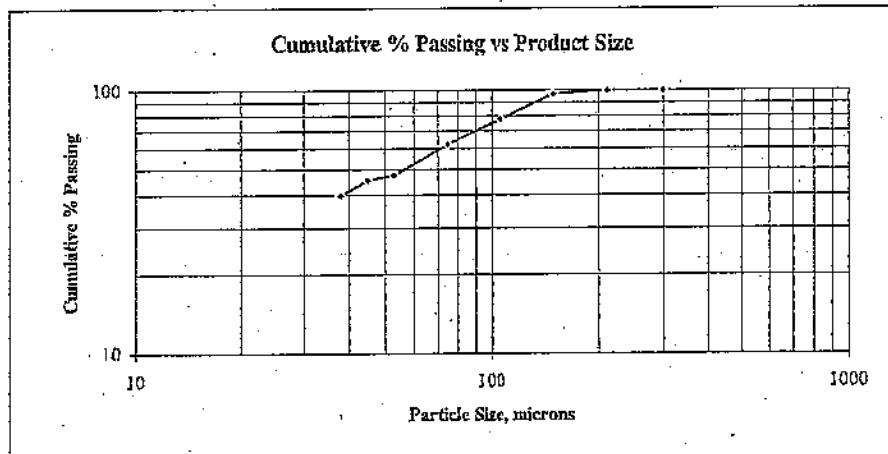
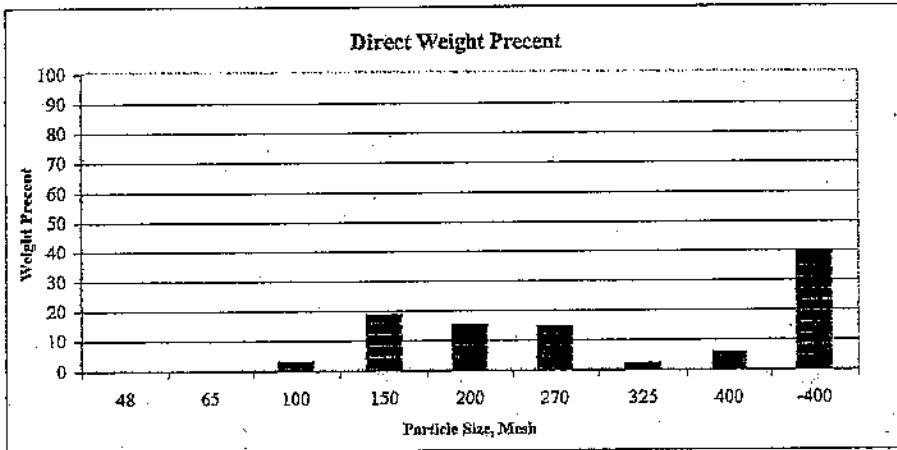
Sample: Approximately 1,000 g of RDI Composite Sample 1.

Procedure: The sample was rod mill ground at 50% solids for 30-minutes. The slurry was then wet screened at 400-mesh and dried. The plus 400-mesh fraction was dry screened (Ro-Tap 20-min) at the screen sizes shown below. The screen fractions were weighed and the particle size distribution calculated.

Results:

Product Mesh (Retained)	Microns	Weight			Cumulative Retained
		Grams	%	Cumulative Passing	
Feed (Calculated)		1000.4	100.0		
48	300	0.1	0.01	99.99	0.01
65	212	2.2	0.22	99.77	0.23
100	150	31.4	3.14	96.63	3.37
150	106	188.2	18.81	77.82	22.18
200	75	153.6	15.35	62.47	37.53
270	53	146.7	14.66	47.80	52.20
325	45	22.9	2.29	45.51	54.49
400	38	56.7	5.67	39.84	60.16
-400	-38	398.6	39.84	0.00	100.00

Product Size Passing 80% (F80): = 101 Microns



95
96

Appendix C

Particle Size Analysis 5

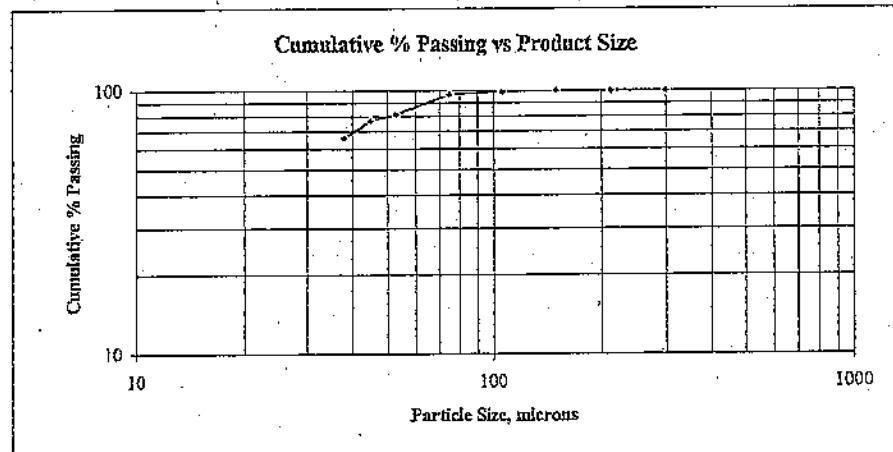
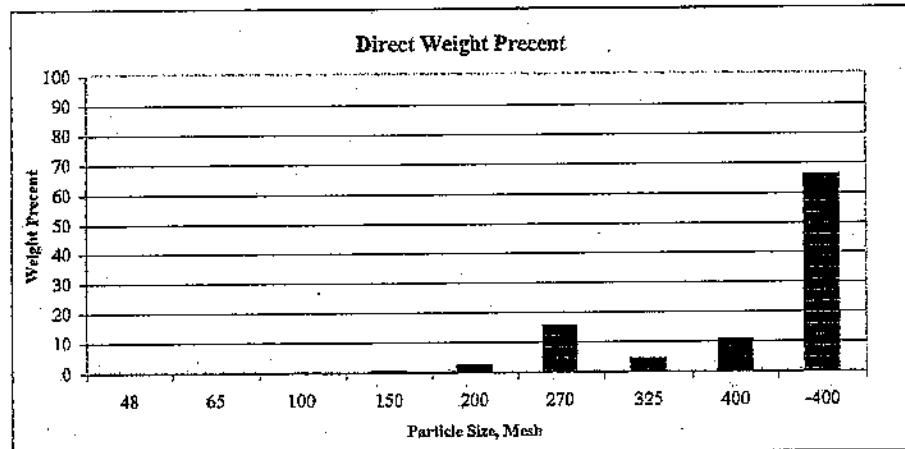
Project:
Date:

02-020

Purpose: To determine the particle size distribution of the sample after rod mill grinding.**Sample:** Approximately 1,000 g of RDI Composite Sample 1.**Procedure:** The sample was rod mill ground at 50% solids for 30-minutes. The slurry was then wet screened at 400-mesh and dried. The plus 400-mesh fraction was dry screened (Ro-Tap 20-min) at the screen sizes shown below. The screen fractions were weighed and the particle size distribution calculated.**Results:**

Product (Retained)	Mesh	Microns	Weight		
			Grams	%	Cumulative Passing
Feed (Calculated)			1004.6	100.0	
	48	300	0.0	0.00	100.00
	65	212	0.3	0.03	99.97
	100	150	1.1	0.11	99.86
	150	106	6.0	0.60	99.26
	200	75	24.6	2.45	96.81
	270	53	156.4	15.57	81.25
	325	45	43.7	4.35	76.90
	400	38	108.0	10.75	66.15
	-400	-38	664.5	66.15	0.00
					100.00

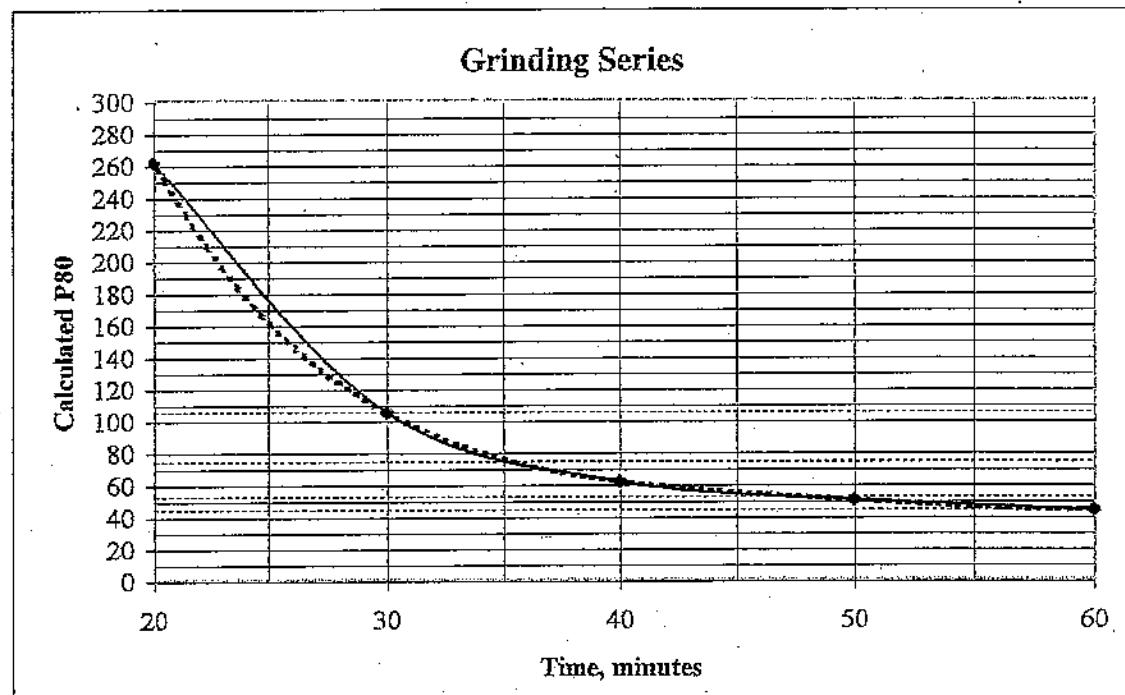
Product Size Passing 80% (F80): = 51 Microns



96
97

Appendix C

Screen	Microns	Cumulative Percent Passing				
		20	30	40	50	60
48	300	98.5	99.99	100.0	100.00	100.0
65	212	88.2	99.77	100.0	99.97	100.0
100	150	72.2	96.63	99.8	99.86	99.9
150	106	60.4	77.82	97.6	99.26	99.6
200	75	49.1	62.47	88.0	96.81	98.9
270	53	42.1	47.80	67.3	81.25	89.6
325	45	34.7	45.51	63.5	76.90	86.2
400	38	32.7	39.84	54.5	66.15	74.4
P ₈₀		262	106	63	51	45



Equation Y = 1522.499655 - 115.3999042 * X + 3.453866672 * pow(X,2) - 0.04639025512 * pow(X,3) + 0.0002340177227 * pow(X,4)
R-squared = 1

Mesh	P80	Time
150	106	30
200	75	35
270	53	48
325	45	60

⁹⁷
98

APPENDIX D
ROUGHER FLOTATION TEST DATA

Appendix D

Flotation Test No. 1

Purpose:

Preliminary flotation test to recover gold by froth flotation.

Sample: Approximately 1,000 grams of the RDJ Composite Sample I.

Procedure:

Operations	min.	Conditions			Reagents, g/mt of flotation feed		
		Solids %	pH-Start	pH-End	PAX	MIBC	
Rod Mill Grind	60	50	7.9				
Conditioning	1.0	26	8.0	8.1	100		20
Ro Flotation -1	5.0	26	8.1	8.3			
Conditioning	5.0	23	8.3	8.3	50		15
Ro Flotation -2	5.0	23	8.3	8.3			
Conditioning	1.0	22	8.3	8.3	50		15
Ro Flotation -3	5.0	22	8.3	8.2			

Total Reagent Used, g/mt of feed
Solution Concentration

200
1%
0.005

Results: Products	Weight g	Chemical Analysis			Percent Distribution	
		%	Au g/mt	As %	Au	As
Feed (analyzed)	1,000.0		19.48	2.84		
Feed (calculated)	1,011.0	100.0	18.22	2.26		
Ro-Froth 1	127.9	12.7	83.78	10.35		
Ro-Froth 2	49.3	4.9	65.75	8.01		
Ro-Froth 3	31.9	3.2	45.11	5.78		
Ro-Tail 3	801.9	79.3	3.77	0.47		
					100.0	100.0

Observations:

Grind in Steel Mill.
Froth in a 3.2 liter cell at 1600 RPM.

Rougher Tailing 1	883.1	87.35	8.72	1.08	41.3	41.9
Rougher Tailing 2	833.8	82.47	5.35	0.67	24.2	24.6
Ro-Froth 1, 2, 3	209.1	20.7	73.63	9.10	83.6	83.5

Appendix D

Rotation Test No. 2

RDI Project No.: 02-020
Date:

99
100

Purpose:

Preliminary flotation test to recover gold by froth flotation. Na_2S was added to the grinding mill.

Sample:

Approximately 1,000 grams of the RDI Composite Sample 1.

Procedure:

Operations	Conditions				Reagents, g/mt of flotation feed			
	min.	Solids %	pH-Start	pH-End	PAX	Na_2S	Cu_2SO_4	MIBC
Rod Mill Grind	60	50	8.6	8.8		100	500	20
Conditioning	1.0	26	8.8	8.9				
Ro Flotation - 1	5.0	26	8.9	8.9				
Conditioning	5.0	23	8.9	8.9		50		15
Ro Flotation - 2	5.0	23	8.9	8.8				
Conditioning	3.0	23	8.8	9.0		30		
Ro Flotation - 3	5.0	23	9.0	8.4				

Total Reagent Used, g/mt of feed
Solution Concentration

Results:

Products	Weight				Chemical Analysis				Percent Distribution			
	g	%	Au g/mt	Ag g/mt	As % g/mt	Au	Ag	As	Au	Ag	As	
Feed (analyzed)	1,000.0		19.48		2.84							
Feed (calculated)	1,009.8	100.0	18.22	12.67	2.29							
Ro-Froth 1	143.1	14.2	112.89	46.83	14.30				100.0	100.0	100.0	
Ro-Froth 2	18.0	1.8	32.92	33.90	3.84							
Ro-Froth 3	22.1	2.2	23.46	22.73	2.73							
Ro-Tail 3	826.6	81.9	1.37	.6.02	.017							

Observations:

Grind in Steel Mill.
Flotation in a 3.2 filter cell at 1600 RPM

Rougher Tailing 1
Rougher Tailing 2
Ro-Froth 1, 2, 3

866.7	85.83	2.59	7.03	0.31	12.2	47.6	11.7
848.7	84.05	1.95	6.46	0.24	9.0	42.8	8.7
183.2	18.1	94.24	42.65	11.88	93.8	61.1	93.9

Appendix D

Flotation Test No. 3

Purpose:

Preliminary Flotation test to recover gold by froth flotation. Conditioning pH reduced with sulfuric acid.
Copper Sulfate added to third Condition/Flootation stage.

Sample:

Approximately 1,000 grams of the RDI Composite Sample 1.

Procedure:

Operations	Conditions				Reagents, g/mt of flotation feed			
	min.	Solids %	pH-Start	pH-End	H ₂ SO ₄	PAX	Cu ₂ SO ₄ / H ₂ O	MIBC
Rod Mill Grind	60	50	8.2					
Conditioning	1.0	26	5.8	5.2	1,853	100		
Conditioning	2.0	26	5.5	6.0				20
Ro Flotation - 1	5.0	26	6.0	6.9				10
Conditioning	2.0	23	6.9	6.4	463			
Conditioning	1.5	23	6.4	7.0		50		15
Ro Flotation - 2	5.0	23	7.0	6.0				
Conditioning	3.0	22	6.0	6.8		30		250
Ro Flotation - 3	5	22	6.8					10

Total Reagent Used, g/mt of feed
Solution Concentration

2,316 180 250 55
97.5% 1% 5% 0.005%

Results:

Products	Weight gr.	%	Chemical Analysis			Percent Distribution		
			Au g/mt	Ag g/mt	As %	Au	Ag	As
Feed (analyzed)	1,000.0							
Feed (enriched)	1,005.4	100.0	19.48	18.84	2.84			
			100.0	100.0	100.0	100.0	100.0	100.0
Ro-Froth 1	149.3	14.8	95.54	24.63	12.35	75.3	22.3	74.9
Ro-Froth 2	38.0	3.8	42.80	7.91	5.22	8.6	1.8	8.1
Ro-Froth 3	23.9	2.4	25.10	6.65	3.32	3.2	1.0	3.2
Ro-Tail 3	794.2	79.0	3.09	15.57	0.43	13.0	74.9	13.9

Observations:

Grind in Steel Mill.

Flootation in a 3.2 liter cell at 1600 RPM

Rougher Tailing 1
Rougher Tailing 2
Ro-Froth 1, 2, 3

Rougher Tailing 1	856.1	85.15	5.47	14.98	0.72	24.7	122.3	174.9
Rougher Tailing 2	818.1	81.37	3.73	15.31	0.51	16.1	24.1	82.9
Ro-Froth 1, 2, 3	211.2	21.0	78.08	19.59	10.05	87.0	25.1	86.1

Appendix D

Flotation Test No. 4

Purpose:

To produce rougher flotation concentrate for cleaner flotation and gravity concentration studies.

Sample:

Approximately 5,000 grams of the RDI Composite Sample 1.
(Five tests of 1,000 grams each)

Procedure:

Operations	min.	Conditions			PAX	Reagents, g/m³ of flotation feed	MIBC
		Solids %	pH-Start	pH-End			
Rod Mill Grind	60	50		8.7			
Conditioning	1.0	26		8.8	100		20
Ro Flotation - 1	5.0	26	8.8	8.8			
Conditioning	2.0		8.8	8.8	50		15
Ro Flotation - 2	5.0		8.8	8.7			
Conditioning	3.0		8.7	9.5	285		
Conditioning			9.5	9.2	30		
Ro Flotation - 3	5					10	

Total Reagent Used, g/m³ of feed
Solution Concentration

180 285 45
1% 100% 0.005

Results:

Products	Weight	%	Au	Chemical Analysis			Percent Distribution
				g/int	As	CO ₂	
Feed (analyzed)	5,000.0		19.48				
Feed (calculated)	4,962.5	100.0	16.02		2.84		
Ro-Froth 1	873.9	17.6	81.96		10.85	7.47	100.0
Ro-Tail 3	4,088.6	82.4	1.92		0.285		100.0

Observations:

Five tests run, repeating reagent levels, average pH values shown.
Grind in Steel Mill.
Flotation in a 3.2 liter cell at 1650 RPM.

102
103

APPENDIX E
CLEANER FLOTATION TESTS DATA

Appendix E

Flotation Test No. 5

Purpose:

To study the effects of multiple cleaner flotation upon the rejection of carbonates, and upgrading of gold.

Sample:

Approximately 265 grams of the Rougher Flotation Test 4, RDi Composite Sample 1.

Procedure:

Operations	min.	Conditions			Reagents, #/mt of Rougher Flotation Feed (Test 4)	
		Solids %	pH-Start	pH-End	PAX	MIBC
Conditioning	1.0	12	8.7	8.7		1
Cl Flotation - 1	10.0	12	8.7	8.3		
Cond Cl Froth #1	2.0	7	8.0	7.9		2
Cl Flot - 2, froth #1	5.0	7	7.9	4		
Cl Flot - 2, froth #2	4.0	2	7.7			2

Total Reagent Used, g/mt of feed
Solution Concentration

Results:

Products	Weight	Chemical Analysis			Percent Distribution				
		g	%	Ag g/mt	As g/mt	CO ₂ %	Au g/mt	As g/mt	CO ₂ %
Feed (analyzed)	265.0			81.96	10.85	7.47			
Feed (calculated)	265.2	100.0	81.00		10.47				
Cl-Con 2, Froth 1	115.0	43.4	118.00	15.00	0.54		63.2		62.1
Cl-Con 2, Froth 2	31.4	11.8	155.14	20.80			22.7		23.5
Cleaner Tail 2	15.6	5.9	98.68	13.10	6.56		7.2		7.4
Cleaner Tail 1	103.2	38.9	14.54	1.89	16.20		7.0		5.2
									84.4
									7.0

Observations:

Flotation in a 2.1 liter cell at 1330 RPM

Appendix E

Flotation Test No. 6

Purpose:

To study the first cleaned flotation kinetics upon the rejection of carbonate, and the recovery of gold.

Sample:

Approximately 275 grams of the Rougher Froth Flotation Test 4, RDI Composite Sample 1.

Procedure:

Operations	Conditions			Reagents, g/mt of Rougher Flotation Feed (Test 4)		
	min.	Solids %	pH-Start	pH-End	PAX	MBIC
Conditioning	1.0	12	8.5	8.5		2
Cl-Flotation - 1	1	12	8.5	8.5		
Cl-Flotation - 2	3.0	7	8.0	8.4		
Conditioning	0.5		8.4	8.2	4	2
Cl-Flotation - 3	6.0					

Total Reagent Used, g/mt of feed
Solution Concentration

10
1%
4
0.005

Results:

Products	Weight g.	%	Chemical Analysis				Percent Distribution			
			Au g/mt	Ag g/mt	As %	CO ₂	Au	Ag	As	CO ₂
Feed (analyzed)	275.0		81.96		10.85	7.47				
Feed (calculated)	273.8	100.0	82.79		10.72	7.17				
Cl-Froth #1	115.2	42.1	122.02		16.10	1.10	62.0	63.2	6.5	
Cl-Froth #2+3	56.4	20.6	130.80		16.40	5.20	32.5	31.5	14.9	
Cleaner Tail #3	102.2	37.3	12.07		1.52	15.10	5.4	5.3	78.6	

Observations:

Flootation in a 2.1 liter cell at 1350 RPM

Cl-Froth #1, 2, 3

171.6 62.7 124.91 16.20 2.45 94.6 94.7 21.4

RDI Project No.: 02-020
Date: 19-Dec-03

104
105

Appendix E

Flotation Test No. 7

RDI Project No.: 02-020
Date: 15-Jan-03

¹⁰⁵
106

Purpose: To produce rougher flotation concentrate for cleaner flotation studies.

Sample: Approximately 10,000 grams of the RDI Composite Sample 1.

Procedure:

Operations	Conditions				Reagents, g/unit of flotation feed			
	min.	Solids %	pH-Start	pH-End	PAX	Na ₂ S	Cu ₂ SO ₄	MBIC
Rod Mill Grind	48	50	9.0	9.0		500		
Conditioning	2.0	35	8.9	9.0		100		
Ro Flotation - 1	8.0	35	9.0	9.3				20
Conditioning	2.0		9.3	9.3		50		
Ro Flotation - 2	5.0		9.3	9.0				15
Conditioning	3.0		9.0					
Conditioning	2.0		9.0					
Ro Flotation - 3	8.0		9.0	8.4				
Total Reagent Used, g/unit of feed					180	500	250	45
Solution Concentration					5%	100%	5%	0.005

Results:

Products	Weight gr.	%	Au	Chemical Analysis			Percent Distribution		
				Agmt	As %	S _T %	Au	As	S _T
Feed (analyzed)	10,000		19.48		2.84	4.83			
Feed (calculated)	100.0	18.31			2.53	3.95			
Ro-Froth 1+2+3	1,886.0	18.9	23.27		13.08	20.6			
Ro-Tail 3	8,114.0	81.1	0.89		0.08	0.08			
Grab Feed (thief)	41.6		19.00		2.26	4.83			
Gmb Tail (thief)	108.4		0.89		0.08	0.08			

Observations:

- One test in a 22 liter unit cell.
- Grind in Steel Mill. (Grind 3.9% plus 200-mesh.)
- Rougher froths combined, filtered, and wet-split. One-half for Flotation Test 8 and one-half for Flotation Test 9.
- Approximately 1,000 grams of final tailing employed for Cyanide Leaching Test 3.

Appendix E

Flotation Test No. 8

Purpose:

To study the first cleaner flotation kinetics upon the rejection of carbonate, and the recovery of gold.

Sample:

Approximately 935 grams of the Rougher Flotation Feed Test 7, RDI Composite Sample 1.

Procedure:

Operations	Conditions				Reagent, g/m ³ of Rougher Flotation Feed (Test 4)
	min.	Solids %	pH-Start	pH-End	
Conditioning Cl-Flotation - 1	1.0 10	19 19	8.7 8.4	8.7 8.4	12 1%

Total Reagent Used, g/m³ of Feed

Solution Concentration

2

12

1%

2

0.005

Results:

Products	Weight gr.	%	Au g/ml	Chemical Analysis			Percent Distribution Au	Percent Distribution As	Percent Distribution S _T
				As	%	S _T			
Feed (analyzed)	936.4	100.0	90.39	12.96	20.6		100.0	100.0	100.0
Feed (calculated)									
Cl-Froth #1	756.3	80.8	108.78	15.7	25.1	97.2		97.9	98.5
Cleaner Tail #1	180.1	19.2	13.17	1.44	1.56	2.8		2.1	1.5
Thief	15.6		13.17						

Observations:

Flootation in a 4.2 liter cell at 1350 RPM

Thief sample of Cl-Tail taken for assay.

Remainder of Cl-Tail employed as feed for Cyanide Leaching Test 2.

Appendix E

Flotation Test No. 9

Purpose:

To study the first cleaner flotation kinetics upon the rejection of carbonate, and the recovery of gold.
Sample reground for 60-minutes in ball mill

Sample:

Approximately 950 grams of the Rougher Froth Flotation Test 7, RDI Composite Sample 1.

Procedure:

Operations	min.	Conditions			Reagents, g/ml of Rougher Flotation Feed (Test A)	
		Solids %	pH-Start	pH-End	PAX	MIBC
Ball Mill Grind	60	~50		8.4		
Conditioning	1.0	11	8.1	8.1		
Cl-Froth #1	8	11	8.1	8.1	24	2
Cleaner Tail #1						2

Total Reagent Used, g/ml of feed
Solution Concentration

24
1%
4
0.005

Results:

Products	Weight gr.	%	Chemical Analysis			Percent Distribution		
			g/ml	%	As	S _T	As	S _T
Feed (analyzed) Feed (calculated)	949.6	100.0	96.10		13.20	20.6	100.0	100.0
Cl-Froth #1	443.2	46.7	116.26		15.60	33.0	56.5	55.2
Cleaner Tail #1	506.4	53.3	78.46		11.10	9.8	43.5	44.8

Observations:

Floation in a 8.0 liter cell at 1350 RPM
Two drops MIBC added in conditioner and 2 additional drops added after 1.5 minutes of flotation.

RDI Project No.: 02-020
Date: 15-Jan-03

107
108

Appendix E

Flotation Test No. 10 (A-F)

Purpose: To produce rougher flotation concentrate for Bio-Oxygen studies.

Sample: Approximately 60,000 grams of the RDI Composite Sample 1.

Procedure:

Operations	min.	Conditions			Reagents, g/m ³ of flotation feed		
		Solids %	pH-Start	pH-End	PAX	Na ₂ S	Cu ₂ SO ₄
Rod Mill Grind Conditioning	48	50	8.6	8.9		500	
Ro Flotation - 1 Conditioning	2.0	35	8.6	8.7	100		20
Ro Flotation - 2 Conditioning	2.0	35	8.7	9.1			
Ro Flotation - 3 Conditioning	5.0		9.1	9.3	50		15
Total Reagent Used, g/m ³ of feed					500		
Solution Concentration					5%	5%	0.005

Total Reagent Used, g/m³ of feed
Solution Concentration

Results:

Products	Weight gr.	%	Chemical Analysis			Percent Distribution		
			Au g/m ³	As %	Sn %	As 100.0	Sn 100.0	
Feed (analyzed) Feed (calculated)	60,000	100.0	18.58	19.48	2.84	4.10	4.19	100.0
Ro-Froth 1+2+3	11,707	19.5	91.98	12.80	21.1	96.6	98.1	
Ro-Tail 3	48,293	80.5	0.79	(0.42)	0.10	3.4	11.9	1.9

Observations:

Six test in a 22 liter unit cell. Average pH and reagent additions shown.

Grind in Steel Mill. (Grind 3.9% plus 200-mesh.)

Rougher Froth combined, filtered, and packaged for shipment to RSA for Bio-Ox testing.

Assays for Ro-Froth taken from average of three sample collected after filtration of total Ro-Froths.

Assays for Ro-Tailing taken from average of three samples collected after each of the six flotation tests.

RDI Project No.: 02-020
Date: 15-Jan-03

108
109

109 **Florin Analytical Services, LLC**
110 7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc

Laboratory No.:031058

Client Number: F174

Corrected Report

Attention: Mr. Edwin Bentzen III

Date:02/11/03

Method: Fire assay, gravimetric finish

Lab code: 4002

Check
Gold

Check
Silver

Element:

Gold

Silver

Detection Limit (@ 1 AT):

0.10

1.71

0.10

1.71

Units:

g/mt

g/mt

g/mt

g/mt

Canarc FT-8 Cl Froth #1 113.79 6.02 114.24 5.16

Richard A. Grondin, QC Manager

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

110 111 **Florin Analytical Services, LLC**
7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc

Laboratory No.:031058

Client Number: F174

Attention: Mr. Edwin bentzen III

Date:02/07/03

Method: 4-Acid Digestion & 2 acid digestion AAS analysis.

Lab code:	7002	7015	7017	7023	7018	7043
Element:	Arsenic	Copper	Iron	Mercury	Lead	Zinc
Detection Limit:	0.01	2	2	0.05	2	2
Units:	%	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg
Canarc FT-8 Cl Froth #1	16.15	871	29.6	0.20	194	350

Richard A. Grondin, QC Manager

Nevada Assembly Bill No. 519,130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

111 112 Florin Analytical Services, LLC

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc

Laboratory No.: 031058

Client Number: F174

Attention: Mr. Edwin Bentzen III

Date: 02/07/03

Method: LECO CS-400

Lab code:	7034	7035	7012
Element:	Total Sulfur	Sulfide Sulfur	Carbonate
Detection Limit:	0.01	0.01	0.01
Units:	%	%	%

Canarc FT-8 Cl Froth #1 25.98 25.63 0.72

Richard A. Grondin, QC Manager

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

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113

APPENDIX F
GRAVITY CONCENTRATION TESTS DATA

113
114

Appendix F

Knelson Concentrator Test 1

RDI Project: 02-020
Date: 19-Dec-02

Purpose: To determine the effectiveness of the gravity concentration in the recovery gold from flotation tailing material.

Sample: Approximately 1,000 grams of Rougher Tailing #3, from Flotation Test 4 (RDI Composite Sample 1).

Procedure: The sample was slurried with water and slowly fed to a 3-1/2 inch diameter Model KC-MD3 Knelson Concentrator with water. The water pressure in the Knelson unit was held at 5 psi. The concentrate and tailing products were collected, filtered and dried. The final products were pulverized, and submitted for assay determination of gold.

Results:

Product	Weight g	Weight %	Assay			% Distribution		
			Au g/mt	CO ₂	As %	Au	CO ₂	As
Feed (analyzed)			1.92		0.285			
Feed (calculated)	966.1	100.0	2.78	17.18	0.417	100.0	100.0	100.0
Knelson Concentrate	81.8	8.5	3.50	7.18	0.535	10.7	3.5	10.9
Knelson Tail	884.3	91.5	2.71	18.10	0.406	89.3	96.5	89.1

114
115

Appendix F

Knelson Concentrator Test 2

RDI Project: 02-020
Date: 19-Dec-02

Purpose: To determine the effectiveness of the gravity concentration in the recovery gold from flotation concentrate material.

Sample: Approximately 250 grams of Rougher Froth #'s 1, 2 and 3 combined, from Flotation Test 4 (RDI Composite Sample 1).

Procedure: The sample was slurried with water and slowly fed to a 3-1/2 inch diameter Model KC-MD3 Knelson Concentrator with water. The water pressure in the Knelson unit was held at 5 psi. The concentrate and tailing products were collected, filtered and dried. The final products were pulverized, and submitted for assay determination of gold.

Results:

Product	Weight g	Weight %	Assay		% Distribution	
			Au g/mt	As %	Au	As
Feed (analyzed)			81.96	10.85		100
Feed (calculated)	242.6	100.0	81.27		100.0	
Knelson Concentrate	23.5	9.7	100.34		12.0	
Knelson Tail	219.1	90.3	79.22	11.83	88.0	98.5

¹¹⁵
116

APPENDIX G
LEACH TESTS DATA

Appendix G

Rougher Flotation Tailing Test 1

117

RDI Project:

02-020

Date:

20-Dec-02

Purpose:

To examine the potential for extracting the gold from the flotation tailing material by agitated leaching techniques.

Sample:

Approximately 1,000 grams of combined Rougher Flotation Tailing Test 4. (RDI Composite Sample 1.)

Procedure:

The filtered flotation tailing material was transferred to an rolling bottle and the slurry density was adjusted to ~40% solids. The pH of the slurry was determined. The The pH was raised to about 11 with lime then sodium cyanide was added to a calculated level of ~5.0 g/l. After 72 hours the solution was measured to determine pH, free cyanide, and gold. The slurry was filtered, re-pulp, re-filter, washed and dried. After drying a representative sample of the solids was submitted for determination of gold and silver content by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids
Approx -325-M	48 hours	5.17 g/l	42% Solids

Summary of Results:

Parameter	Au
Extraction, % (1)	23.3
Calculated Head, g/mt	2.13
Actual Head, g/mt	1.92
Final Tail Assay, g/mt	1.63
Cyanide Consumption	3.882 kg NaCN/metric ton ore
Lime Added	0.835 kg CaO/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g		Residual Reagents NaCN g/l	pH	
			NaCN	Ca(OH) ₂		Initial	Adjust
0	2503	1450	7.50	1.16		8.4	
48	2499	1447			2.36	11.1	
Total			7.50	1.16 (0.88)	CaO Equivalent		

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/t	Au ppm
Feed (analyzed)	1050		1.92	
Feed (computed)			2.13	
48 hour Preg		1447	0.36	
48 hour Dry Residue	1052		1.63	

(1) Based on Calculated Head

Appendix G
Rolling Bottle Leaching Test 2

118

RDI Project: 02-020
Date: 16-Jan-03

Purpose: To examine the potential for extracting the gold from the flotation cleaner-tailing material by agitated leaching techniques.

Sample: Approximately 165 grams of Flotation Cleaner Tailing #1, Flotation Test 8, (RDI Composite Sample 1.)

Procedure: The filtered flotation tailing material was transferred to an rolling bottle and the slurry density was adjusted to ~40% solids and rolled in an open bottle to aerate. The slurry was then adjusted to a density ~30% solids. The pH of the slurry was determined. The pH was raised to about 11 with lime then sodium cyanide was added to a calculated level of ~5.0 g/l. After 24 hours the solution was measured to determine pH, free cyanide, and gold. The slurry was filtered, re-pulp, re-filter, washed and dried. After drying a representative sample of the solids was submitted for determination of gold and silver content by fire assay techniques.

Conditions:	Grind	Leach Time	NaCN Concentration	% Solids
	Approx -325-M	24 hours	4.58 g/l	24% Solids

Summary of Results:

Parameter	Au
Extraction, % (1)	17.9
Calculated Head, g/mrt	13.15
Actual Head, g/mrt	13.17
Final Tail Assay, g/mrt	10.80
Cyanide Consumption	4.696 kg NaCN/metric ton ore
Lime Added	3.084 kg CaO/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soin Volume ml	Reagents Added, g		Residual Reagents		pH	
			NaCN	Ca(OH) ₂	NaCN g/l	D.O. ppm	Initial	Adjust
0	500	336				5.35		
24	500	336						
0	674	509	2.33	0.67		5.35	8.1	11.1
4	674	509	0.56		2.80	5.27	11.1	
24	674	509			4.16	5.03	11.1	
Total			2.89	0.67 (0.51)				
					CaO Equivalent			

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/t	Au ppm
Feed (analyzed)	165		13.17	
Feed (computed)			13.15	
24 hour Preg		509	0.76	
24 hour Dry Residue	165		10.80	

(1) Based on Calculated Head.

Appendix G
Rolling Bottle Leaching Test 3

T19

RDI Project:
Date:

02-020
16-Jan-03

Purpose: To examine the potential for extracting the gold from the rougher flotation tailing material by rolling bottle leaching techniques.

Sample: Approximately 1,000 grams of Rougher Flotation Tailing Test 7. (RDI Composite Sample 1.)

Procedure: The filtered flotation tailing material was transferred to an rolling bottle and the slurry density was adjusted to ~40% solids. The pH of the slurry was determined. The pH was raised to about 11 with lime then sodium cyanide was added to a calculated level of ~5.0 g/l. After 24 hours the solution was measured to determine pH, free cyanide, and gold. The slurry was filtered, re-pulp, re-filter, washed and dried. After drying a representative sample of the solids was submitted for determination of gold and silver content by fire assay techniques.

Conditions:	Grind	Leach Time	NaCN Concentration	% Solids
	Approx -325-M	24 hours	4.93 g/l	40% Solids

Summary of Results:

Parameter	Au
Extraction, % (1)	29.5
Calculated Head, g/int	0.92
Actual Head, g/int	0.89
Final Tail Assay, g/int	0.65
Cyanide Consumption	4.139 kg NaCN/metric ton ore
Lime Added	1.738 kg CaO/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g		Residual Reagents		pH	
			NaCN	Ca(OH) ₂	NaCN g/l	D.O. ppm	Initial	Adjust
0	2526	1520	7.50	2.31		2.61	8.8	11.1
4	2526	1520	3.90			2.40	2.66	11.3
24	2526	1520				4.76	3.04	11.3
Total			11.40	2.31 (1.75) CaO Equivalent				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au
			g/t
Feed (analyzed)			0.89
Feed (computed)	1006		0.92
24-hour Preg		1520	0.18
24-hour Dry Residue	1006		0.65

(1) Based on Calculated Head.

119

120

CANARC RESOURCE CORPORATION

PRESSURE OXIDATION TESTING OF
POLARIS TAKU
FLOTATION CONCENTRATES



RESOURCE
DEVELOPMENT
INCORPORATED



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TABLE OF CONTENTS

	<u>Page</u>
INTRODUCTION	1
SUMMARY	2
RESULTS AND DISCUSSION	3
FEED CHARACTERIZATION	3
BATCH ACID PRESSURE OXIDATION	3
DIRECT CYANIDATION OF FEED AND OXIDIZED MATERIAL	4
ORDER-OF-MAGNITUDE CAPITAL AND OPERATING COSTS	7
Process Flowsheet	7
Capital Cost	9
Operating Cost	11

APPENDICES

- Appendix A: Acid Pressure Oxidation Test Data
- Appendix B: Cyanidation Bottle Roll Test Data
- Appendix C: Bio-oxidation Amenability Test Report
- Appendix D: Oxygen Requirement Calculations

INTRODUCTION

Resource Development Inc., (RDI) undertook a metallurgical test program on a bulk sample from Polaris Taku deposit in British Columbia, Canada for Canarc Resources Corporation. Earlier study focused on determining the deportation of gold in various minerals, optimize the flotation process conditions and produce a flotation concentrate containing maximum gold recovery and minimum amount of carbonates¹. Approximately 12 kg of rougher concentrate was produced for follow-up bio-oxidation and pressure oxidation studies. The gold recovery in the flotation concentrate, assaying 91.84 g/T Au and 14.72% As, was 96.6%.

The objective of the present testwork was to determine gold extraction as a function of sulfur oxidation using rougher concentrate as the feed to pressure oxidation. A back-of-the-envelope capital and operating costs for pressure oxidation and bio-oxidation was undertaken for processing 600 stpd of ore to flotation process.

This report discusses the test procedures and results obtained in the study.

¹Metallurgical Testing of Polaris Taku Sample, RDI Report dated March 3, 2003.

SUMMARY

Resource Development Inc., (RDI) performed batch acid pressure oxidation tests on flotation rougher concentrate generated from Polaris Taku ore to determine gold extraction as a function of sulfur oxidation. Oxidor Corporation had done bio-oxidation amenability testing of the same flotation concentrate. RDI prepared a back-of-the envelope capital and operating costs for the two processing options of extracting gold from flotation rougher concentrate. The basis of the estimate was 600 stpd feed to the flotation plant and 120 stpd of rougher concentrate to the oxidation plant.

The highlights of the study indicate the following:

- The flotation concentrate readily oxidizes in the pressure oxidation process (i.e., autoclaves). The sulfur oxidation of 95% to 100% was achieved in 30 minutes to 90 minutes.
- The acid generated in the oxidation process would be sufficient for pre-oxidation treatment of flotation concentrate. The excess acid can be neutralized by mixing with the flotation tailing.
- The gold extraction in the cyanide leach process from oxidized samples ranged from 95% to 100%. The average gold extraction of 98.5% in 24 hours of leach time was assumed for the capital and operating cost study. The NaCN consumption was also reasonable (i.e., 0.8 to 1.4 Kg/T).
- Bio-oxidation process would recover 90% to 93% of gold in CIL circuit following 9 to 12 days of oxidation time. An average gold recovery of 92% was assumed in the capital and operating cost study. The actual recovery may be lower in the process.
- The overall gold recoveries in the flotation/oxidation/leach process were 95.1% for pressure oxidation process and 88.9% for bio-oxidation process. Pressure oxidation process would produce \$2.9 MM per year of additional gold revenue.
- The capital cost for 600 stpd feed to flotation plant was estimated to be \$17,485,000 for pressure oxidation process and \$11,635,000 for bio-oxidation process. The costs do not include crushing, grinding, flotation plant and tailings handling.
- The operating costs were estimated to be \$26.35/st for pressure oxidation process and \$37.40/st for bio-oxidation process. Again, the costs for flotation plant is not included in the estimate. The difference in operating cost is equivalent to \$2.3 MM per year.

Considering the gold recoveries and capital and operating costs, the pressure oxidation process appears to be more attractive than the bio-oxidation process for treatment of Polaris Taku flotation concentrates.

RESULTS AND DISCUSSION

Six batch acid pressure oxidation tests were performed on the flotation concentrate. The flotation concentrate and the six final oxidation residue samples were subjected to cyanidation leach tests.

FEED CHARACTERIZATION

The feed to the pressure oxidation tests was the bulk rougher concentrate generated in an earlier study¹. The flotation concentrate analyses are reported in Table 1.

Table 1. Flotation Rougher Concentrate Analyses

Component	Assay
Au, g/T	91.84
Ag, g/T	11.96
As, %	14.72
Sb, ppm	518
Cu, ppm	754
Fe, %	24.0
Hg, ppm	<0.05
Pb, ppm	94
Zn, ppm	266
C _{inorganic} , %	0.93
S _{Total} , %	20.3
S ²⁻ , %	19.6
Carbonate, %	~3.4

BATCH ACID PRESSURE OXIDATION

Due to the high sulfur content of the concentrate (~20%), the pressure oxidation tests were performed using approximately 100 grams of feed material. The 100-gram concentrate was pre-treated with sulfuric acid to destroy the carbonates in the feed (~3.4%). Acid was added till the slurry pH was stabilized at pH 2 for at least one hour. The acid consumption

125

126

ranged from 120 to 141 kg/T which is significantly higher than the stoichiometric acid demand of 76 kg/T to 80 kg/T for 3.4% carbonates in the flotation concentrate. The high demand indicated the presence of other acid consumers in addition to carbonates in the feed.

The sulfuric acid generated during pressure oxidation (400 to 600 kg/T) exceeds the amount required for carbonate destruction in a pre-oxidation step. Thus, no purchased H_2SO_4 will be required for the pressure oxidation process option. This is a cost benefit.

Following the pre-treatment of the feed samples with acid, batch acid pressure oxidation tests were conducted at pulp density of $\pm 10\%$. The first three tests were conducted in a standard 2 liter Parr autoclave at 180° . The variable in the test series was oxidation time (i.e., 30, 60 and 90 minutes). The second series of tests were conducted at $200^\circ C$ with varying oxidation time as the first series in the same Parr unit.

The acid pressure oxidation test data are given in Appendix A and summarized in Tables 2 & 3. The reaction was exothermic once the oxygen was introduced to the slurry as the sulfide sulfur was readily oxidized to 95% to 99% within 30 minutes and 100% by 60 minutes.

DIRECT CYANIDATION OF FEED AND OXIDIZED MATERIAL

The flotation concentrate (i.e., autoclave feed) and the residues from oxidation tests after washing were subjected to cyanidation bottle roll tests at pH 11, 33% solids and 1 g/L NaCN for 48 hours.

The test results are given in Appendix B and summarized in Tables 2 and 3. The highlights of the test results indicate the following:

- Following oxidation, the gold leaches readily. A leach time of 24 hours is sufficient to extract gold that is leachable.
- The gold extraction dropped when the leach time was extended from 24 hours to 48 hours. There may be preg-robbing minerals present in the flotation concentrate (i.e., element sulfur).
- The NaCN consumption ranged from 0.8 to 1.4 kg/T and the lime consumption was between 5.7 kg/T and 7.8 kg/T.
- The sulfides oxidize readily and the gold extraction in the leach process is greater than 98%.

126

127

Table 2. Pressure Oxidation and Gold Extraction for Tests Performed at 180°C

Oxidation Time, hrs	0.0	0.50	1.0	1.5
H ₂ SO ₄ required for CO ₂ , kg/T	-	145	136	138
Potential H ₂ SO ₄ from sulfide oxidation, Kg/T	-	485	682	663
% S ²⁻ Oxidation	0	98.8	99.4	100
% Weight Loss	0	19.4	24.4	24.7
S _{elemental} , %	-	8.63	1.42	1.11
% Gold Extraction in Cyanide Leach				
6 hrs	15.3	64.3	75.2	83.3
24 hrs	17.6	93.5	100.0	97.1
48 hrs	9.4	94.7	98.4	98.2
Residue, g/T Au	81.1	4.8	1.58	1.71
Cal. Leach Head, g/T Au	89.5	91.1	96.7	94.2
48 hour NaCN Consumption, kg/T	1.333	1.18	1.41	1.033

127
128

Table 3. Pressure Oxidation and Gold Extraction for Tests Performed at 200°C.

Oxidation Time, hrs	0.0	0.50	1.0	1.5
H ₂ SO ₄ required for CO ₂ , kg/T	-	138	121	125
Potential H ₂ SO ₄ from sulfide oxidation, Kg/T	-	293	644	625
% S ²⁻ Oxidation	0	95.1	100.0	100.0
% Weight Loss	0	10.9	26.4	22.3
S _{elemental} , %	-	11.92	2.03	1.32
% Gold Extraction in Cyanide Leach				
6 hrs	15.3	77.8	92.0	97.7
24 hrs	17.6	88.5	100.0	100.0
48 hrs	9.4	83.7	98.7	98.0
Residue, g/T Au	81.1	15.57	1.30	1.92
Cal. Leach Head, g/T Au	89.5	95.8	102.4	94.3
48 hour NaCN Consumption, kg/T	1.333	0.789	1.235	1.185

ORDER-OF-MAGNITUDE CAPITAL AND OPERATING COSTS

Two processing options are being investigated for extraction of gold from sulfide-bearing flotation concentrate. Preliminary pressure oxidation testwork was performed at RDI and discussed in the previous section. The preliminary bio-oxidation testwork was done by Oxidor Corporation and the report is given in Appendix C.

Process Flowsheet

A concentrate process flowsheet was developed for treating gold-bearing sulfide ore in order to determine the order-of-magnitude capital and operating cost for the two processes. The flowsheet is given in Figure 1.

The majority of the processing steps are the same for the two processes. The only difference in the two processes are:

- Pressure oxidation will be done in an autoclave and will require an oxygen generating plant.
- Bio-oxidation will be done in tanks and will require 9 to 12 days of oxidation time. The bio-oxidation process will require tanks for preparation of nutrients for bacteria. The oxygen will be provided by air blowers.
- Both processes will require sulfuric acid for pre-oxidation at the start up. However, the pressure oxidation process will generate more than sufficient acid for pre-oxidation. The excess acid will be neutralized using the flotation plant tailings which contain a high quantity of carbonates.

129
130

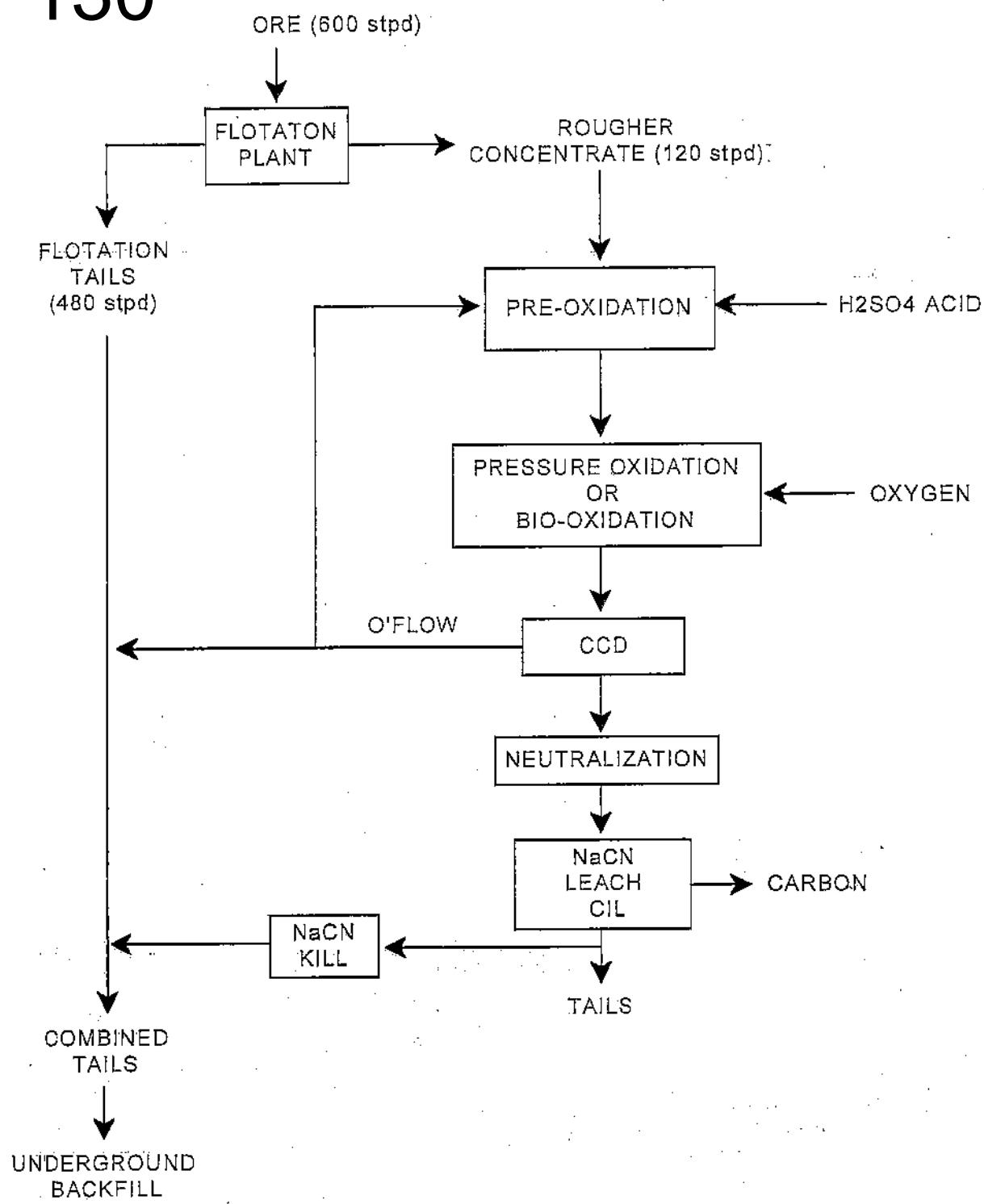


FIGURE 1. CONCEPTUAL PROCESS FLOWSHEET

Capital Cost

The design criteria for estimation of the capital cost is given in Table 4. The oxygen requirements for the pressure oxidation plant would be 50 stpd. The calculations are given in Appendix D. A pressure swing adsorption (PSA) plant is recommended for the O₂ requirement.

Table 4. Design Criteria for Capital Cost Estimation

Parameter	Value
Plant Feed, stpd	600
Feed Grade: g/T Au	18.6
% As	2.84
% S	4.19
Rougher Concentrate Recovery, stpd	120
% Wt.	20
% Au	96.6
% As	88.1
% S	98.1
Concentrate Grade: g/T Au	92
% As	14.72
% S	21.1
Pressure Oxidation Process Recovery: % Au	98.5
Overall: % Au	95.1
Bio-oxidation Process Recovery: % Au	92
Overall: % Au	88.9

The capital costs were estimated based on prior experiences in costing plants of similar size and process. The capital cost is given in Table 5 for both pressure oxidation and bio-oxidation options. The capital cost exclude flotation plant, cyanide destruction and handling of tailings.

The capital cost was estimated to be \$17,485,000 for the pressure oxidation process. The overall gold recovery for the flotation pressure oxidation process would be 95.1%. The technology is proven and currently practiced at several operations.

The capital cost for the flotation/bio-oxidation process was estimated to be \$11,635,000. The overall gold recovery for the flotation/bio-oxidation process would be 88.9%. Assuming

a feed grade of 18.6 g/T, the loss of gold using this process over the pressure oxidation process would be 1.15 g/T or \$13.80/ton of feed based on \$12/g Au. This is equivalent to \$8280 per day (600 stpd) or \$2.9 MM (based on 350 day per year operation).

Table 5: Order-of-Magnitude Capital Cost for Pressure Oxidation and Bio-oxidation Plants (120 stpd)

Pressure	Cost, \$ ¹	
	Pressure Oxidation	Bio-oxidation
Pre-oxidation	250,000	250,000
Pressure Oxidation (Autoclaves)	8,000,000	—
Oxygen Plant	1,500,000	—
Bio-oxidation (Tanks and Blowers)	—	3,000,000
CCD	500,000	500,000
Neutralization	250,000	250,000
CIL ²	250,000	250,000
Carbon Plant/EW	1,000,000	1,000,000
Refinery	200,000	200,000
Reagent Handling	500,000	500,000
Process Building ³	1,000,000	3,000,000
Total Direct	13,450,000	8,950,000
Indirect (30%)	4,035,000	2,685,000
Total Capital	17,485,000	11,635,000

Note: ¹Excludes flotation plant and tailings handling.

²Includes carbon handling, regeneration, acid wash, stripping and electrowinning.

³Process building for bio-oxidation is going to be lot bigger than pressure oxidation building.

Operating Cost

The operating costs including crushing, grinding and flotation were estimated from other projects using similar tonnage and process. The operating costs are given in Table 6. The crushing, grinding and flotation process costs were estimated to be \$5/ton.

The operating costs were estimated to be \$26.35/ton of ore for pressure oxidation and \$37.40/ton of ore for bio-oxidation. The major difference in cost for the two processes is in power and consumables (i.e., nutrients). Also, it is important to note that pressure oxidation technology is well proven and gives higher gold extraction as compared to bio-oxidation.

The difference in operating cost is equivalent to \$6,630/day or \$2.3 MM per year. Combined with the incremental gold recovery cost (\$2.9 MM/yr), pressure oxidation is favored by \$5.2 MM/year.

Table 6. Order-of-Magnitude Operating Cost for Pressure Oxidation and Bio-oxidation Plants (600 stpd ore, 120 stpd concentrate)

Item	Operating Cost, \$/st	
	Pressure Oxidation	Bio-oxidation
Labor	17.00	16.00
Power	5.00	12.60
Consumables	6.97	11.42
Other Supplies	2.38	2.38
Sub-total	31.35	42.40
Crushing/Grinding/Flotation	5.00	5.00
Process Cost	26.35	37.40
Incremental Cost/st ore	0	11.05

¹³³
134

APPENDIX A
ACID PRESSURE OXIDATION TEST DATA

134
135Appendix
Autoclave Oxidation Test 1.RDI Project: 04-011
Date: 14-Apr-04

Purpose: Acid Pressure Oxidation of the sample. Test 1 in a series of 6 tests.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2.

Procedure: The sample was conditioned with sulfuric acid to lower the pH to approximately 2.0, and transferred to a 1-liter Parr autoclave. The solids were adjusted and the unit sealed. The temperature was increased and at the set point oxygen was introduced to produce an over pressure. The unit was operated for 30 minutes, cooled, opened and the sample extracted. The slurry was filtered to remove excess acid. The filter cake was then employed as feed to Bottle Roll Leaching Test 1.

Autoclave Test Conditions

Target Temp.	°F	356	Agitator	rpm	700	Observations:	
Total pressure	psig	350	Pulp, g	initial	1000	Time	Temp
O ₂ over pressure	psi	205		final			
Solids,	wt %	10.0	pH	initial	7.03	0	354
Reagent,	ml	7.90		w/reagent	1.97	10	387
	grams	14.54		final	1.46	15	384
	type	H ₂ SO ₄	emf, mv	initial		20	379
	Grade	95.9%		w/reagent		25	373
	dosage, kg/mt	145		final	323	30	367
Warm-up,	minutes	50	Wt change	%	-19.4		
Time,	minutes	30					
Off-gas bleed,	cc/min						

Products and Assays

Product	Wt, g or Vol, ml	Assays													
		Au g/mt	Ag g/mt	Au ppm	Ag ppm	Fe %	Fe ⁺² g/l	Fe g/l	Cu ppm	Cu mg/l	Sulfide %	Sulfate %	As %	As g/l	H ₂ SO ₄ g/l
Feed	100	91.89	12.80			24.00			754		42.3		12.8		
Autoclave Slurry														49.53	
Autoclave Filtrate	980														
Autoclave Wash															
Autoclave Solids (calc)	80.6		114.0										0.63	8.63	13.84

S⁻ oxidation, %: 98.8
H₂SO₄ consumption: -340 kg H₂SO₄/mt test feed

135
136Appendix
Autoclave Oxidation Test 2.RDI Project: 04-011
Date: 19-Apr-04

Purpose: Acid Pressure Oxidation of the sample. Test 2 in a series of 6 tests.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2.

Procedure: The sample was conditioned with sulfuric acid to lower the pH to approximately 2.0, and transferred to a 1-liter Parr autoclave. The solids were adjusted and the unit sealed. The temperature was increased and at the set point oxygen was introduced to produce an over pressure. The unit was operated for 60 minutes, cooled, opened and the sample extracted. The slurry was filtered to remove excess acid. The filter cake was then employed as feed to Bottle Roll Leaching Test 2.

Autoclave Test Conditions

Target Temp,	°F	356	Agitator	rpm	700	Observations:	
Total pressure	psig	350	Pulp, g	initial	1000	Time	Temp
O ₂ over pressure	psi	205	pH	initial	7.04	0	356
Solids,	wt %	10.0		w/reagent	1.71	10	389
Reagent,	ml	7.40		final	1.44	15	391
	grams	13.62		initial		20	386
	type	H ₂ SO ₄	emf, mv	w/reagent		25	376
	Grade	95.9%		final	340	30	370
	dosage, kg/mt	136	Wt change	%	-24.4	40	357
Warm-up,	minutes	57				50	352
Time,	minutes	60				60	354
Off-gas bleed,	cc/min						

Products and Assays

Product	Wt, g or Vol, ml	Assays													
		Au g/mt	Ag g/mt	Au ppm	Ag ppm	Fe %	Fe ⁺² g/l	Fe g/l	Cu ppm	Cu mg/l	S _{sulfide} %	S _{native} %	As %	As g/l	H ₂ SO ₄ g/l
Feed	100	91.89	12.80			24.00			754	42.3			12.8		
Autoclave Slurry														69.64	
Autoclave Filtrate	980														
Autoclave Wash															
Autoclave Solids (calc)	75.6	121.5									0.35	1.42	16.31		

S²⁻ oxidation, %: 99.4
H₂SO₄ consumption: 546 kg H₂SO₄/mt test feed

136
137Appendix A
Autoclave Oxidation Test 3.RDI Project: 04-011
Date: 15-Apr-04

Purpose: Acid Pressure Oxidation of the sample. Test 3 in a series of 6 tests.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2.

Procedure: The sample was conditioned with sulfuric acid to lower the pH to approximately 2.0, and transferred to a 1-liter Parr autoclave. The solids were adjusted and the unit sealed. The temperature was increased and at the set point oxygen was introduced to produce an over pressure. The unit was operated for 90 minutes, cooled, opened and the sample extracted. The slurry was filtered to remove excess acid. The filter cake was then employed as feed to Bottle Roll Leaching Test 3.

Autoclave Test Conditions

			Agitator	rpm	700	Observations:		
Target Temp.	°F	356	Pulp, g	initial	1000 <th>Time</th> <th>Temp</th> <th>°F</th>	Time	Temp	°F
Total pressure	psig	350		final				
O ₂ over pressure	psi	205	pH	initial	7.08			
Solids,	wt %	10.0		w/reagent	1.52	0	356	
Reagent,	ml	7.50		final	1.03	10	394	
grams		13.80	emf, mv	initial		30	365	
type		H ₂ SO ₄		w/reagent		35	336	
Grade		95.9%		final	352	90	357	
dosage, kg/ml		138	Wt change	%	-24.7			
Warm-up,	minutes	61						
Time,	minutes	90						
Off-gas bleed,	cc/min							

Products and Assays

Product	Wt, g or Vol, ml	Au g/ml	Ag g/ml	Au ppm	Ag ppm	Fe %	Fe ⁺² g/l	Fe g/l	Cu ppm	Cu mg/l	S _{sulfide} %	S _{native} %	As %	As g/l	H ₂ SO ₄ g/l	
Feed	100	91.89	12.80			24.00			754		42.3		12.8			
Autoclave Slurry															67.68	
Autoclave Filtrate	980															
Autoclave Wash																
Autoclave Solids (calc)	75.3		122.0											<0.01	1.11	16.62

S²⁻ oxidation, %H₂SO₄ consumption: 100.0-525 kg H₂SO₄/mt test feed

137
138

Appendix A
Autoclave Oxidation Test 4.

RDI Project: 04-011
Date: 20-Apr-04

Purpose: Acid Pressure Oxidation of the sample. Test 4 in a series of 6 tests.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2.

Procedure: The sample was conditioned with sulfuric acid to lower the pH to approximately 2.0, and transferred to a 1-liter Parr autoclave. The solids were adjusted and the unit sealed. The temperature was increased and at the set point oxygen was introduced to produce an over pressure. The unit was operated for 30 minutes, cooled, opened and the sample extracted. The slurry was filtered to remove excess acid. The filter cake was then employed as feed to Bottle Roll Leaching Test 4.

Autoclave Test Conditions

		Target Temp, °F	391	Agitator rpm	700	Observations:	
Total pressure	psig	350	Pulp, g	initial	1000	Time	Temp
O ₂ over pressure	psi	124		final			°F
Solids,	wt %	10.0	pH	initial	7.08	0	391
Reagent,	ml	7.50		w/reagent	1.52	5	414
	grams	13.80		final	1.09	10	424
	type	H ₂ SO ₄	emf, mV	initial		15	421
	Grade	95.9%		w/reagent		20	414
	dosage, kg/mt	138		final	352	25	405
Warm-up,	minutes	50	Wt change	%	-10.9	30	396
Time,	minutes	30					
Off-gas bleed,	cc/min						

Products and Assays

Product	Wt, g or Vol, ml	Au g/mt	Ag g/mt	Au ppm	Ag ppm	Fe %	Fe ⁺² g/l	Fe g/l	Cu ppm	Cu mg/l	Sulfide %	S native %	As %	As g/l	H ₂ SO ₄ g/l
Feed	100	91.89	12.80			24.00			754	42.3			12.8		
Autoclave Slurry															29.91
Autoclave Filtrate	980														
Autoclave Wash															
Autoclave Solids (calc)	89.1			103.1									2.34	11.92	12.08

S⁻ oxidation, %: 95.1
H₂SO₄ consumption: -155 kg H₂SO₄/mt test feed

138

139

Appendix I Autoclave Oxidation Test 5.

RDI Project: 04-011
Date: 20-Apr-04

Purpose: Acid Pressure Oxidation of the sample. Test 5 in a series of 6 tests.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2.

Procedure: The sample was conditioned with sulfuric acid to lower the pH to approximately 2.0, and transferred to a 1-liter Parr autoclave. The solids were adjusted and the unit sealed. The temperature was increased and at the set point oxygen was introduced to produce an over pressure. The unit was operated for 60 minutes, cooled, opened and the sample extracted. The slurry was filtered to remove excess acid. The filter cake was then employed as feed to Bottle Roll Leaching Test 5.

Autoclave Test Conditions

		391	Agitator	rpm	700	Observations:	
	Total pressure	psig	Pulp, g	initial	1000	Time	Temp °F
O ₂ over pressure	psi	124					
Solids, Reagent,	wt %	10.0	pH	initial	7.03	0	390
ml	6.60		w/reagent	1.72		5	408
grams	12.14		final	0.97		10	412
type	H ₂ SO ₄		emf, mv	initial		15	408
Grade	95.9%		w/reagent			20	401
dosage, kg/m ³	121		final	336		30	388
Warm-up, minutes	60		Wt change	%	-26.4	40	399
Time, minutes	60					50	392
Off-gas bleed, cc/min						60	400

Products and Assays

Product	Wt, g or Vol, ml	Assay Data													
		Au g/ml	Ag g/ml	Au ppm	Ag ppm	Fe %	Fe ⁺² g/l	Fe g/l	Cu ppm	Cu mg/l	S _{sulfide} %	S _{native} %	As %	As g/l	H ₂ SO ₄ g/l
Feed	100	91.89	12.80			24.00			754		42.3		12.8		
Autoclave Slurry														65.74	
Autoclave Filtrate	980														
Autoclave Wash															
Autoclave Solids (calc)	73.6	124.9											0.02	2.03	16.58

S²⁻ oxidation, %: 100.0
H₂SO₄ consumption: -523 kg H₂SO₄/mt test feed

139
140

Appendix A
Autoclave Oxidation Test 6.

RDI Project: 04-011
Date: 20-Apr-04

Purpose: Acid Pressure Oxidation of the sample. Test 6 in a series of 6 tests.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2.

Procedure: The sample was conditioned with sulfuric acid to lower the pH to approximately 2.0, and transferred to a 1-liter Parr autoclave. The solids were adjusted and the unit sealed. The temperature was increased and at the set point oxygen was introduced to produce an over pressure. The unit was operated for 90 minutes, cooled, opened and the sample extracted. The slurry was filtered to remove excess acid. The filter cake was then employed as feed to Bittle Roll Leaching Test 6.

Autoclave Test Conditions

Target Temp.	°F	390	Agitator	rpm	700	Observations:	
Total pressure	psig	350	Pulp, g	initial	1000		
O ₂ over pressure	psi	124		final		Time	Temp
Solids,	wt %	10.0	pH	initial	7.05		°F
Reagent,	ml	6.80		w/reagent	1.72	0	390
	grams	12.51		final	0.94	5	412
	type	H ₂ SO ₄	emf, mV	initial		10	410
	Grade	95.9%		w/reagent		15	406
	dosage, kg/mt	125		final	369	20	399
Warm-up,	minutes	63	Wt change	%	-22.5	30	388
Time,	minutes	90				90	392
Off-gas bleed,	cc/min						

Products and Assays

Product	Wt, g or Vol, ml	Assay Data													
		Au g/ml	Ag g/mt	Au ppm	Ag ppm	Fe %	Fe ⁺² g/l	Fe g/l	Cu ppm	Cu mg/l	S _{sulfide} %	S _{native} %	As %	As g/l	H ₂ SO ₄ g/l
Feed	100	91.89	12.80			24.00			754		42.3		12.8		
Autoclave Slurry														63.75	
Autoclave Filtrate	980														
Autoclave Wash															
Autoclave Solids (calc)	77.7		118.3										0.01	1.32	16.25

S²⁻ oxidation, %: 100.0
H₂SO₄ consumption: ~500 kg H₂SO₄/mt test feed

140
141

Appendix

Project No:
Date:

04-011
15-Jan-03

Sample Description and Preparation:

RDI Sample No: 2

Client's Identification: Bulk Concentrate.

Date Received: 15-Jan-03

Sample Weight: Approximately 1-Kilograms

Sample Container: Plastic Bag.

Sample Description: Rougher flotation concentrate, Flotation Test 10 A-F.

Method of Preparation: The sample was filtercake, and was wet split into smaller charges for testing.

Chemical Analysis:

	Au ¹ g/mt	Ag ¹ g/mt	Fe %	As %	C _{inorganic} %	Sb ppm
	91.89	12.80	24.0	12.80	0.93	518
	S _{native} %	S _{total} %	S _{sulfide} %	Hg ppm	Cu ppm	Zn ppm
Check	1.96	21.1	19.6	< 0.05	754	266
			17.9			94

¹ Fire Assay
Analyses by Florin Analytical Laboratory, Reno, Nevada.

¹⁴¹
142

APPENDIX B
CYANIDATION BOTTLE ROLL TEST DATA

Appendix B

Bottle RPD Leaching Test

143

RDI Project: 04-011
Date: 21-Apr-04

Purpose: To examine the extraction of gold from the sample after autoclave oxidation.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2, after autoclave oxidation.
Autoclave Test 1.

Procedure: After pressure oxidation, the sample was removed from the autoclave and filtered to remove excess acid. The sample was transferred to an Erlenmeyer flask and the slurry was adjusted to 33% solids. The pH of the slurry was determined. The pH was raised to about -11.0 with lime and then sodium cyanide was added to a calculated level of 1.0 g/l. At 2, 6 and 24 hours, the pH and free cyanide were determined and a sample of the solution was assayed for gold. The pH was adjusted with lime to -11.0, and the free cyanide increased to 1.0 g/l with sodium cyanide. After 48 hours, the solution was measured to determine pH, free cyanide, and gold. The slurry was filtered, re-pulp, re-filter, washed and dried. After drying, a representative sample of the solids was submitted for determination of gold content by fire assay techniques.

Conditions:	Grind	Leach Time	NaCN Concentration	% Solids
	80% minus 200 mesh	48 hours	0.97 g/l	33% Solids

Summary of Results:

Parameter	Au	Extraction, % (1)		NaCN Consumed kg/mt
		Au	Au	
Extraction, % (1)	94.7	2 Hours	74.5	0.625
Assayed Head, g/mt (pre-oxidation)	91.89	6 Hours	64.3	1.133
Assayed Head, g/mt (post-oxidation)	114.0	24 Hours	93.5	1.068
Calculated Head; g/mt	91.1	48 Hours	94.7	1.180
Final Tail Assay, g/ml	4.80			

Cyanide Consumption 1.180 kg NaCN/metric ton ore
Lime Added 7.444 kg Ca(OH)₂/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g		Residual Reagents		pH Initial	pH Adjust
			NaCN	Ca(OH) ₂	NaCN	g/l		
0	224	124	0.12	0.47			2.5	11.1
2	224	124	0.07	0.00	0.56		11.0	
6	245	145	0.05	0.13	0.68		10.6	11.3
24	248	148	0.00	0.00	1.04		11.2	
48	245	145			1.00		10.9	
Total			0.24	0.60				
			(0.45)		CaO Equivalent			

B. Product and Analyses

Leach Product	Weight g	Volume ml	Au g/t	Au ppm	Volume Thief
Feed-(analyzed)	100		114.01		
Feed (computed)			91.09		
2 hour Preg		124	44.0		15
6 hour Preg		145	28.0		15
24 hour Preg		148	39.1		15
48 hour Preg		145	36.5		
48 hour Dry Residue	80.6		4.80		
Autoclave loss,	19.4				

(1) Based on calculated head assays.

Appendix B

Batch 144 Leaching Test 2

144

RDI Project: .04-011

Date: 21-Apr-84

Purpose: To examine the extraction of gold from the sample after autoclave oxidation.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2, after autoclave oxidation.
Autoclave Test 2.

Procedure: After pressure oxidation, the sample was removed from the autoclave and filtered to remove excess acid. The sample was transferred to an Erlenmeyer flask and the slurry was adjusted to 33% solids. The pH of the slurry was determined. The pH was raised to about ~11.0 with lime and then sodium cyanide was added to a calculated level of 1.0 g/l. At 2, 6 and 24 hours, the pH and free cyanide were determined and a sample of the solution was assayed for gold. The pH was adjusted with lime to ~11.0, and the free cyanide increased to 1.0 g/l with sodium cyanide. After 48 hours, the solution was measured to determine pH, free cyanide, and gold. The slurry was filtered, re-pulp, re-filter, washed and dried. After drying, a representative sample of the solids was submitted for determination of gold content by fire assay techniques.

Conditions:	Grind	Leach Time	NaCN Concentration	% Solids
	80% minus 200 mesh	48 hours	1.17 g/l	33% Solids

Summary of Results:

Parameter	Au	Extraction, % (1)		NaCN Consumed kg/mt
		Au	Au	
Extraction, % (1)	98.4	2 Hours	51.2	0.758
Assayed Head, g/mt (pre-oxidation)	91.89	6 Hours	75.2	1.153
Assayed Head, g/mt (post-oxidation)	121.5	24 Hours	104.6	1.145
Calculated Head, g/mt	96.7	48 Hours	98.4	1.411
Final Tail Assay, g/mt	1.58			

Cyanide Consumption 1.411 kg NaCN/metric ton ore
Lime Added 5.688 kg Ca(OH)₂/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g		Residual Reagents NaCN g/l	pH	
			NaCN	Ca(OH) ₂		Initial	Adjust
0	194	94	0.11	0.31		9.1	11.1
2	194	94	0.07	0.00	0.56	11.3	
6	229	129	0.04	0.05	0.72	10.8	11.2
24	228	128	0.00	0.07	1.04	10.7	11.2
48	229	129			0.88	11.2	
Total		0.22	0.43				
			(0.33)				CaO Equivalent

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/l	ppm	Volume Thief
Feed (analyzed)	100		121.55		
Feed (computed)			96.72		
2 hour Preg		94	39.8		15
6 hour Preg		129	38.0		15
24 hour Preg		128	50.5		15
48 hour Preg		129	40.9		
48 hour Dry Residue	75.6		1.58		
Autoclave loss.	24.4				

(1) Based on calculated head assays.

Appendix B

Batch 144 Leaching Test 3

145

RDI Project: 04-011
Date: 21-Apr-84

Purpose: To examine the extraction of gold from the sample after autoclave oxidation.

Sample: Approximately 100 grams of Bull Concentrate, RDI Sample 2, after autoclave oxidation.
Autoclave Test 3.

Procedure: After pressure oxidation, the sample was removed from the autoclave and filtered to remove excess acid. The sample was transferred to an Erlenmeyer flask and the slurry was adjusted to 33% solids. The pH of the slurry was determined. The pH was raised to about ~11.0 with lime and then sodium cyanide was added to a calculated level of 1.0 g/l. At 2, 6 and 24 hours, the pH and free cyanide were determined and a sample of the solution was assayed for gold. The pH was adjusted with lime to ~11.0, and the free cyanide increased to 1.0 g/l with sodium cyanide. After 48 hours, the solution was measured to determine pH, free cyanide, and gold. The slurry was filtered, re-pulp, re-filter, washed and dried. After drying, a representative sample of the solids was submitted for determination of gold content by fire assay techniques.

Conditions:	Grind	Leach Time	NaCN Concentration	% Solids
	80% minus 200 mesh	48 hours	1.18 g/l	33% Solids

Summary of Results:

Parameter	Au	Extraction, % (1)	Au	NaCN Consumed
				kg/mt
Extraction, % (1)	98.2	2 Hours	61.7	0.717
Assayed Head, g/mt (pre-oxidation)	91.89	6 Hours	83.3	0.778
Assayed Head, g/mt (post-oxidation)	122.0	24 Hours	97.1	0.839
Calculated Head, g/mt	94.2	48 Hours	98.2	1.033
Final Tail Assay, g/mt	1.71			

Cyanide Consumption 1.033 kg NaCN/metric ton ore
 Lime Added 6.109 kg Ca(OH)₂/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp g	Net Sols ml	Reagents Added, g		Residual Reagents		pH	
			NaCN	Ca(OH) ₂	NaCN	g/l	Initial	Adjust
0	193	93	0.11	0.29			3.9	11.2
2	193	93	0.06	0.00	0.60		11.0	
6	227	127	0.02	0.09	0.88		10.5	11.1
24	227	127	0.00	0.08	1.00		10.6	11.1
48	228	128			0.88		11.0	
Total			0.19	0.46				
			(0.35)		CoO Equivalent			

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/l	ppm	Volume Thief
Feed (analyzed)	100		122.03		
Feed (computed)			94.19		
2 hour Preg		93	46.9		15
6 hour Preg		127	41.1		15
24 hour Preg		127	43.9		15
48 hour Preg		128	39.1		
48 hour Dry Residue	75.3		1.71		
Autoclave loss.	24.7				

(1) Based on calculated head assays.

Appendix B

145
146

Batch Leach Test

RDI Project: 04-011
Date: 21-Apr-04

Purpose: To examine the extraction of gold from the sample after autoclave oxidation.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2, after autoclave oxidation.
Autoclave Test 4.

Procedure: After pressure oxidation, the sample was removed from the autoclave and filtered to remove excess acid. The sample was transferred to an rolling bottle and the slurry was adjusted to 33% solids. The pH of the slurry was determined. The pH was raised to about ~11.0 with lime and then sodium cyanide was added to a calculated level of 1.0 g/l. At 2, 6 and 24 hours, the pH and free cyanide were determined and a sample of the solution was assayed for gold. The pH was adjusted with lime to ~11.0, and the free cyanide increased to 1.0 g/l with sodium cyanide. After 48 hours, the solution was measured to determine pH, free cyanide, and gold. The slurry was filtered, re-pulp, re-filter, washed and dried. After drying, a representative sample of the solids was submitted for determination of gold content by fire assay techniques.

Conditions:	Grind	Leach Time	NaCN Concentration	% Solids
	80% minus 200 mesh	48 hours	1.02 g/l	33% Solids

Parameter	Au	Extraction, % (1)		NaCN Consumed kg/mt
		Au	kg/mt	
Extraction, % (1)	83.7	2 Hours	59.7	0.655
Assayed Head, g/mt (pre-oxidation)	91.89	6 Hours	77.8	0.684
Assayed Head, g/mt (post-oxidation)	103.1	24 Hours	88.5	0.832
Calculated Head, g/mt	95.8	48 Hours	83.7	0.789
Final Tail Assay, g/mt	15.57			

Cyanide Consumption: 0.789 kg NaCN/metric ton ore
Lime Added: 6.846 kg Ca(OH)₂/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soin Volume ml	Reagents Added, g		Residual Reagents NaCN g/l	pH	
			NaCN	Ca(OH) ₂		Initial	Adjust
0	228	128	0.13	0.47		2.9	11.2
2	228	128	0.08	0.00	0.56	11.2	
6	269	169	0.02	0.08	0.88	10.8	11.2
24	269	169	0.01	0.06	0.92	10.8	11.1
48	270	170			1.00	10.8	
Total			0.24	0.61			
			(0.46)	CaO Equivalent			

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au		Volume Thief
			g/t	ppm	
Feed (analyzed)	100		103.13		
Feed (computed)			95.80		
2 hour Preg		128	39.8		15
6 hour Preg		169	35.7		15
24 hour Preg		169	37.9		15
48 hour Preg		170	32.1		
48 hour Dry Residue	89.1		15.57		
Autoclave loss.	10.9				

(1) Based on calculated head assays.

Appendix B

Batch Leaching Test

147

RDI Project: 04-011
Date: 21-Apr-04

Purpose: To examine the extraction of gold from the sample after autoclave oxidation.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2, after autoclave oxidation.
Autoclave Test 5.

Procedure: After pressure oxidation, the sample was removed from the autoclave and filtered to remove excess acid. The sample was transferred to an Erlenmeyer flask and the slurry was adjusted to 33% solids. The pH of the slurry was determined. The pH was raised to about ~11.0 with lime and then sodium cyanide was added to a calculated level of 1.0 g/l. At 2, 6 and 24 hours, the pH and free cyanide were determined and a sample of the solution was assayed for gold. The pH was adjusted with lime to ~11.0, and the free cyanide increased to 1.0 g/l with sodium cyanide. After 48 hours, the solution was measured to determine pH, free cyanide, and gold. The slurry was filtered, re-pulp, re-filter, washed and dried. After drying, a representative sample of the solids was submitted for determination of gold content by fire assay techniques.

Conditions:	Grind	Lече Time	NaCN Concentration	% Solids
	80% minus 200 mesh	48 hours	1.21 g/l	33% Solids

Summary of Results:

Parameter	Au	Extraction, % (1)		NaCN Consumed kg/ml
		Au	Au	
Extraction, % (1)	98.7	2 Hours	66.7	0.801
Assayed Head, g/ml (pre-oxidation)	91.89	6 Hours	92.0	0.824
Assayed Head, g/ml (post-oxidation)	124.9	24 Hours	101.0	0.905
Calculated Head, g/ml	102.4	48 Hours	98.7	1.235
Final Tail Assay, g/ml	1.30			

Cyanide Consumption 1.235 kg NaCN/metric ton ore
Lime Added 6.793 kg Ca(OH)₂/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp g	Net Soln ml	Reagents Added, g		Residual Reagents NaCN g/l	pH	
			NaCN	Ca(OH) ₂		Initial	Adjust
0	191	91	0.11	0.35		3.0	11.1
2	191	91	0.07	0.15	0.56	10.8	11.1
6	224	124	0.01	0.00	0.96	11.5	
24	223	123	0.00	0.00	1.00	11.1	
48	224	124			0.80	10.6	
Total		0.19	0.50				
			(0.38)				CaO Equivalent

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/t	Volume Thief
Feed (analyzed)	100		124.85	
Feed (computed)			102.41	
2 hour Preg	91		55.20	
6 hour Preg	124		49.10	
24 hour Preg	123		49.00	
48 hour Preg	124		41.50	
48 hour Dry Residue	73.6	1.30		
Autoclave loss.	26.4			

(1) Based on calculated head assays.

Appendix B

Batch RDI Leaching Test 6

148

RDI Project: 04-011

Date: 21-Apr-84

Purpose: To examine the extraction of gold from the sample after autoclave oxidation.

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2, after autoclave oxidation.
Autoclave Test 6.

Procedure: After pressure oxidation, the sample was removed from the autoclave and filtered to remove excess acid. The sample was transferred to an rolling bottle and the slurry was adjusted to 33% solids. The pH of the slurry was determined. The pH was raised to about ~11.0 with lime and then sodium cyanide was added to a calculated level of 1.0 g/l. At 2, 6 and 24 hours, the pH and free cyanide were determined and a sample of the solution was assayed for gold. The pH was adjusted with lime to ~11.0, and the free cyanide increased to 1.0 g/l with sodium cyanide. After 48 hours, the solution was measured to determine pH, free cyanide, and gold. The slurry was filtered, re-pulp, re-filter, washed and dried. After drying, a representative sample of the solids was submitted for determination of gold content by fire assay techniques.

Conditions:	Grind	Leach Time	NaCN Concentration	% Solids
	80% minus 200 mesh	48 hours	1.22 g/l	33% Solids

Summary of Results:

Parameter	Au	Extraction, % (1)		NaCN Consumed kg/ml
		Au	Au	
Extraction, % (1)	98.0	2 Hours	69.5	0.834
Assayed Head, g/mt (pre-oxidation)	91.89	6 Hours	97.7	0.782
Assayed Head, g/mt (post-oxidation)	118.3	24 Hours	104.4	0.838
Calculated Head, g/mt	94.3	48 Hours	98.0	1.185
Final Tail Assay, g/mt	1.92			

Cyanide Consumption 1.185 kg NaCN/metric ton ore
Lime Added 7.851 kg Ca(OH)₂/metric ton ore

Detailed Results:**A. Cyanidation Conditions**

Time hrs	Net Pulp g	Net Sols ml	Reagents Added, g		Residual Reagents NaCN g/l	pH	
			NaCN	Ca(OH) ₂		Initial	Adjust
0	199	99	0.12	0.39		3.0	11.2
2	199	99	0.07	0.22	0.56	10.4	11.2
6	235	135	0.01	0.00	0.96	11.4	
24	235	135	0.00	0.00	1.00	11.1	
48	235	135			0.80	10.7	
Total			0.20	0.61			
			(0.46)		CuO Equivalent		

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/l	ppm	Volume Thief
Feed (analyzed)	100		118.26		
Feed (compiled)			94.27		
2 hour Preg		99	51.6		15
6 hour Preg		135	47.4		15
24 hour Preg		135	45.7		15
48 hour Preg		135	37.1		
48 hour Dry Residue	77.7	1.92			
Autoclave loss.	22.3				

(1) Based on calculated head assays.

Appendix B

Bottle Pulp Leaching Test 7

138
149

RDI Project: 04-011

Date: 21-Apr-04

Purpose: To examine the extraction of gold from the sample prior to autoclave oxidation. (Baseline Test).

Sample: Approximately 100 grams of Bulk Concentrate, RDI Sample 2, (no autoclave oxidation).

Procedure: The sample was transferred to an rolling bottle and the slurry was adjusted to 33% solids. The pH of the slurry was determined. The pH was raised to about ~11.0 with lime and then sodium cyanide was added to a calculated level of 1.0 g/l. At 2, 6 and 24 hours, the pH and free cyanide were determined and a sample of the solution was assayed for gold. The pH was adjusted with lime to ~11.0, and the free cyanide increased to 1.0 g/l with sodium cyanide. After 48 hours, the solution was measured to determine pH, free cyanide, and gold. The slurry was filtered, re-pulp, re-filter, washed and dried. After drying, a representative sample of the solids was submitted for determination of gold content by fire assay techniques.

Conditions:	Grind	Leach Time	NaCN Concentration	% Solids
	80% minus 200 mesh	48 hours	0.96 g/l	33% Solids

Summary of Results:

Parameter	Au	Extraction, % (1)		NaCN Consumed kg/mt
		Au	2 Hours	
Extraction, % (1)	9.4		5.5	0.625
Assayed Head, g/mt (pre-oxidation)	91.89		15.3	0.648
Assayed Head, g/mt (post-oxidation)	92.0		17.6	1.013
Calculated Head, g/mt	89.5		9.4	1.333
Final Tail Assay, g/mt	81.10			

Cyanide Consumption 1.333 kg NaCN/metric ton ore
 Lime Added 7.107 kg Ca(OH)₂/metric ton ore

Detailed Results:**A. Cyanidation Conditions**

Time hrs	Net Pulp g	Net Sols ml	Reagents Added, g		Residual Reagents NaCN g/l	pH	
			NaCN	Ca(OH) ₂		Initial	Adjust
0	256	156	0.15	0.71		7.1	11.2
2	256	156	0.09	0.00	0.56	11.2	
6	299	199	0.02	0.00	0.88	11.2	
24	299	199	0.04	0.00	0.80	11.1	
48	299	199			0.84	11.0	
Total			0.30	0.71			
			(0.54)		CaO Equivalent		

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au		Volume Thief
			g/t	ppm	
Feed (analyzed)	100		91.98		
Feed (computed)			89.54		
2 hour Preg		156	3.1		15
6 hour Preg		199	6.7		15
24 hour Preg		199	7.2		15
48 hour Preg		199	3.0		
48 hour Dry Residue	99.9		81.10		

(1) Based on calculated head assays.

**149
150**

Florin Analytical Services, LLC

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc

Laboratory No.: 041294

Client Number: F174

Attention: Mr. Edwin Bentzen III

Date: 05/13/04

Method: Fire assay, gravimetric finish

Lab code:	4002	Check
Element:	Gold	Gold
Detection Limit (@ 1 AT):	0.10	0.10
Units:	g/mt	g/mt

CAPO 1L Residue	4.80	
CAPO 2L Residue	1.58	
CAPO 3L Residue	1.71	
CAPO 4L Residue	15.57	
CAPO 5L Residue	1.30	
CAPO 6L Residue	1.92	
CAPO 7L Residue	80.50	81.70

Richard A. Grondin, QC Manager

Nevada Assembly Bill No. 519-130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

150
151

Florin Analytical Services, LLC

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc

Laboratory No.: 041294

Client Number: F174

Attention: Mr. Edwin Bentzen III

Date: 05/12/04

Method: LECO CS-400

Lab code:	7058	7035
Element:	Native Sulfur	Sulfide Sulfur
Detection Limit:	0.01	0.01
Units:	%	%

CAPO 1L Residue	0.63	8.63
CAPO 2L Residue	0.35	1.42
CAPO 3L Residue	<0.01	1.11
CAPO 4L Residue	2.34	11.92
CAPO 5L Residue	0.02	2.03
CAPO 6L Residue	0.01	1.32
CAPO 7L Residue	1.96	17.90

Richard A. Grondin, QC Manager

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

151

Florin Analytical Services, LLC

152

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Resource Development, Inc

Laboratory No.: 041294

Client Number: F174

Attention: Mr. Edwin Bentzen III

Date: 05/12/04

Method: 2-Acid digestion, ICP Analysis.

Lab code:	7002	Check
Element:	Arsenic	Arsenic
Detection Limit:	0.01	0.01
Units:	%	%

CAPO 1L Residue	13.81	13.86
CAPO 2L Residue	16.31	
CAPO 3L Residue	16.62	
CAPO 4L Residue	12.08	
CAPO 5L Residue	16.58	
CAPO 6L Residue	16.25	
CAPO 7L Residue	14.06	

Richard A. Grondin QC Manager

Nevada Assembly Bill No. 519;130 requires the following statement: "The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project."

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153

APPENDIX C
BIO-OXIDATION AMENABILITY TEST REPORT



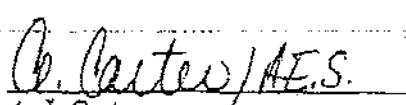
OXIDOR
CORPORATION

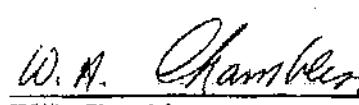
Canarc Resource Corporation
New Polaris Concentrate

REPORT
Biooxidation Amenability Testing of New Polaris Concentrate

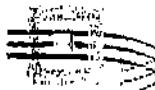
CONFIDENTIAL


C. J. Hunter
Senior Chemist - Process Development


A. J. Carter
Vice President - Technology


W.H. Chambliss
Chief Operations Officer -
Metallurgical Division

154
155



OXIDOR
LABORATORIES

Deepak
FYI

Le Harris

1/24/04

January 14, 2004

Bradford Cooke
Canarc Resource Corporation
800 ~ 850 West Hastings Street
Vancouver, BC
Canada V6C 1E1

Subject - Biooxidation Amenability Testing Of New Polaris Concentrate

Dear Brad:

Per our agreement I am sending you two copies of the report on OXIDOR's amenability test on the New Polaris Concentrate. As you will note, the rates of biooxidation make the concentrate an excellent candidate for enhanced recovery of gold using a biooxidation pretreatment. I have included an invoice for \$10,000.00.

Most service companies suggest future work following an initial viability or amenability assessment. OXIDOR will do the same. In this case, we do so with great enthusiasm. The sample of New Polaris Concentrate not only showed great promise for biooxidation pretreatment, its characteristics introduced new twists. First, we used specifically adapted inoculum that survives the initial high arsenic and low redox environments. Further, we have completed preliminary tests that suggest biooxidation by-products may consume cyanide and indicate potential means of ameliorating their impact. Finally, the indicated rate of reaction may allow for cost-savings on reactor design and operation. As a side issue, Leonard Harris suggested the novelty of the work on New Polaris Concentrates may be suitable for presenting a paper in the fall.

To follow up we would propose further tests. First, to proceed to plant scale design OXIDOR would propose operating our pilot-scale, continuous tank biooxidation unit. To attain high quality engineering data, we would need a minimum of 200 kg of New Polaris Concentrate of a quality similar to the sample that your agent, Deepak Malhotra, provided. We could gather data with less, but to gain experience to guarantee a process, we would need at least 200 kg.

Second, OXIDOR could proceed with studying the causes and solution to high cyanide consumption with samples and intermediates already in hand. While we were very careful with the limited sample that you sent us...normally, we would prefer 10 kg of concentrate for a full amenability test...we retained and save as much as possible.

155
156

Should you have an interest in having OXIDOR work on either project, we would submit a proposal. I will follow up with by phone.

Sincerely,



W. H. Chambliss
Chief Operating Officer

156
157

New Polaris Concentrate
Oxidor Laboratories Job No. 1042 - January 14, 2004

OXIDOR DISCLAIMER

THESE TEST RESULTS ARE BASED SOLELY UPON THE CONTENT OF THE SAMPLE(S) OF MINERALOGICAL MATERIALS SUBMITTED TO OXIDOR CORPORATION. THE TEST RESULTS FOR OTHER SAMPLES OF MINERALOGICAL MATERIALS OBTAINED FROM THE SAME OPERATION MAY VARY SUBSTANTIALLY.

CERTAIN TESTS CONDUCTED ON THE SAMPLES OF MINERALOGICAL MATERIALS SUBMITTED TO OXIDOR CORPORATION, THE RESULTS OF WHICH ARE INCLUDED IN THIS REPORT, MAY HAVE BEEN PERFORMED BY THIRD PARTY COMMERCIAL LABORATORIES. OXIDOR CORPORATION DISCLAIMS ANY WARRANTY AS TO THE ACCURACY OF THE RESULTS OF ANY TESTS CONDUCTED BY SUCH THIRD PARTY LABORATORIES.

New Polaris Concentrate
Oxidor Laboratories Job No. 1042 - January 14, 2004

EXECUTIVE SUMMARY

OXIDOR CORPORATION, INC agreed to conduct a limited amenability test for Canarc Resource Corporation on a New Polaris Concentrate (NPC) sample. The tests were further limited by having only 5 kg of sample, or half of what was requested. The primary focus, as agreed, would be to generate data to show the percent gold recovery versus percent oxidation and the percent oxidation versus time. Nonetheless, OXIDOR believes its findings fairly represent biooxidation potential as presented by the sample received.

A batch test was conducted on a concentrate sample from New Polaris to determine its amenability to biooxidation treatment followed by cyanidation of the biooxidized residue. The sample was designated as the New Polaris Concentrate or NPC. According to the accompanying analysis, the NPC sample contained $91.84 \text{ g Au m}^{-3}$, $11.96 \text{ g Ag m}^{-3}$ and 19.6 % sulfide sulfur. The concentrate was characterized as highly arsenopyritic. Canarc advised OXIDOR that previous work suggested that NPC might undergo biooxidation with difficulty. OXIDOR identified an inoculum specifically adapted to high-arsenopyritic ores. The inoculum selected was from OXIDOR's database and patented library.

Baseline metallurgical testing indicates that the NPC sample is refractory. Gold and silver recovery using Carbon in leach (CIL) was 8.2 % and 17.6 % respectively after 24 hours. OXIDOR achieved 90 % recovery of gold in about 9 days using biooxidation pretreatment. Testing of the NPC sample indicates that it is amenable to biooxidation.

A twenty-liter air sparged CSTR vessel was used for bacterial adaptation and inoculum build-up. Adaptation was initiated using an active mixed culture designated as OXL-1014-R-13. The build up was atypical, undoubtedly owing to NPC's high arsenopyrite content. When NPC was added to OXIDOR's specifically selected inoculum, the slurry's redox potential fell from 825 mV E_h to 590 mV E_h in two days. No bacterial activity was observed. By day three oxygen uptake, indicating bacteria activity, revived. By day 10, bacterial activity was exceptionally high.

Canarc had mentioned that preliminary oxidation work suggested cyanide consumption might be high. OXIDOR did confirm high cyanide consumption under the laboratory conditions but washing the biooxidation residues appeared to reduce it. Counter current decantation on a plant scale will help. OXIDOR has explored alternatives to reduce generation of cyanide consuming by-products should that be an issue.

OXIDOR recommends that a 60-liter biooxidation test-work program be implemented in order to generate engineering data. Lime and acid consumption is typically critical to tank biooxidation economics and needs to be determined. Given the apparent high biooxidation rate, an unusual inoculum build-up profile and need for especially adapted bacteria required for NPC, tank biooxidation designed may be dramatically enhanced to exploit its unusual character. Based on preliminary data, biooxidation of NPC appears facile and economically attractive.

New Polaris Concentrate
Oxidor Laboratories Job No. 1042 - January 14, 2004

TABLE OF CONTENTS

1.0	<u>INTRODUCTION</u>	1
2.0	<u>EXPERIMENTAL PROCEDURE</u>	2
2.1	Procedure Summary.....	2
2.2	Bacterial Adaptation.....	2
	Figure No. 1 - Reactor Operation Plot Showing Redox Transients in Arsenopyrite System.....	3
3.0	<u>RESULTS</u>	3
3.1	New Polaris Concentrate Sample Characterization	3
	Table No. 1 – Analysis of Sample.....	3
3.2	Biooxidation Amenability	4
	Figure No. 2 Sulfide Oxidation and Gold Recovery vs. Time.....	4
	Figure No. 3 Gold Recovery vs. Sulfide Oxidation.....	5
4.0	<u>CONCLUSIONS & RECOMMENDATIONS</u>	5

New Polaris Concentrate
Oxidor Laboratories Job No. 1042, January 14, 2004

1.0 INTRODUCTION

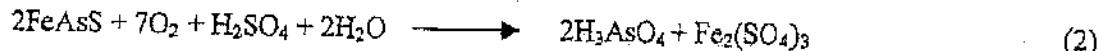
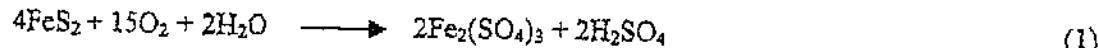
Biooxidation is a biohydrometallurgical process for pre-cyanidation treatment of refractory gold ores. This process offers an alternative to the more conventional roasting or pressure oxidation techniques with the following advantages:

- Reduced capital costs.
- A highly flexible and easily controllable process.
- Improved recovery rates.
- A process that is environmentally friendly.

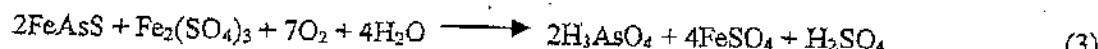
The biooxidation process utilizes a naturally occurring mixed bacterial consortium comprising *Thiobacillus ferrooxidans*, *Thiobacillus Thiooxidans* and *Leptospirillum ferrooxidans*. These bacteria, under controlled conditions, oxidize gold-bearing sulfide ores or concentrates due to a chemolithotrophic mode of metabolism. This means that they require inorganic compounds for the acquisition of both energy and carbon. Inorganic substrates such as sulfides, elemental sulfur, and ferrous iron are oxidized by the bacteria to provide chemical energy. This is enzymatically converted, by oxidative phosphorylation, to ATP, a form of metabolic energy utilized by the bacteria for various cellular functions. The carbon requirements of the bacteria for biosynthesis of cellular biomass are met by CO₂ in the atmosphere or from dissolution of carbonate minerals in the ore.

The mixed bacterial population can oxidize a wide range of metal sulfide minerals. These include pyrite, arsenopyrite, pyrrhotite, chalcopyrite, chalcocite, covellite, stibnite, pentlandite and galena. The mechanism of sulfide breakdown is usually a combination of direct enzymatic attack and indirect chemical attack by metabolic by-products of the bacteria.

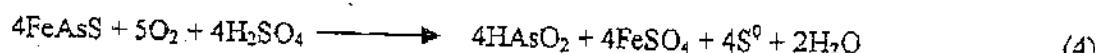
For direct enzymatic oxidation, attachment of the bacteria to the sulfide mineral is essential. Under ideal conditions bacterial oxidation of pyrite and arsenopyrite are illustrated in the following equations:



The ferric sulfate produced contributes to further sulfide breakdown by indirect chemical attack:



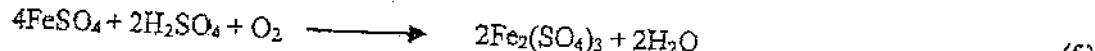
Partial arsenopyrite oxidation may also occur by acid attack:



The arsenic acid produced from oxidation of arsenopyrite is effectively neutralized with lime to form non-polluting ferric arsenate precipitates, which have proved to be stable provided the Fe:As molar ratio in the biooxidation liquor is greater than 3:1.

The Ferrous sulfate and elemental sulfur are then bacterially re-oxidized to ferric sulfate and sulfuric acid, respectively:

New Polaris Concentrate
Oxidor Laboratories Job No. 1042, January 14, 2004



Dmp

Under controlled continuous process conditions, the number of bacterial cells and their activity is optimized to attain the highest rate of sulfide oxidation. Detailed laboratory and pilot plant studies have indicated that the bacteria require a very acidic environment (pH 1.2 to 1.8), a temperature of between 30 °C to 45 °C and a steady supply of oxygen and carbon dioxide for optimum growth and activity. The optimum conditions for the bacteria are not favorable for the growth of most other microbes, thus eliminating the need for sterility during the biooxidation process.

The NPC sample that OXIDOR tested presented issues not usually encountered. The natural redox potential of the sample fell below 620 mV E_h. Instead of generating As⁺⁵ as shown in equation 4, a competing side reaction under low redox conditions generates As⁺³. Arsenite, As⁺³, is toxic to the bacteria. The bacteria must be adapted to high arsenite levels.

Batch biooxidation amenability tests evaluate the amenability of a concentrate or an ore to biooxidation and identifies potential process problems prior to pilot testing.

2.0 EXPERIMENTAL PROCEDURE

2.1 Procedure Summary

The NPC sample was prepared by the client and was split as received without further milling. Head assays for gold, silver, sulfide and iron were performed prior to each phase of test-work. Oxidative pretreatment using bio-oxidation was carried out to improve leach recoveries. During the oxidation phase, following inoculum build-up, numerous sample splits were treated to achieve incremental oxidation levels for determination of leach improvement. Following oxidation, recovery was determined using the carbon-in-leach procedure. Baseline tests were performed concurrently to determine pre-oxidation recovery. Oxidation test results were presented based on iron dissolution (from the bio-oxidation procedure) and leach test results were presented based on the entire calculated gold grade accounted for in each test (vice a single head grade).

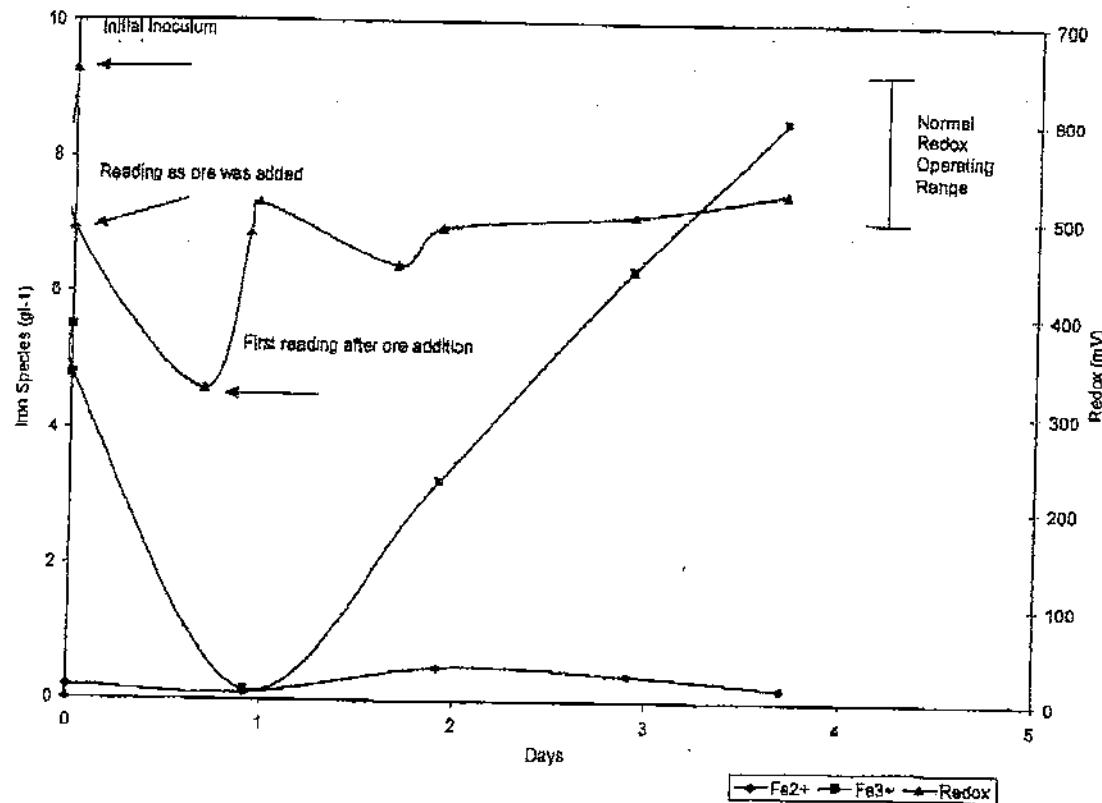
2.2 Bacterial Adaptation

A twenty-liter air sparged CSTR vessel was used for bacterial adaptation and inoculum build-up. Adaptation was initiated using an active mixed culture designated as OXL-1014-R-13. The build up was atypical, undoubtedly owing to NPC's high arsenopyrite content. When NPC was added to OXIDOR's specifically selected inoculum, the slurry's redox potential fell from 825 mV E_h to 590 mV E_h in two days. No bacterial activity was observed. By day three oxygen uptake, indicating bacteria activity, revived. By day 10, bacterial activity was unusually high. See Figure No. 1.

Ferric and ferrous iron concentrations, temperature, dissolved oxygen, pH and redox potential were determined on a daily basis. Adjustments to the slurry pH were accomplished by using a limestone slurry or concentrated sulfuric acid as required.

New Polaris Concentrate
Oxidor Laboratories Job No. 1042, January 14, 2004

Figure No. 1 - Reactor Operation Plot Showing Redox Transients in Arsenopyrite System



3.0 RESULTS

3.1 New Polaris Concentrate Sample Characterization

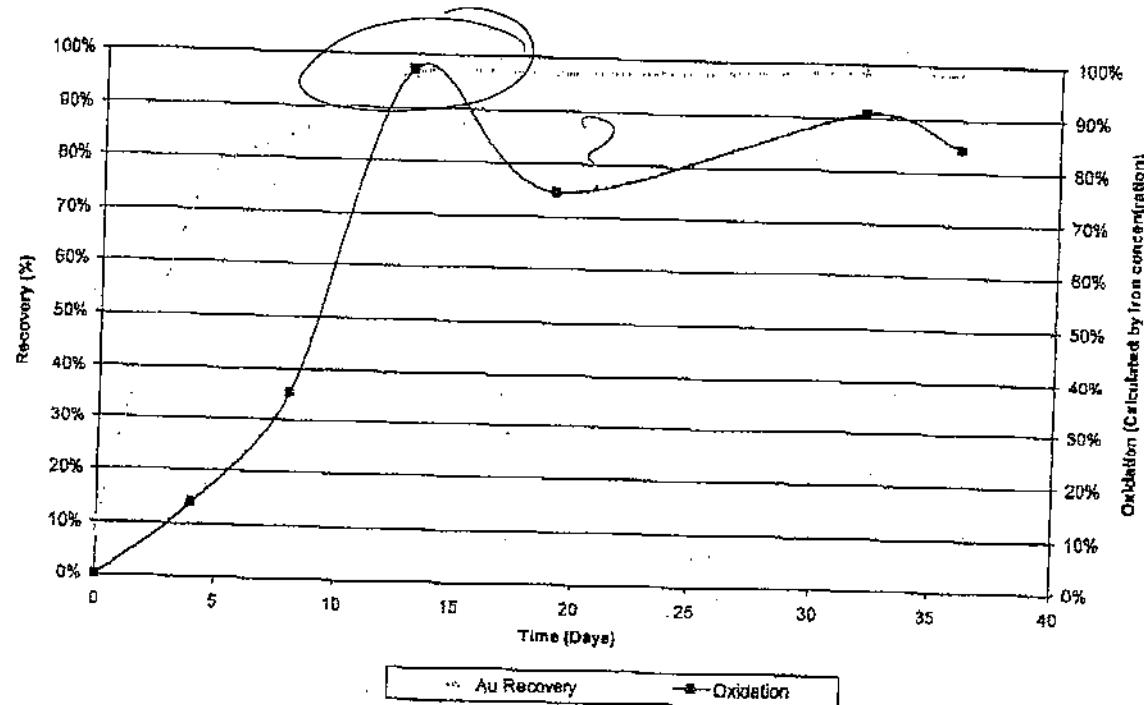
Table No. 1 – Analysis of Sample

Component	Client-Supplied	OXIDOR
Au, g mt ⁻¹	91.84	93.96
Ag, g mt ⁻¹	11.96	12.94
As, %	14.72	
Sb, ppm	518	
Cu, ppm	754	
Fe, %	24.0	
Hg, ppm	<0.05	
Pb, ppm	94	
Zn, ppm	266	
C _{inorganic}	0.93	
S _{total}	20.3	
S ²⁻	19.6	

New Polaris Concentrate
Oxidor Laboratories Job No. 1042, January 14, 2004

3.2 Biooxidation Amenability

Figure No. 2 – Sulfide Oxidation and Gold Recovery vs. Time



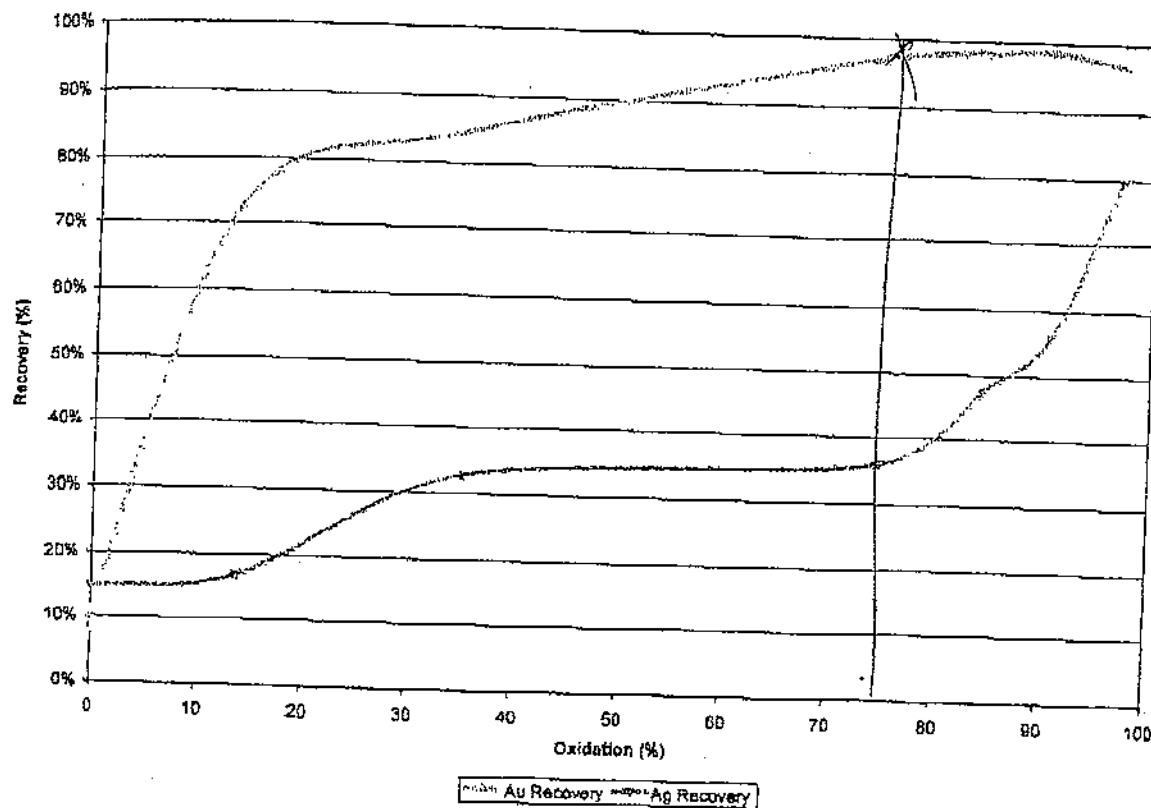
Results show that the New Polaris Concentrate sample is amenable to biooxidation treatment, and the biooxidized residues are amenable to cyanidation and CIL treatment. Gold recovery from the NP sample improved from the baseline 8.2% to 90 % after 9 days of biooxidation treatment. See Figure No. 2 and 3.

The residence time required for a continuous biooxidation process should be determined by pilot testing but excellent kinetics are indicated. Higher effective rates of oxidation are achieved in continuous biooxidation, principally because of a greater and more active bacterial population in the process.

Jones
very
slow

New Polaris Concentrate
OXIDOR Laboratories Job No. 1042, January 14, 2004

Figure No. 3 – Gold Recovery vs. Sulfide Oxidation Based on Calculated Head



4.0 CONCLUSIONS & RECOMMENDATIONS

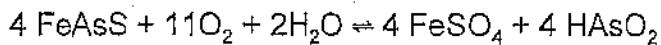
The New Polaris Concentrate sample is very amenable to biooxidation pretreatment. Batch oxidation rates are rapid and the resultant improvement in gold recovery is excellent.

OXIDOR recommends that a continuous biooxidation test-work program be implemented in order to generate engineering data for a feasibility analysis. To guarantee engineering data, OXIDOR would need a minimum of 200 kg of concentrate. Lesser quality information could be attained with smaller samples. Lime and acid consumption is typically critical to tank biooxidation economics and needs to be determined. Given the apparent high biooxidation rate, an unusual inoculum build-up profile and need for especially adapted bacteria required for NPC, tank biooxidation designed may be dramatically enhanced to exploit its exceptional character. Based on preliminary data, biooxidation of NPC appears facile and economically attractive.

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165

APPENDIX D
OXYGEN REQUIREMENT CALCULATIONS

165

166Overall Oxidation Reaction**Design Basis:**

$$\text{Fw} = \text{FeAsS} = 55.85 + 74.92 + 32 = 162.77$$

$$\% \text{Fe} = \frac{55.87}{162.77} = 34.3$$

$$\% \text{As} = \frac{74.92}{162.77} = 46.0$$

$$\% \text{S} = \frac{32}{162.77} = 19.7$$

Assume all S²⁻ in design basis is as FeAsS

$$(0.211) (120 \text{ stpd}) = 25.32 \text{ stpd S}^{2-}$$

$$(0.197) (X \text{ stpd FeAsS}) = 25.32$$

$$X = \frac{25.32}{0.197} = 128.53 \text{ stpd}$$

This tonnage is more than the daily concentrate tonnage of 120 stpd. Hence all the S²⁻ cannot be associated with the FeAsS.

Assume all the As in design basis is as FeAsS.

$$(0.147) 120 \text{ stpd} = 17.64 \text{ stpd As}$$

$$\text{Hence } (0.46) (X \text{ FeAsS}) = 17.64$$

$$\text{FeAsS} = \frac{17.64}{0.46} = 38.3 \text{ stpd}$$

This calculation implies a grade of 31.9% FeAsS in the flotation concentrate.

$$\text{Amount of S Associated with FeAsS} = (0.197) (0.319) (120) = 7.54 \text{ stpd}$$

$$\text{Hence } 25.3 - 7.54 = 17.78 \text{ stpd of S associated with pyrite or other sulfides}$$

166
167

For O₂ requirements calculations assume no other sulfides except pyrite and arsenopyrite are associated with sulfur.

$$Fw = FeS_2 = 55.85 + 64 = 119.85$$

$$\%S = \frac{64}{119.85} = 53.4\%$$

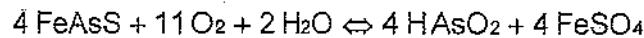
$$\%Fe = 46.6\%$$

$$\text{Hence stpd } FeS_2 = (0.534)(X) = 17.78$$

$$X = 33.3 \text{ stpd}$$

O₂ Requirement

Reaction 1:



$$(651.08) (352)$$

$$38.3 \text{ stpd } X$$

$$X = \frac{(352)(38.3)}{651.08} = 20.7 \text{ stpd O}_2 \text{ (for FeAsS)}$$

Reaction 2:



$$(239.7) (120)$$

$$33.3 \text{ stpd}$$

$$X = \frac{(120)(33.3)}{239.7} = 16.7 \text{ stpd O}_2 \text{ (for FeS}_2\text{)}$$

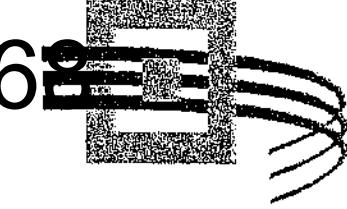
$$\text{Total O}_2 \text{ required} = 20.7 + 16.7 = 37.4 \text{ stpd}$$

Assuming 80% O₂ utilization

$$\text{O}_2 \text{ requirement} = \frac{37.4}{0.8} = 47 \text{ stpd}$$

Hence build a pressure swing adsorption (PSA) plant producing 50 stpd of O₂.

167
168



OXIDOR
CORPORATION

Canarc Resource Corporation
New Polaris Concentrate

REPORT
Biooxidation Amenability Testing of New Polaris Concentrate

CONFIDENTIAL


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169

OXIDOR DISCLAIMER

THESE TEST RESULTS ARE BASED SOLELY UPON THE CONTENT OF THE SAMPLE(S) OF MINERALOGICAL MATERIALS SUBMITTED TO OXIDOR CORPORATION. THE TEST RESULTS FOR OTHER SAMPLES OF MINERALOGICAL MATERIALS OBTAINED FROM THE SAME OPERATION MAY VARY SUBSTANTIALLY.

CERTAIN TESTS CONDUCTED ON THE SAMPLES OF MINERALOGICAL MATERIALS SUBMITTED TO OXIDOR CORPORATION, THE RESULTS OF WHICH ARE INCLUDED IN THIS REPORT, MAY HAVE BEEN PERFORMED BY THIRD PARTY COMMERCIAL LABORATORIES. OXIDOR CORPORATION DISCLAIMS ANY WARRANTY AS TO THE ACCURACY OF THE RESULTS OF ANY TESTS CONDUCTED BY SUCH THIRD PARTY LABORATORIES.

170

EXECUTIVE SUMMARY

OXIDOR CORPORATION, INC agreed to conduct a limited amenability test for Canarc Resource Corporation on a New Polaris Concentrate (NPC) sample. The tests were further limited by having only 5 kg of sample, or half of what was requested. The primary focus, as agreed, would be to generate data to show the percent gold recovery versus percent oxidation and the percent oxidation versus time. Nonetheless, OXIDOR believes its findings fairly represent biooxidation potential as presented by the sample received.

A batch test was conducted on a concentrate sample from New Polaris to determine its amenability to biooxidation treatment followed by cyanidation of the biooxidized residue. The sample was designated as the New Polaris Concentrate or NPC. According to the accompanying analysis, the NPC sample contained $91.84 \text{ g Au m}^{-1}$, $11.96 \text{ g Ag m}^{-1}$ and 19.6 % sulfide sulfur. The concentrate was characterized as highly arsenopyritic. Canarc advised OXIDOR that previous work suggested that NPC might undergo biooxidation with difficulty. OXIDOR identified an inoculum specifically adapted to high-arsenopyritic ores. The inoculum selected was from OXIDOR's database and patented library.

Baseline metallurgical testing indicates that the NPC sample is refractory. Gold and silver recovery using Carbon in leach (CIL) was 8.2 % and 17.6 % respectively after 24 hours. OXIDOR achieved 90 % recovery of gold in about 9 days using biooxidation pretreatment. Testing of the NPC sample indicates that it is amenable to biooxidation.

A twenty-liter air sparged CSTR vessel was used for bacterial adaptation and inoculum build-up. Adaptation was initiated using an active mixed culture designated as OXL-1014-R-13. The build up was atypical, undoubtedly owing to NPC's high arsenopyrite content. When NPC was added to OXIDOR's specifically selected inoculum, the slurry's redox potential fell from 825 mV E_h to 590 mV E_h in two days. No bacterial activity was observed. By day three oxygen uptake, indicating bacteria activity, revived. By day 10, bacterial activity was exceptionally high.

Canarc had mentioned that preliminary oxidation work suggested cyanide consumption might be high. OXIDOR did confirm high cyanide consumption under the laboratory conditions but washing the biooxidation residues appeared to reduce it. Counter current decantation on a plant scale will help. OXIDOR has explored alternatives to reduce generation of cyanide consuming by-products should that be an issue.

OXIDOR recommends that a 60-liter biooxidation test-work program be implemented in order to generate engineering data. Lime and acid consumption is typically critical to tank biooxidation economics and needs to be determined. Given the apparent high biooxidation rate, an unusual inoculum build-up profile and need for especially adapted bacteria required for NPC, tank biooxidation designed may be dramatically enhanced to exploit its unusual character. Based on preliminary data, biooxidation of NPC appears facile and economically attractive.

171

TABLE OF CONTENTS

1.0	<u>INTRODUCTION</u>	1
2.0	<u>EXPERIMENTAL PROCEDURE</u>	2
2.1	Procedure Summary	2
2.2	Bacterial Adaptation	2
	Figure No. 1 - Reactor Operation Plot Showing Redox Transients in Arsenopyrite System	3
3.0	<u>RESULTS</u>	3
3.1	New Polaris Concentrate Sample Characterization	3
	Table No. 1 – Analysis of Sample	3
3.2	Biooxidation Amenability	4
	Figure No. 2 Sulfide Oxidation and Gold Recovery vs. Time	4
	Figure No. 3 Gold Recovery vs. Sulfide Oxidation	5
4.0	<u>CONCLUSIONS & RECOMMENDATIONS</u>	5

1.0 INTRODUCTION

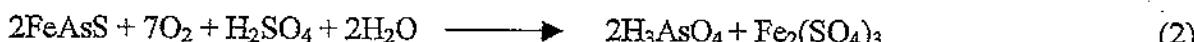
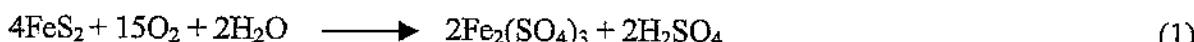
Biooxidation is a biohydrometallurgical process for pre-cyanidation treatment of refractory gold ores. This process offers an alternative to the more conventional roasting or pressure oxidation techniques with the following advantages:

- Reduced capital costs.
- A highly flexible and easily controllable process.
- Improved recovery rates.
- A process that is environmentally friendly.

The biooxidation process utilizes a naturally occurring mixed bacterial consortium comprising *Thiobacillus ferrooxidans*, *Thiobacillus Thiooxidans* and *Leptospirillum ferrooxidans*. These bacteria, under controlled conditions, oxidize gold-bearing sulfide ores or concentrates due to a chemolithotrophic mode of metabolism. This means that they require inorganic compounds for the acquisition of both energy and carbon. Inorganic substrates such as sulfides, elemental sulfur, and ferrous iron are oxidized by the bacteria to provide chemical energy. This is enzymatically converted, by oxidative phosphorylation, to ATP, a form of metabolic energy utilized by the bacteria for various cellular functions. The carbon requirements of the bacteria for biosynthesis of cellular biomass are met by CO₂ in the atmosphere or from dissolution of carbonate minerals in the ore.

The mixed bacterial population can oxidize a wide range of metal sulfide minerals. These include pyrite, arsenopyrite, pyrrhotite, chalcopyrite, chalcocite, covellite, stibnite, pentlandite and galena. The mechanism of sulfide breakdown is usually a combination of direct enzymatic attack and indirect chemical attack by metabolic by-products of the bacteria.

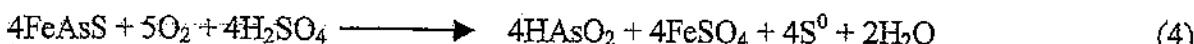
For direct enzymatic oxidation, attachment of the bacteria to the sulfide mineral is essential. Under ideal conditions bacterial oxidation of pyrite and arsenopyrite are illustrated in the following equations:



The ferric sulfate produced contributes to further sulfide breakdown by indirect chemical attack:



Partial arsenopyrite oxidation may also occur by acid attack:



The arsenic acid produced from oxidation of arsenopyrite is effectively neutralized with lime to form non-polluting ferric arsenate precipitates, which have proved to be stable provided the Fe:As molar ratio in the biooxidation liquor is greater than 3:1.

The Ferrous sulfate and elemental sulfur are then bacterially re-oxidized to ferric sulfate and sulfuric acid, respectively:



Under controlled continuous process conditions, the number of bacterial cells and their activity is optimized to attain the highest rate of sulfide oxidation. Detailed laboratory and pilot plant studies have indicated that the bacteria require a very acidic environment (pH 1.2 to 1.8), a temperature of between 30 °C to 45 °C and a steady supply of oxygen and carbon dioxide for optimum growth and activity. The optimum conditions for the bacteria are not favorable for the growth of most other microbes, thus eliminating the need for sterility during the biooxidation process.

The NPC sample that OXIDOR tested presented issues not usually encountered. The natural redox potential of the sample fell below 620 mV E_h. Instead of generating As⁺⁵ as shown in equation 4, a competing side reaction under low redox conditions generates As⁺³. Arsinite, As⁺³, is toxic to the bacteria. The bacteria must be adapted to high arsenite levels.

Batch biooxidation amenability tests evaluate the amenability of a concentrate or an ore to biooxidation and identifies potential process problems prior to pilot testing.

2.0 EXPERIMENTAL PROCEDURE

2.1 Procedure Summary

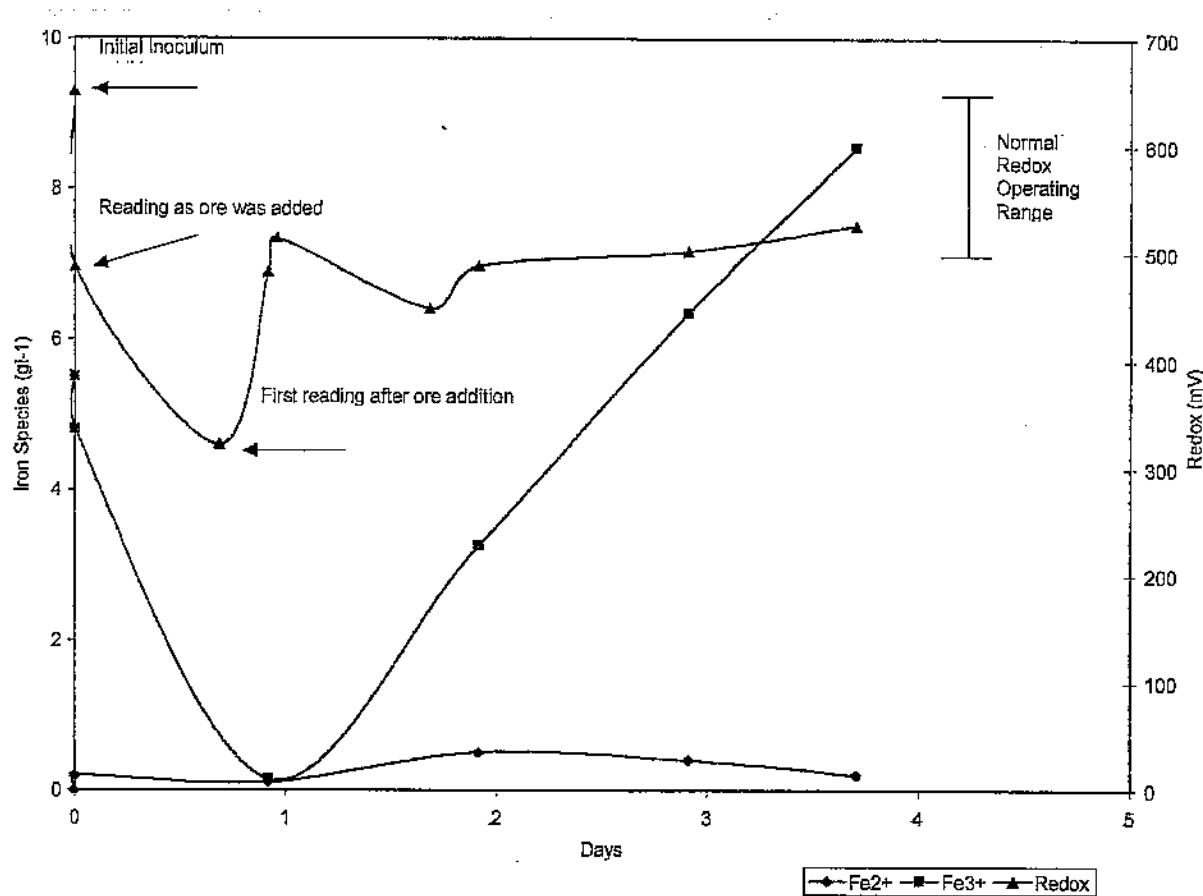
The NPC sample was prepared by the client and was split as received without further milling. Head assays for gold, silver, sulfide and iron were performed prior to each phase of test-work. Oxidative pretreatment using bio-oxidation was carried out to improve leach recoveries. During the oxidation phase, following inoculum build-up, numerous sample splits were treated to achieve incremental oxidation levels for determination of leach improvement. Following oxidation, recovery was determined using the carbon-in-leach procedure. Baseline tests were performed concurrently to determine pre-oxidation recovery. Oxidation test results were presented based on iron dissolution (from the bio-oxidation procedure) and leach test results were presented based on the entire calculated gold grade accounted for in each test (vice a single head grade).

2.2 Bacterial Adaptation

A twenty-liter air sparged CSTR vessel was used for bacterial adaptation and inoculum build-up. Adaptation was initiated using an active mixed culture designated as OXL-1014-R-13. The build up was atypical, undoubtedly owing to NPC's high arsenopyrite content. When NPC was added to OXIDOR's specifically selected inoculum, the slurry's redox potential fell from 825 mV E_h to 590 mV E_h in two days. No bacterial activity was observed. By day three oxygen uptake, indicating bacteria activity, revived. By day 10, bacterial activity was unusually high. See Figure No. 1.

Ferric and ferrous iron concentrations, temperature, dissolved oxygen, pH and redox potential were determined on a daily basis. Adjustments to the slurry pH were accomplished by using a limestone slurry or concentrated sulfuric acid as required.

Figure No. 1 - Reactor Operation Plot Showing Redox Transients in Arsenopyrite System



3.0 RESULTS

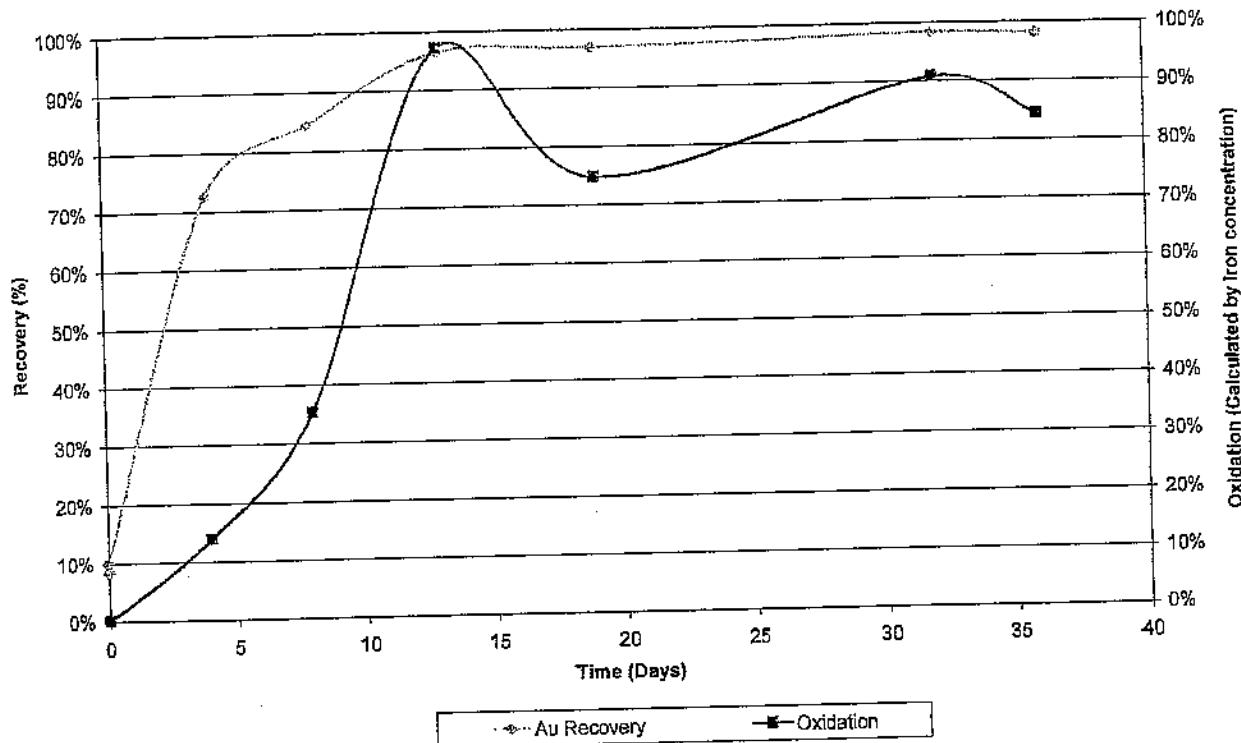
3.1 New Polaris Concentrate Sample Characterization

Table No. 1 – Analysis of Sample

Component	Client-Supplied	OXIDOR
Au, g mt ⁻¹	91.84	93.96
Ag, g mt ⁻¹	11.96	12.94
As, %	14.72	
Sb, ppm	518	
Cu, ppm	754	
Fe, %	24.0	
Hg, ppm	<0.05	
Pb, ppm	94	
Zn, ppm	266	
C _{inorganic}	0.93	
S _{total}	20.3	
S ²⁻	19.6	

3.2 Biooxidation Amenability

Figure No. 2 – Sulfide Oxidation and Gold Recovery vs. Time

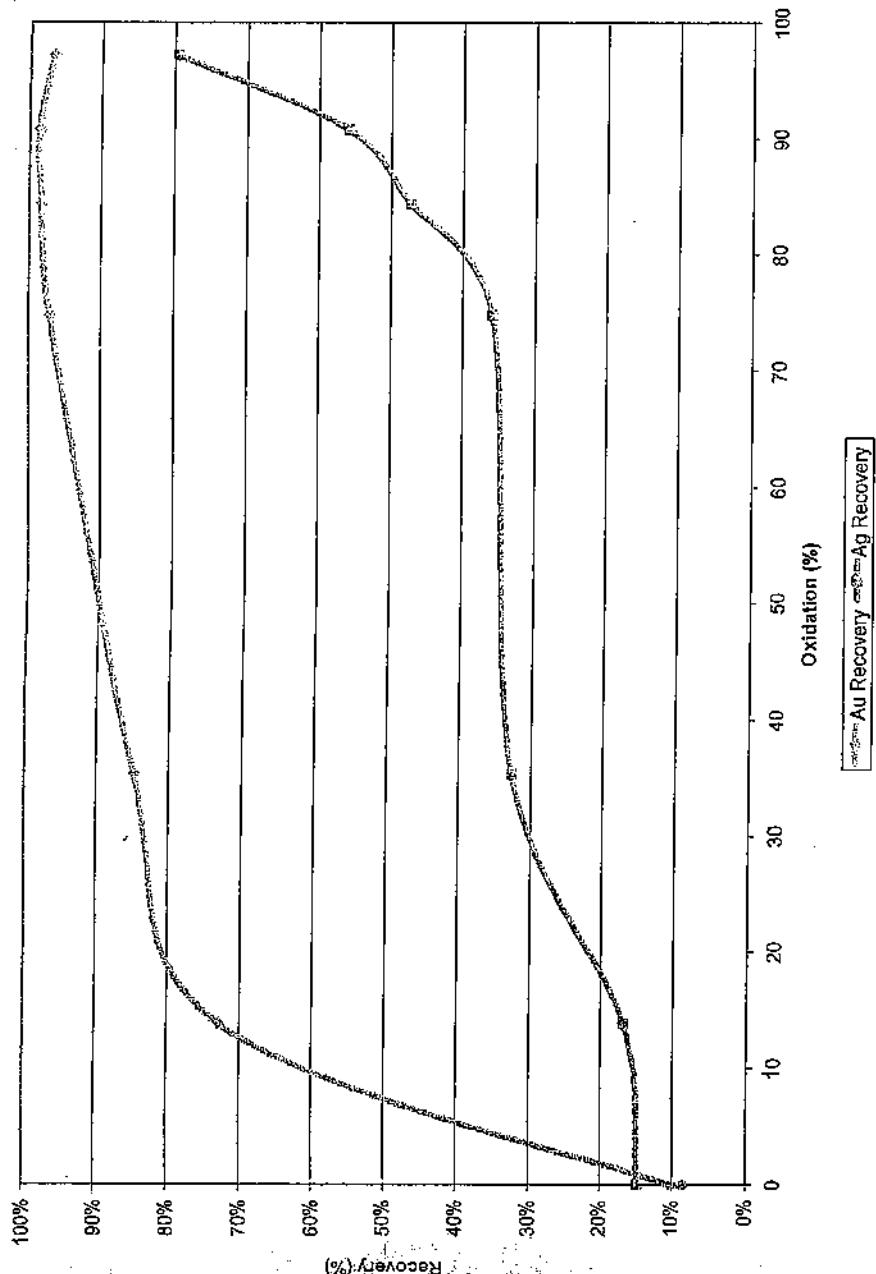


Results show that the New Polaris Concentrate sample is amenable to biooxidation treatment, and that the biooxidized residues are amenable to cyanidation and CIL treatment. Gold recovery from the NPC sample improved from the baseline 8.2% to 90 % after 9 days of biooxidation treatment. See Figures No. 2 and 3.

The residence time required for a continuous biooxidation process should be determined by pilot testing, but excellent kinetics are indicated. Higher effective rates of oxidation are achieved in continuous biooxidation, principally because of a greater and more active bacterial population in the process.

Figure No. 3 - Gold Recovery vs. Sulfide Oxidation Based on Calculated Head

175
176



4.0 CONCLUSIONS & RECOMMENDATIONS

The New Polaris Concentrate sample is very amenable to biooxidation pretreatment. Batch oxidation rates are rapid and the resultant improvement in gold recovery is excellent.

OXIDOR recommends that a continuous biooxidation test-work program be implemented in order to generate engineering data for a feasibility analysis. To guarantee engineering data, OXIDOR would need a minimum of 100 kg of material.



BIOTECHNOLOGY DIVISION

PRELIMINARY TEST WORK REPORT

DIAGNOSTIC AND BIOLEACH TESTING OF A SAMPLE OF REFRACTORY GOLD CONCENTRATE FROM THE NEW POLARIS DEPOSIT, LOCATED IN NORTHWESTERN BRITISH COLUMBIA, CANADA

Compiled for
CANARC RESOURCE CORPORATION

by
JOHN NEALE & MARIEKIE GERICKE

17 APRIL 2003

CONFIDENTIAL

Inhibitory substance
Elemental S?
Stibnite?
ash bed M.
cause of high O₂ consumption in solids?
and consumption?

EXECUTIVE SUMMARY

177 Set of preliminary, entry-level metallurgical tests was conducted on a sample of a refractory gold flotation concentrate from the New Polaris deposit, located in north-western British Columbia, Canada.

178 The objective of these tests was to evaluate the suitability potential for oxidising the sulphides in the concentrate using a bioleaching process. The test work included a set of diagnostic leach tests, as well as a set of preliminary batch bioleach tests. It is important to note that these entry-level tests must be viewed as a precursor to a more detailed assessment of the amenability of the concentrate to bioleaching.

Assaying of the concentrate showed that it contains 31.1 % arsenopyrite (FeAsS) and 25.4 % pyrite (FeS_2). It can therefore be characterised as a high-arsenic concentrate. The particle size of the as-received concentrate was measured as $77.3 \mu\text{m}$.

Diagnostic leaching of the concentrate using nitric acid showed that, following dissolution of the sulphides, up to 93.7 % of the gold could be recovered using cyanide, compared with just 17.6 % by direct cyanidation of the unoxidised concentrate. Relatively large doses of cyanide were required to achieve this dissolution level on the oxidised residue.

The preliminary batch bioleach testing showed that very high levels of iron and arsenic extraction were achieved on the concentrate, although the tests indicated that there may be an inhibitory substance associated with the concentrate. The results are considered suitably encouraging to recommend that further small-scale bioleach testing of the concentrate should be undertaken, using the remainder of the sample that was supplied for these tests.

1. INTRODUCTION

178 sample of approximately 12 kg of a refractory gold flotation concentrate from the New Polaris deposit, located in north-western British Columbia, Canada, was delivered to Mintek by RDI Inc.

A preliminary set of test work was conducted on sub-samples of the concentrate, in order to evaluate the suitability of this material for bioleaching. The test work included a set of diagnostic leach tests, which were aimed at evaluating the improvement in gold dissolution with increasing levels of sulphide oxidation. In addition, preliminary bioleach tests were conducted to assess the potential for oxidising the sulphides using a bioleaching process.

It must be noted that these tests are considered entry-level tests, and are merely a precursor to a more detailed assessment of the amenability of the concentrate to bioleaching.

2. CHARACTERISATION OF THE CONCENTRATE

2.1. Chemical Analysis

Several sets of chemical assays were undertaken on the concentrate sample. Initially, a limited set of data was supplied by the client, based on assays undertaken during the flotation test work. A further, more comprehensive set of assays was undertaken by the client at a later stage, and the results were supplied to Mintek. Limited chemical analyses were also carried out at Mintek, as part of the diagnostic leach procedure. All of these results are summarised in Table 2.1.

Table 2.1. Summary of chemical assay data

Element	Client-supplied		Mintek	Units
	Initial	Detailed		
Au	94.2	92.18	89.6/90.2*	g/t
Ag	42.7	10.08	8.55/8.56*	g/t
Fe	-	24.0	-	%
S _{total}	-	20.3	-	%
S ²⁻	-	19.6	19.7	%
As	11.9	4.72	14.3	%
Sb	-	518	-	mg/kg
Cu	-	754	-	mg/kg
Zn	-	266	-	mg/kg
Pb	-	94	-	mg/kg
Hg	-	<0.05	-	mg/kg

* Duplicate assays performed

There are a few notable discrepancies in these data:

- The original silver assay is clearly at variance with the value measured subsequently, and the value determined at Mintek.

- There is considerable variance in the arsenic assays. The original value provided by the client is of the same order as the value measured at Mintek, but the subsequent value provided by the client is considerably lower.
- 179
180

There is, however, good agreement in the values reported for gold and sulphide-sulphur. For the purposes of this report, the gold, silver, sulphide-sulphur and arsenic values determined at Mintek will be used. Based on these analyses, it is estimated that the concentrate contains 31.1 % arsenopyrite (FeAsS), 25.4 % pyrite (FeS₂), and that the sulphide-associated iron grade is 22.5 %. Non-sulphide-associated iron therefore comprises 1.5 %.

The values of silver, antimony, lead and mercury present in the concentrate are all low, and are not expected to pose any problems in the bioleaching process. Ideally, a semi-quantitative scan of the concentrate should be conducted to confirm that there are no potentially inhibitory elements present in the concentrate. This was requested, but not provided, and such a scan should be undertaken if any further bioleach testing of the concentrate is performed. An XRF powder analysis of the ore from which the concentrate was derived was provided by the client, and this analysis also did not indicate the presence of any potential bacterial inhibitors.

2.2. Physical Analysis

The particle size of the concentrate was measured at Mintek, and a d₉₀ of 77.3 µm was determined. The following metallurgical test programme was conducted on the concentrate as received, with no regrinding being undertaken. Any future testing should evaluate the effect of regrinding on the kinetics of sulphide oxidation, and the overall gold dissolution levels attainable.

3. DIAGNOSTIC LEACH TESTS

3.1. Methods

A set of diagnostic leach tests was conducted on the as-received concentrate. The set of tests comprised the following:

- ◆ A CIL test on the as-received concentrate, using excess reagents.
- ◆ A dilute nitric acid digestion of the as-received concentrate (10 % HNO₃, 70 °C, 4 hours), followed by a CIL test on the residue, using excess reagents (20 g/L carbon, 5 kg/t NaCN, 24 hours).
- ◆ A concentrated nitric acid digestions of the as-received concentrate (27.5 % HNO₃, 70 °C, 4 hours), followed by a CIL test on the residues, using excess reagents (20 g/L carbon, 5 kg/t NaCN, 24 hours). However, an additional concentrated nitric acid digestion was performed after it was noted that the free cyanide levels at the end of the CIL test was low. The residue from this digestion was subjected to

a CIL test under the same conditions as before, but with 50 kg/t NaCN addition. Sulphide and arsenic assays were not repeated on the nitric-leach residue.

**180
181**

3.2. Results and Discussion

The following tables summarise the results of the diagnostic leach tests.

Table 3.1. CIL test on as-received concentrate

	As-received concentrate	CIL Residue
Mass (g)	300.00	
NaCN addition (kg/t)	5.00	
NaCN consumption (kg/t)		2.70
Au (g/t)	89.6	73.8
Ag (g/t)	8.55	4.45
S ²⁻ (%)	19.7	
As (%)	14.3	
Au dissolution (%)		17.6
Ag dissolution (%)		48.0

Table 3.2. CIL test on dilute nitric acid leach residue

	As-received concentrate	10 % HNO ₃ residue	CIL residue
Mass (g)	300.00	241.67	
Mass loss (%)		19.4	
NaCN addition (kg/t)		5.00	
NaCN consumption (kg/t)			4.85
Au (g/t)	89.6	108	42.9
Ag (g/t)	8.55	10.1	8.62
S ²⁻ (%)	19.7	18.1	
As (%)	14.3	10.2	
S ²⁻ dissolution (%)		26.0	
As dissolution (%)		42.5	
Au dissolution (%)			60.3
Ag dissolution (%)			14.7

Table 3.3. CIL test on concentrated nitric acid leach residue: 5 kg/t NaCN

181

	As-received concentrate	27.5 % HNO ₃ residue	CIL residue
Mass (g)	300.00	130.18	
Mass loss (%)		56.6	
NaCN addition (kg/t)		5.00	
NaCN consumption (kg/t)			4.90
Au (g/t)	89.6	195	54.7
Ag (g/t)	8.55	12.6	6.26
S ²⁻ (%)	19.7	1.66	
As (%)	14.3	0.79	
S ²⁻ dissolution (%)		96.3	
As dissolution (%)		97.6	
Au dissolution (%)			71.9
Ag dissolution (%)			50.3

Table 3.4. CIL test on concentrated nitric acid leach residue: 50 kg/t NaCN

	As-received concentrate	27.5 % HNO ₃ residue	CIL residue
Mass (g)	400.00	262.00	
Mass loss (%)		34.5	
NaCN addition (kg/t)		50.0	
NaCN consumption (kg/t)			49.3
Au (g/t)	90.2	136	8.60
Ag (g/t)	8.56	8.61	7.78
Au dissolution (%)			93.7
Ag dissolution (%)			9.6

Even in the repeated test, with 50 kg/t of NaCN addition, most of the cyanide was consumed, and it may be that higher doses of NaCN addition will yield even higher gold-dissolution levels.

The following graph shows the relationships between the levels of sulphide-sulphur and arsenic oxidation, and gold dissolution, achieved in these diagnostic leach tests. However, the gold-dissolution level obtained on the dilute nitric acid leach residue is almost certainly lower than can be expected at a higher NaCN dose.

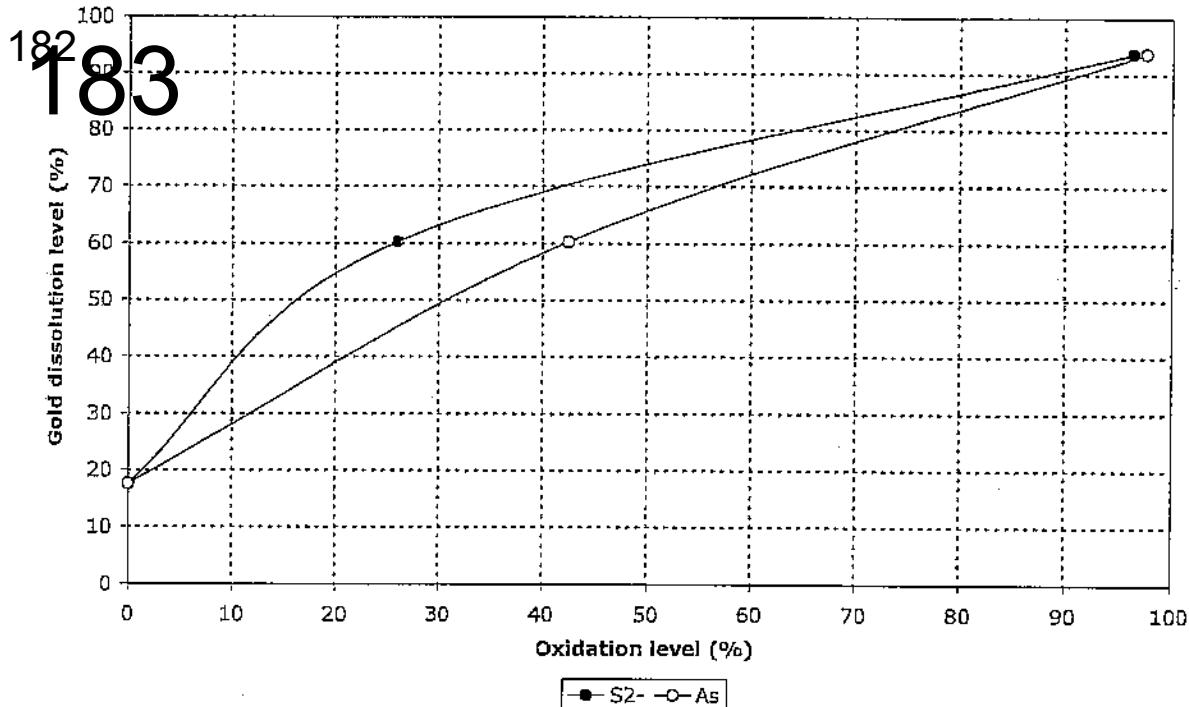


Figure 3.1. Sulphide oxidation-gold dissolution relationship

The following points arise from an examination of these results:

- ◆ The concentrate is highly refractory, with just 17.6 % of the gold recoverable by direct cyanidation of the concentrate.
- ◆ A partial oxidation of the sulphides was achieved in a dilute nitric acid leach test, with 42.5 % of the arsenic and 26.0 % of the sulphide being oxidised. Gold dissolution increased to around 60 % after this level of oxidation, but a higher level of dissolution may be possible by the addition of more cyanide.
- ◆ A virtually complete oxidation of the sulphides was achieved in a concentrated nitric acid leach test, with 97.6 % of the arsenic and 96.3 % of the sulphide being oxidised.
- ◆ For the concentrated nitric acid leach residues, gold dissolution was a strong function of cyanide addition. At a NaCN addition of 5 kg/t, a gold dissolution of around 72 % was obtained. Increasing the NaCN addition to 50 kg/t resulted in a gold dissolution level of almost 94 % being attained. Even at this very high NaCN addition, virtually all of the cyanide was consumed, and further increases in NaCN addition may be required to improve the gold dissolution further.
- ◆ The high levels of NaCN consumption on the nitric acid leach residues indicate that this aspect warrants careful attention during any further bioleach testing that is conducted on the concentrate.
- ◆ The levels of silver dissolution were erratic, with no discernible trend being observed. However, since the silver represents approximately 0.1 % of the value in the concentrate at current bullion prices, the silver-dissolution results are of no significance.

4. BIOLEACH TESTS

183. Methods

184. The bioleach tests were conducted in mechanically agitated, aerated 1-litre reactors. The impeller speed was set at 450 r/min. The air supply was enriched with 0.3 % CO₂. The tests were performed on the 'as-received' concentrate at a 10 % solids concentration.

Active mesophilic bacteria, harvested from maintenance reactors treating pyrite-arsenopyrite concentrates, were used as the inoculum.

The progress of the bioleach was followed by daily monitoring of the pH and redox potential levels. Liquor samples were removed from each reactor daily, and the soluble iron and arsenic extractions determined. Pulp samples were also removed and contacted with hydrochloric acid, to re-dissolve any precipitates formed. The samples were filtered and the filtrates analysed for iron and arsenic. The filtered residues were returned to the reactors.

4.2. Results and Discussion

4.2.1. Control Test

This test was conducted to indicate the level of extraction under controlled conditions, where no bacterial activity is possible. The progress of events during this test is summarised below.

Table 4.1. Progression of the bioleach control test

Day	Comments
1	Start-up and acid conditioning at 40 °C. Initial pH level set at 1.8.
2	Add 20 mL thymol (stock solution = 5 % thymol in ethanol) to prevent bacterial activity.
9	Lower temperature to 37 °C. Similar to test 1 (see below).
15	Stop test.

The routine measurements during the course of this test are depicted in the following four graphs.

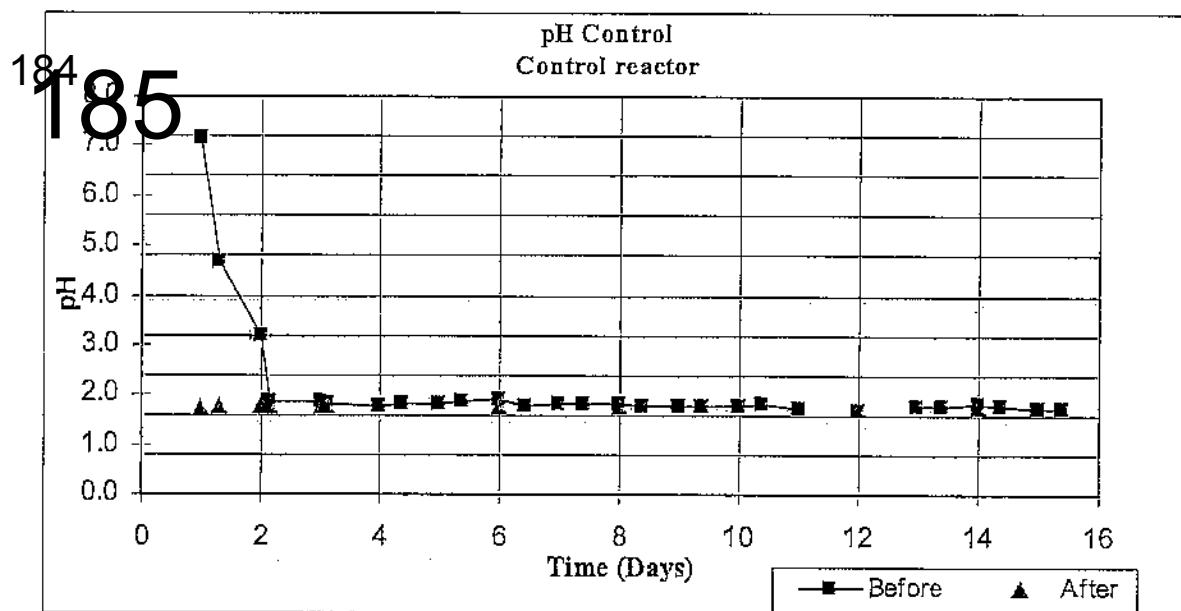


Figure 4.1. pH control: control test

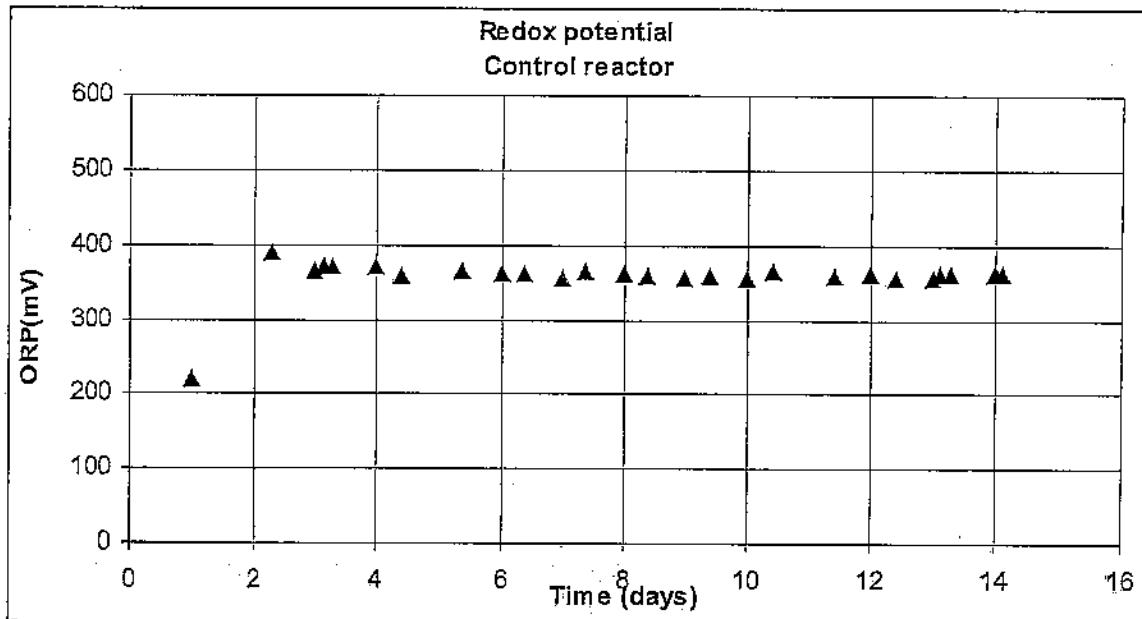


Figure 4.2. Redox potential: control test

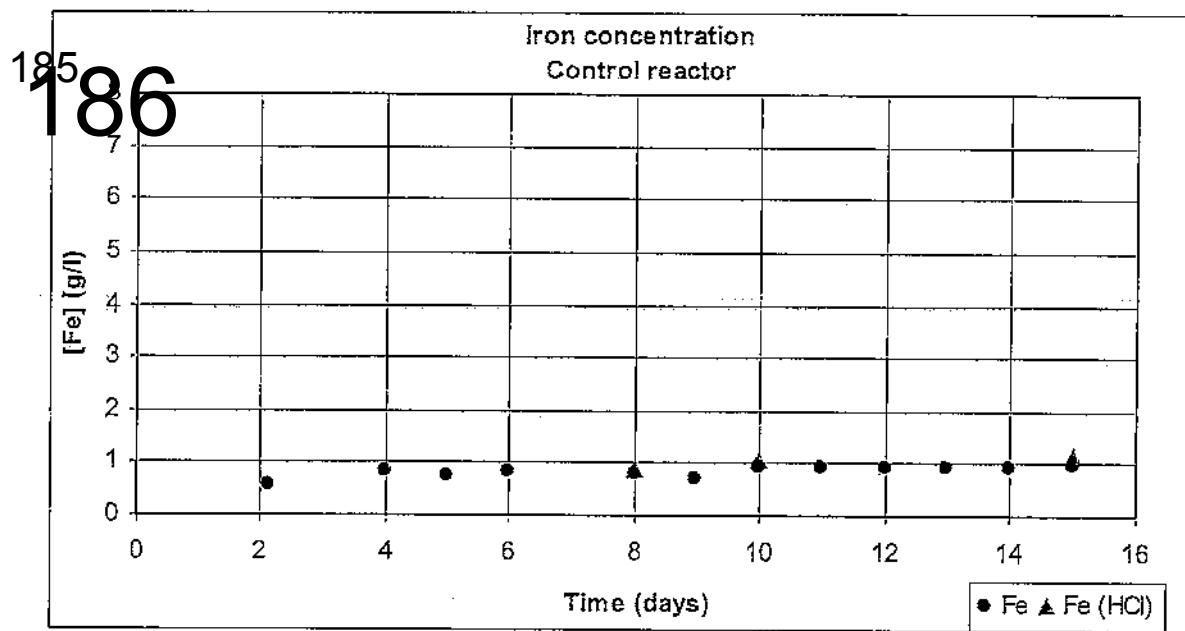


Figure 4.3. Fe concentrations: control test

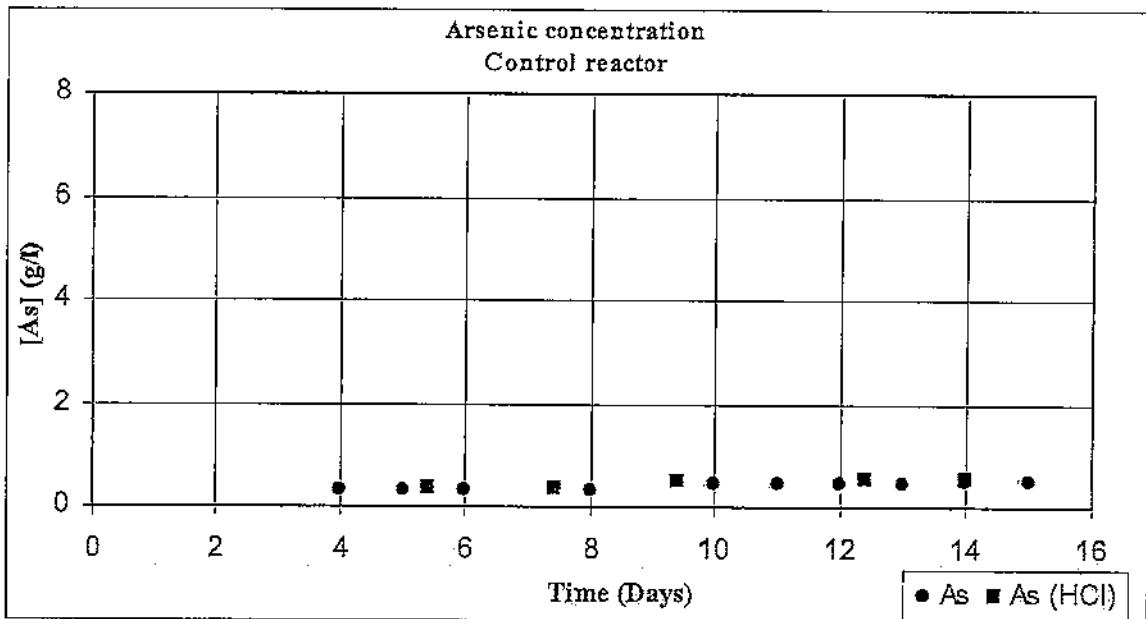


Figure 4.4. As concentrations: control test

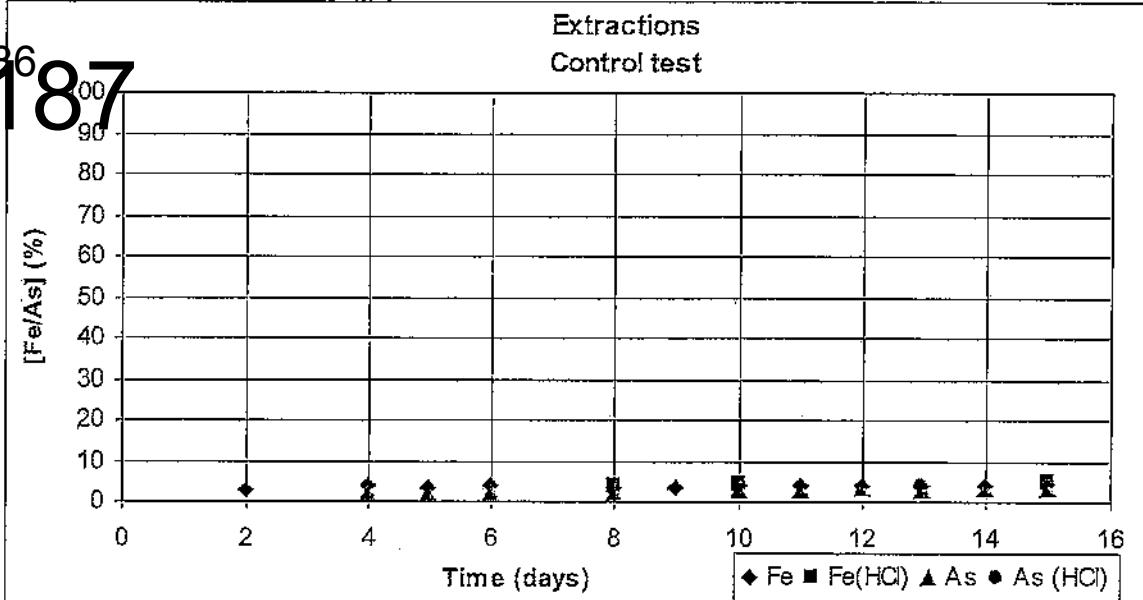


Figure 2.5. Fe and As extractions: control test

Note: Fe extractions based on sulphide-associated Fe

Acid addition during this test was 133 kg/t.

Foaming was experienced for the duration of test. It is possible that the thymol could have contributed to the foaming problem.

The redox potential remained at a level of around 380 mV for the duration of this test. As expected, very low extractions were obtained. The total iron extraction was about 5 %, and the total arsenic extraction was about 4 %.

4.2.2. Bioleach test 1

The summary of events for this test is listed below.

Table 4.2. Sequence of events for bioleach test 1

Day	Comments
1	Start-up at 40 °C. Acid conditioning (initial pH level set at 1.8).
2	Inoculate.
6	Re-inoculate.
9	Lower temperature to 37 °C.
12	Re-inoculate.
15	Remove reactor content. Separate solids and bacteria from liquor by centrifugation. Re-suspend solids and bacteria in 0K before adding back to reactor. Objective: to remove possible inhibitory substances (such as As (III), flotation reagents) from solution.
16	Re-inoculate.
27	Stop test.

The routine measurements during the course of this test are depicted in
the following four graphs.

188

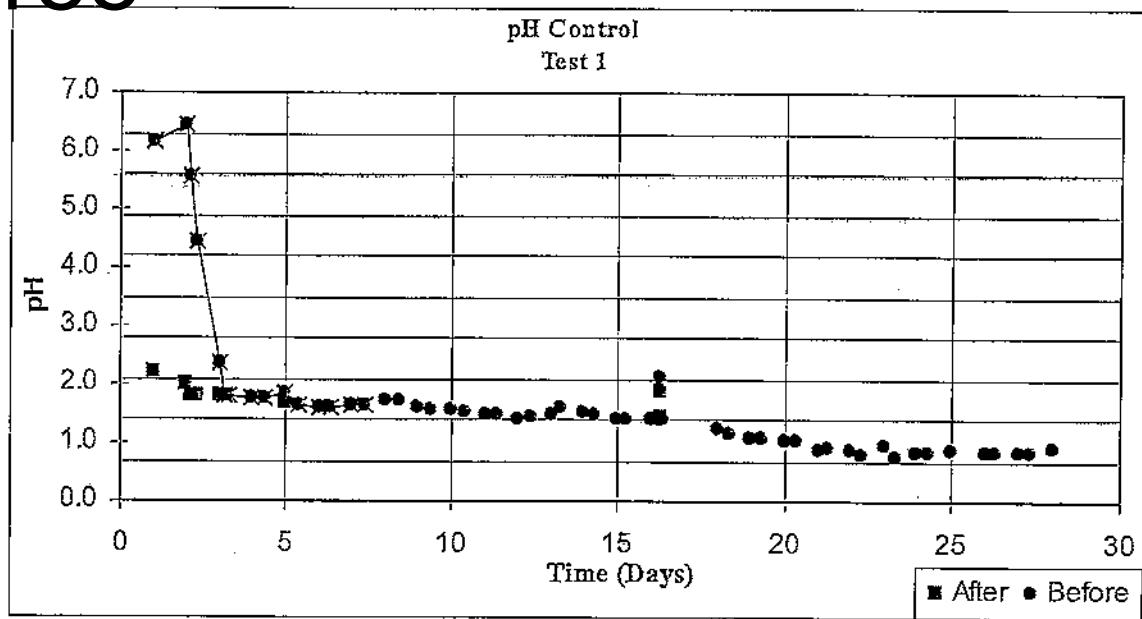


Figure 4.6. pH control: bioleach test 1

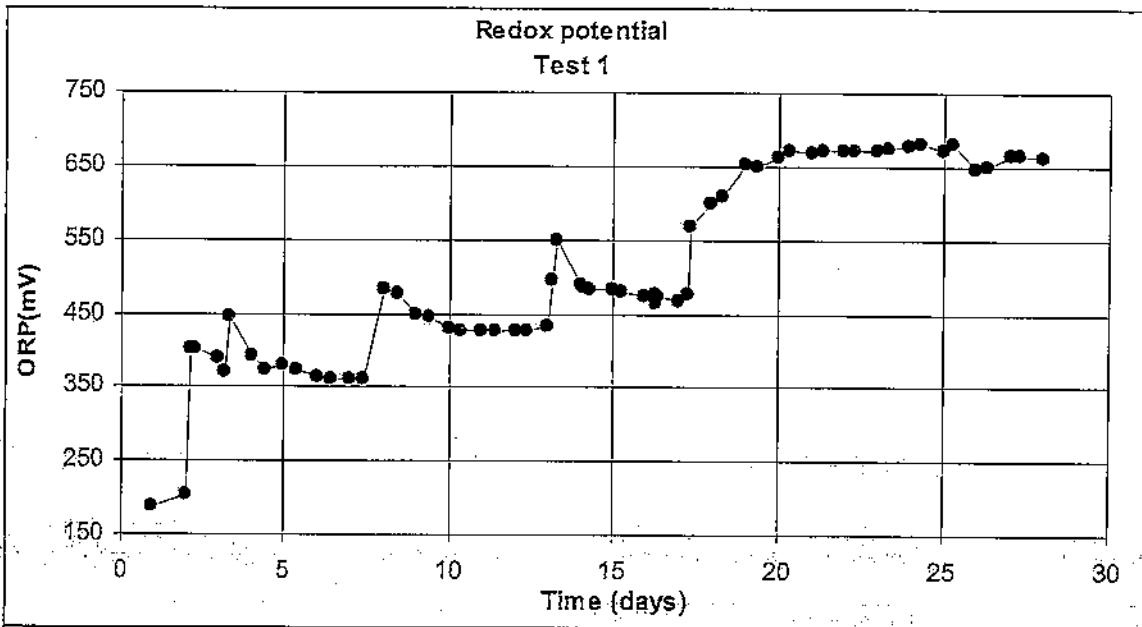


Figure 4.7. Redox potential: bioleach test 1

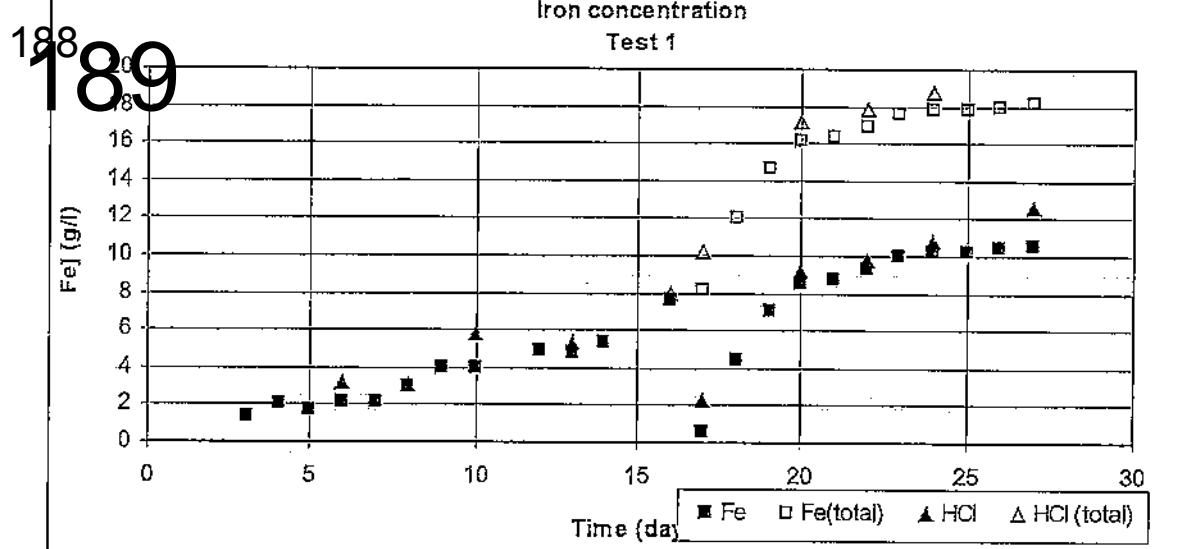


Figure 4.8. Fe concentrations: bioleach test 1*

* Fe (total): Sum of Fe measured before and after replacing the bioleach liquor

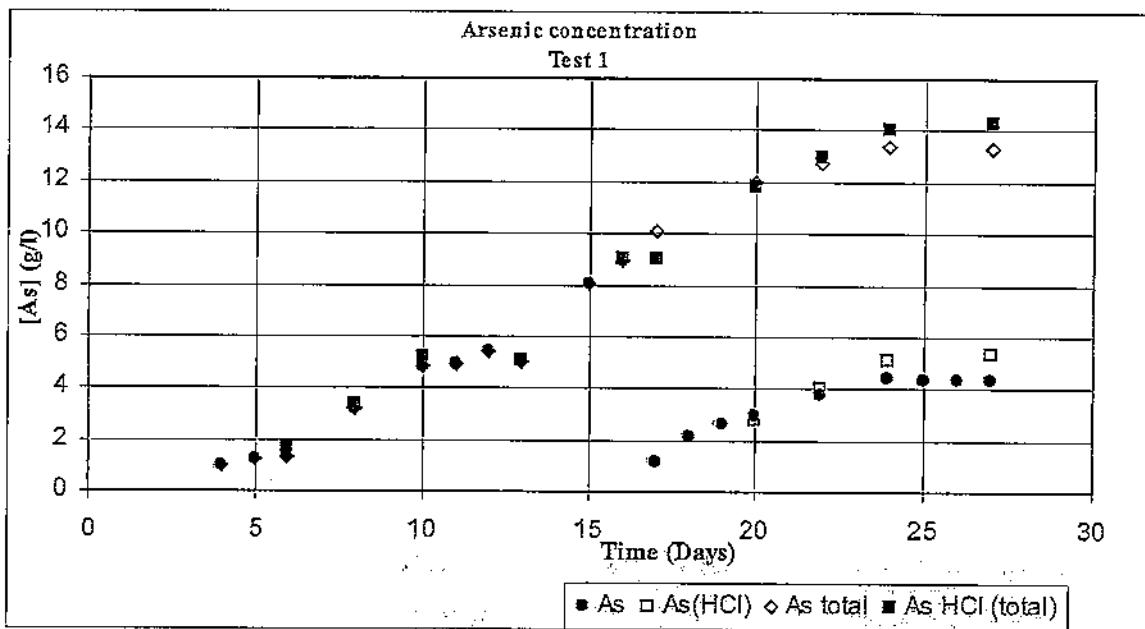


Figure 4.9. As concentrations: bioleach test 1*

* As(total): Sum of As concentrations measured before and after replacing bioleach liquor

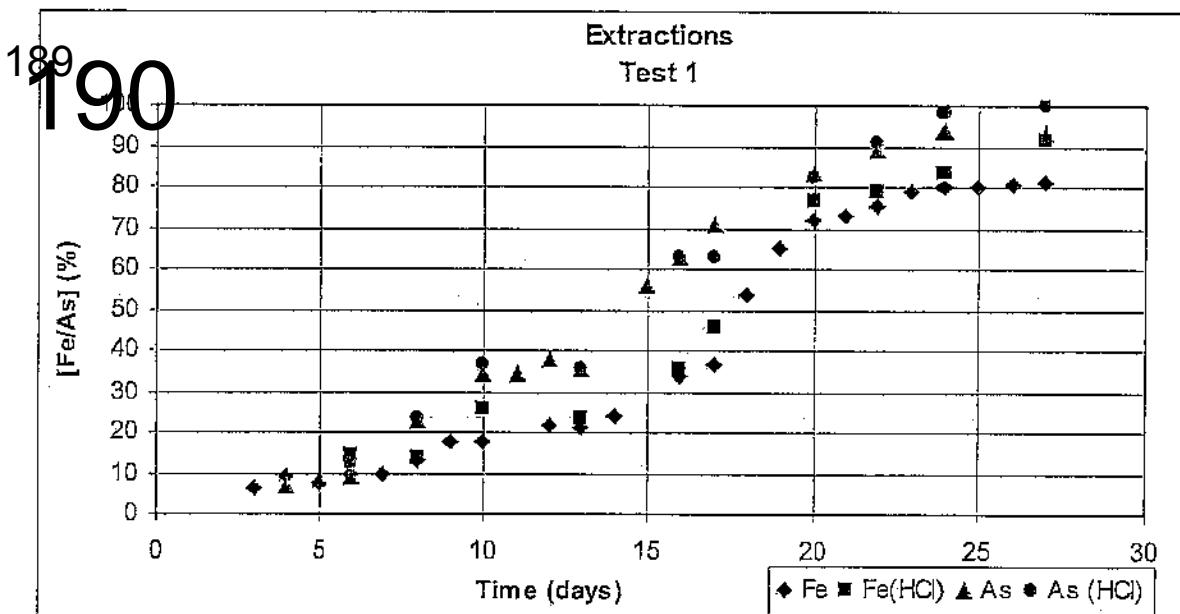


Figure 4.10. Fe and As extractions: bioleach test 1

Note: Fe extractions based on sulphide-associated Fe

Acid addition during this test was 150 kg/t.

High bacterial cell concentrations were observed microscopically, even during the stages where low redox potential levels were recorded.

The estimated levels of iron and arsenic oxidation at the completion of the test were as follows:

Soluble Fe: 81.3 %

Total Fe: 91.4 %

Soluble As: 93.2 %

Total As: 100.0 %

However, as seen in Figure 4.10, these high levels of oxidation were only achieved after dilution and re-inoculation of the test. It is interesting to note the redox potential profile in Figure 4.7. It shows that, prior to the dilution on day 15, each re-inoculation caused a spike in the redox potential, followed by a fairly rapid decline. This suggests that an inhibitory substance may be present in the concentrate, and that it is rapidly being leached, leading to an inhibition of bacterial and oxidative activity.

4.2.3. Bioleach test 2

As a result of the observations made during test 1, a second bioleach test was conducted, similar to test 1, but with pre-washing of the concentrate with acetone, followed by an acid-ferric wash (using 5 g/L ferric in H₂SO₄, at pH level of 1.4), prior to the bioleach test. The objective of the pre-treatments was to remove any potentially inhibitory materials on the surface of the concentrate, and to remove any readily leachable

arsenopyrite that may be creating a high concentration of As(III) at the onset of the bioleach test.

191

The summary of events for this test is listed below.

Table 4.3. Sequence of events for bioleach test 2

Day	Comments
1	Start-up test with pre-washed solids, 10 % solids.
2	Inoculate, 37 °C.
9	Stop test.

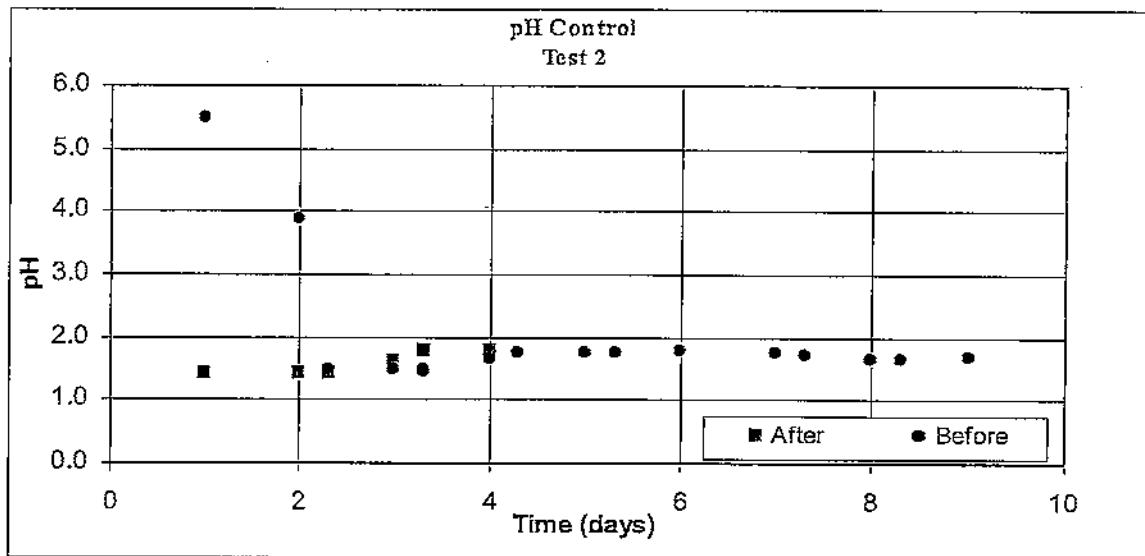


Figure 4.11. pH control: bioleach test 2

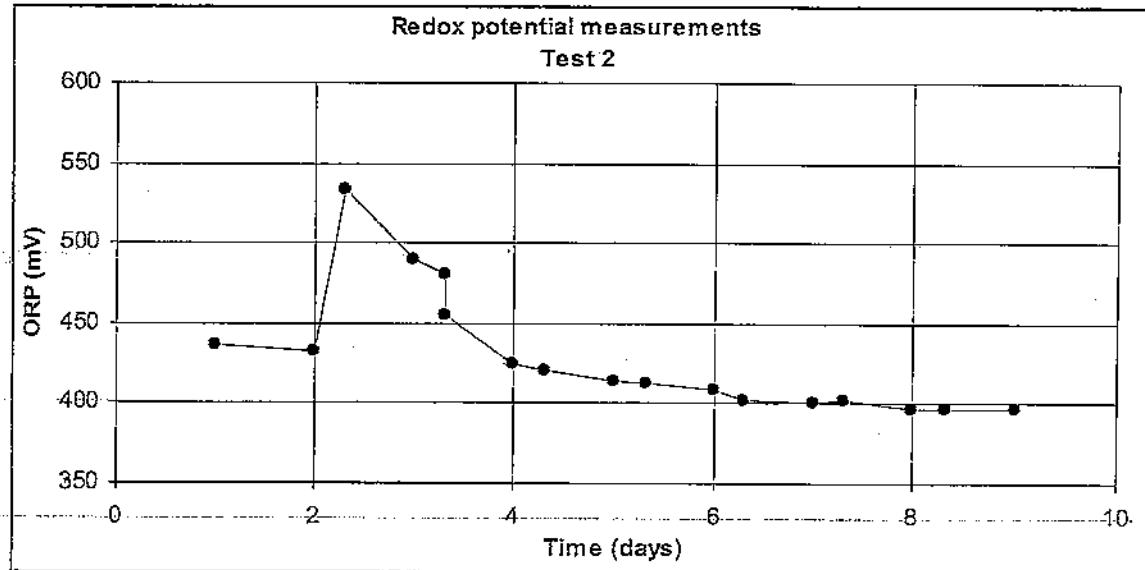


Figure 4.12. Redox potential: bioleach test 2

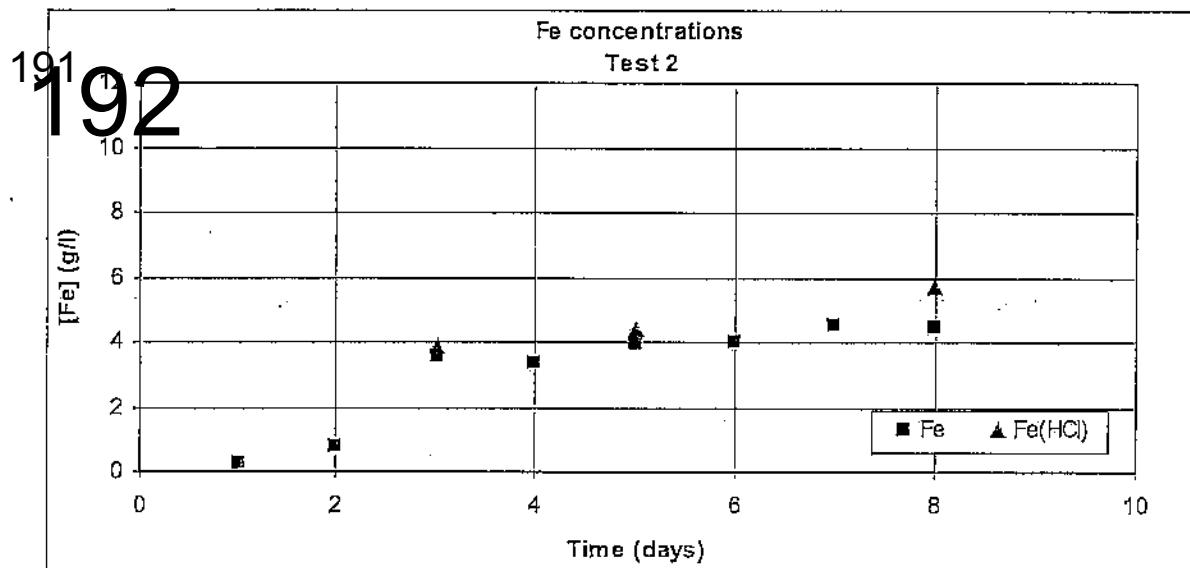


Figure 4.13. Fe concentrations: bioleach test 2

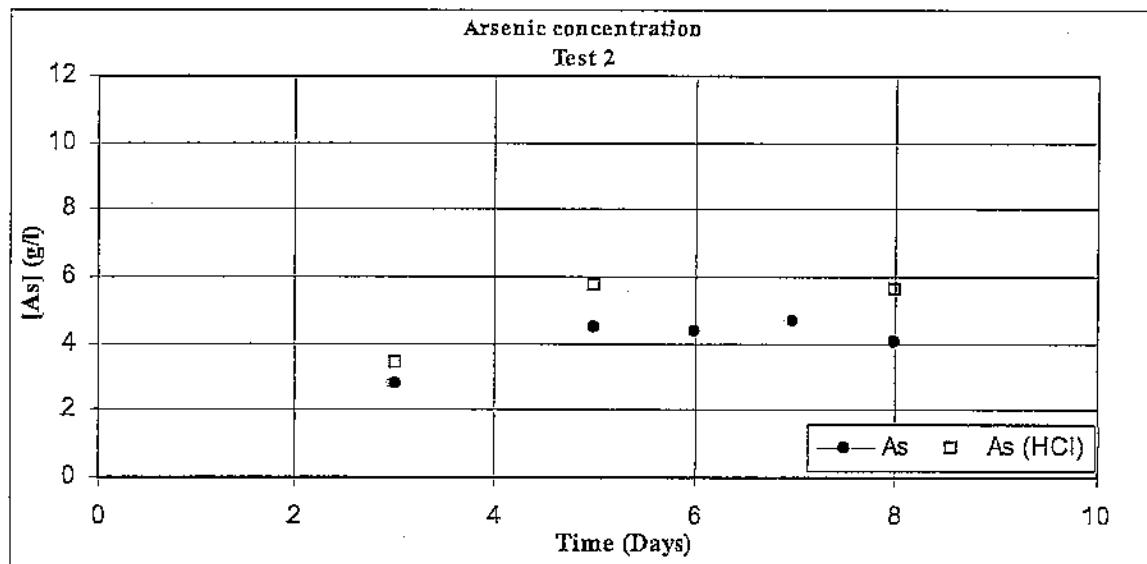


Figure 4.14. As concentrations: bioleach test 2

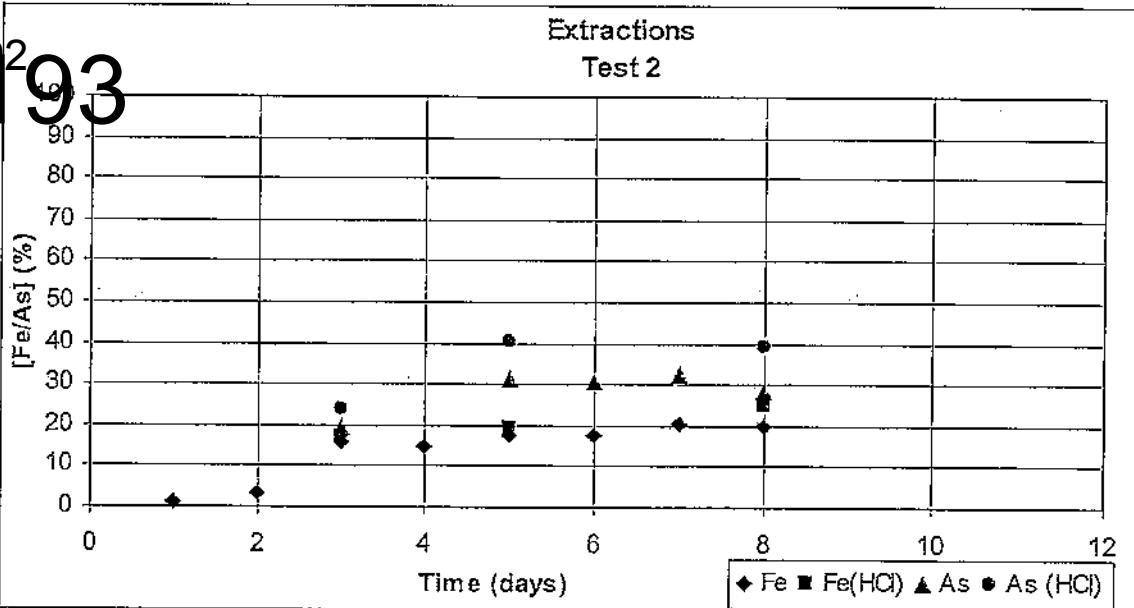


Figure 2.15. Fe and As extractions: bioleach test 2

Note: Fe extractions based on sulphide-associated Fe

Directly after inoculation, the redox potential increased to 532 mV. The redox potential, however, dropped to a level of about 400 mV by day 9. Both the Fe and As concentrations increased while the redox potential was high, but both reached a plateau as the redox potential decreased. This test exhibited a similar trend to that of test 1, and so it was terminated after nine days.

The results of this test confirm that bacterial activity is apparently inhibited by an element appearing in solution during the bioleach test. While the presence of As(III) is considered a possibility, it is not considered likely based on previous test work on similar concentrates using mesophilic bacterial cultures.

5. CONCLUSIONS AND RECOMMENDATIONS

- ◆ Nitric acid digestion of the concentrate showed that the sulphides are readily oxidised.
- ◆ Cyanide leaching of the nitric acid digestion residues showed that very high doses of cyanide are required to achieve high levels of gold dissolution. This aspect warrants further attention in any further test programme.
- ◆ At a NaCN addition of 50 kg/t, a gold dissolution level of 93.7 % is achieved after oxidation of the majority of the sulphides using nitric acid. This is considerably higher than the level previously achieved using a pyrometallurgical diagnostic procedure on an ore sample.
- ◆ The New Polaris concentrate is amenable to bioleaching, with high levels of iron and arsenic oxidation being demonstrated in these entry-level tests. However, there are indications that there may be an inhibitory substance associated with the concentrate, which may or may not influence a continuous process.

◆ This type of behaviour has been observed previously in batch test work conducted at Mintek. However, continuous leaching of the concentrates in question has been possible.

- 193** **194**
- ◆ These preliminary, entry-level test results are considered sufficiently positive to suggest that a more detailed programme of bioleach testwork is justified, to evaluate the potential of a bioleach processing option for treating New Polaris concentrate.
 - ◆ To this end, further testing in continuous operation is recommended. Such testing can be carried out at Mintek on small quantities of concentrate (a few kilograms would be required.) This phase of test work should evaluate the effect of regrinding of the concentrate to a suitable particle size. Regrinding will have the benefit of improving the rate of pyrite oxidation, as may improve the overall gold recovery by liberating some of the gold that may remain locked in quartz. The test work should also include a preliminary evaluation of gold and silver recovery from the bacterially-oxidised residues.
 - ◆ This test work programme can be conducted using Mintek's micro-scale continuous bioleach reactors. The existing sample of concentrate that Mintek has (of which about 8 kg are left) can be utilised for this test programme. A budget estimate of the costs of such a programme of work is between US\$25,000 and US\$30,000.
 - ◆ At the completion of this phase of test work, it is recommended that a preliminary assessment of the capital and operating costs of the proposed process be undertaken, as part of a prefeasibility assessment of the viability of the process.
 - ◆ If the test work is successful, and the prefeasibility study is positive, then a more detailed pilot plant campaign should be undertaken to establish the process design parameters for the bioleach process, and the associated downstream unit operations. These operations include solid-liquid separation, liquor neutralisation with iron and arsenic precipitation, cyanide leaching and CIP processing, cyanide-species detoxification, including waste characterisation and stability testing.
 - ◆ An alternative operating strategy that could be evaluated at this stage is to operate at a lower solids concentration in the primary bioleach reactors, and to utilise a solid-liquid separation between the primary and secondary bioleach reactors, allowing operation at a higher solids concentration in the secondary reactors.
 - ◆ Such testing would require the provision of a few hundred kilograms of concentrate, and would be conducted in Mintek's existing continuous miniplant facility.

Appendix III Assay Results

Table :Assay Results For the 2003 to 2005 New Polaris Drill Programs

Hole No.	From	To	Interval	Au (gpt)	Sample No.	AS (ppm)	SB (ppm)
03-P01	0.00	22.56	22.56	0.00	NE	0	0
03-P01	22.56	23.01	0.46	0.07	20501	369	49
03-P01	23.01	24.08	1.07	2.40	20502	3570	74
03-P01	24.08	25.60	1.52	0.10	20503	683	70
03-P01	25.60	26.49	0.88	9.57	20504	9280	2680
03-P01	26.49	27.28	0.79	12.10	20505	22900	283
03-P01	27.28	27.89	0.61	0.45	20506	2030	40
03-P01	27.89	29.20	1.31	0.03	20507	527	25
03-P01	29.20	141.67	112.47	0.00	NE	0	0
03-P01	141.67	142.07	0.40	0.00	20508	44	18
03-P01	142.07	142.95	0.88	0.00	20509	59	13
03-P01	142.95	143.74	0.79	0.00	20510	53	17
03-P01	143.74	144.48	0.73	0.00	20511	38	23
03-P01	144.48	158.80	14.33	0.00	NE	0	0
03-P01	158.80	159.78	0.98	0.00	20512	5	9
03-P01	159.78	160.93	1.16	0.03	20513	5	5
03-P01	160.93	161.94	1.01	0.00	20514	5	5
03-P01	161.94	174.71	12.77	0.00	NE	0	0
03-P01	174.71	175.41	0.70	0.00	20515	27	12
03-P01	175.41	184.40	8.99	0.00	NE	0	0
03-P01	184.40	185.07	0.67	0.10	20516	2810	34
03-P01	185.07	185.99	0.91	0.00	20517	318	9
03-P01	185.99	243.54	57.55	0.00	NE	0	0
03-P01	243.54	244.42	0.88	0.00	20518	69	18
03-P01	244.42	245.06	0.64	0.00	20519	114	12
03-P01	245.06	245.67	0.61	0.00	20520	23	12
03-P01	245.67	252.65	6.98	0.00	NE	0	0
03-P01	252.65	254.20	1.55	0.00	20521	26	13
03-P01	254.20	255.00	0.79	0.00	20522	139	13
03-P01	255.00	255.79	0.79	0.00	20523	117	18
03-P01	255.79	256.85	1.07	0.00	20524	14	12
03-P01	256.85	257.46	0.61	0.00	20525	62	14
03-P01	257.46	331.41	73.94	0.00	NE	0	0
03-P01	331.41	332.02	0.61	0.00	20526	6	9
03-P01	332.02	365.46	33.44	0.00	NE	0	0
03-P02	0.00	22.86	22.86	0.00	NE	0	0
03-P02	22.86	24.02	1.16	0.00	20527	66	13
03-P02	24.02	32.92	8.90	0.00	NE	0	0
03-P02	32.92	33.56	0.64	0.00	20545	98	29
03-P02	33.56	36.09	2.53	0.00	NE	0	0
03-P02	36.09	36.58	0.49	1.06	20528	664	24
03-P02	36.58	38.65	2.07	0.00	NE	0	0
03-P02	38.65	39.26	0.61	0.17	20529	356	29

195
196

03-P02	39.26	45.42	6.16	0.00	NE	0	0
03-P02	45.42	46.70	1.28	0.14	20530	520	23
03-P02	46.70	51.82	5.12	0.00	NE	0	0
03-P02	51.82	52.64	0.82	0.00	20531	78	13
03-P02	52.64	53.19	0.55	0.00	20532	83	19
03-P02	53.19	54.65	1.46	0.00	20533	49	24
03-P02	54.65	103.78	49.13	0.00	NE	0	0
03-P02	103.78	104.70	0.91	0.00	20534	176	17
03-P02	104.70	105.86	1.16	1.20	20535	3480	30
03-P02	105.86	107.02	1.16	6.48	20536	11600	45
03-P02	107.02	107.90	0.88	0.55	20537	1960	23
03-P02	107.90	108.94	1.04	9.91	20538	18300	104
03-P02	108.94	109.58	0.64	0.03	20539	507	42
03-P02	109.58	110.15	0.58	8.19	20540	26400	146
03-P02	110.15	110.83	0.67	15.15	20541	8340	10000
03-P02	110.83	111.56	0.73	27.19	20542	24600	452
03-P02	111.56	112.32	0.76	0.10	20543	345	110
03-P02	112.32	113.84	1.52	0.03	20544	181	31
03-P02	113.84	115.70	1.86	0.00	NE	0	0
03-P02	115.70	116.77	1.07	0.00	20546	287	15
03-P02	116.77	135.70	18.93	0.00	NE	0	0
03-P02	135.70	136.55	0.85	0.07	20547	796	18
03-P02	136.55	138.07	1.52	9.46	20548	16200	19
03-P02	138.07	139.87	1.80	0.00	20549	156	9
03-P02	139.87	140.82	0.94	2.13	20550	1285	13
03-P02	140.82	141.88	1.07	0.00	20551	215	5
03-P02	141.88	192.48	50.60	0.00	NE	0	0
03-P02	192.48	194.08	1.62	0.00	20552	5	5
03-P02	194.10	202.69	8.63	0.00	NE	0	0
03-P02	202.69	203.85	1.16	0.00	20553	9	5
03-P02	203.85	226.77	22.92	0.00	NE	0	0
03-P02	226.77	227.87	1.10	0.00	20554	44	5
03-P02	227.87	228.63	0.76	0.00	20555	72	5
03-P02	228.63	230.03	1.40	0.00	20556	5	5
03-P02	230.03	230.95	0.91	0.00	20557	5	6
03-P02	230.95	232.20	1.25	0.00	20558	5	6
03-P02	232.20	233.11	0.91	0.00	20559	5	5
03-P02	233.11	234.60	1.49	0.00	20560	8	5
03-P02	234.60	245.55	10.94	0.00	NE	0	0
03-P02	245.55	247.04	1.49	0.00	20561	34	7
03-P02	247.04	254.60	7.56	0.00	NE	0	0
03-P02	254.60	255.30	0.70	0.00	20562	117	11
03-P02	255.30	275.60	20.30	0.00	NE	0	0
03-P02	275.60	276.67	1.07	0.00	20563	43	48
03-P02	276.67	277.86	1.19	0.00	20564	97	34
03-P02	277.86	341.07	63.22	0.00	NE	0	0
03-P02	341.07	341.86	0.79	0.03	20565	1445	85

196
197

03-P02	341.86	376.61	34.75	0.00	NE	0	0
03-P02	376.61	378.04	1.43	0.00	20566	15	7
03-P02	378.04	412.18	34.14	0.00	NE	0	0
03-P02	412.18	412.97	0.79	0.03	20567	13	5
03-P02	412.97	426.72	13.75	0.00	NE	0	0
03-P03	0.00	22.83	22.83	0.00	NE	0	0
03-P03	22.83	24.08	1.25	0.07	20568	1500	28
03-P03	24.08	25.18	1.10	38.81	20569	31000	68
03-P03	25.18	26.06	0.88	45.19	20570	34900	82
03-P03	26.06	27.13	1.07	21.50	20571	40200	68
03-P03	27.13	28.10	0.98	13.89	20572	18900	78
03-P03	28.10	29.32	1.22	0.21	20573	884	26
03-P03	29.32	30.18	0.85	0.72	20574	1870	35
03-P03	30.18	30.78	0.61	5.42	20575	7590	61
03-P03	30.78	31.39	0.61	8.67	20576	5460	94
03-P03	31.39	32.19	0.79	0.07	20577	270	23
03-P03	32.19	33.13	0.94	0.03	20578	208	18
03-P03	33.13	106.13	73.00	0.00	NE	0	0
03-P03	106.13	107.90	1.77	0.41	20579	1405	27
03-P03	107.90	112.59	4.69	0.00	NE	0	0
03-P03	112.59	113.72	1.13	0.00	20580	96	11
03-P03	113.72	114.45	0.73	0.00	20581	53	7
03-P03	114.45	115.89	1.43	0.86	20582	2990	46
03-P03	115.89	117.32	1.43	0.48	20583	1005	64
03-P03	117.32	184.92	67.60	0.00	NE	0	0
03-P03	184.92	185.93	1.01	0.00	20584	35	15
03-P03	185.93	187.15	1.22	0.00	20585	24	15
03-P03	187.15	188.03	0.88	0.00	20586	73	21
03-P03	188.03	204.98	16.95	0.00		0	0
03-P03	204.98	206.38	1.40	0.00	20587	117	17
03-P03	206.38	273.19	66.81	0.00	NE	0	0
03-P03	273.19	274.44	1.25	0.00	20588	245	53
03-P03	274.44	275.42	0.98	0.65	20589	1125	41
03-P03	275.42	300.23	24.81	0.00	NE	0	0
03-P03	300.23	300.81	0.58	0.00	20590	14	5
03-P03	300.81	324.19	23.38	0.00	NE	0	0
03-P03	324.19	325.34	1.16	0.00	20591	14	5
03-P03	325.34	363.96	38.62	0.00	NE	0	0
03-P03	363.96	364.88	0.91	0.00	20592	35	11
03-P03	364.88	365.76	0.88	0.00	20593	14	9
03-P03	365.76	370.06	4.30	0.00	NE	0	0
03-P03	370.06	371.25	1.19	0.00	20594	100	21
03-P03	371.25	372.04	0.79	0.93	20595	2550	70
03-P03	372.04	373.56	1.52	0.00	20596	68	28
03-P03	373.56	374.45	0.88	0.00	20597	97	17
03-P03	374.45	375.67	1.22	0.00	20598	33	20
03-P03	375.67	402.52	26.85	0.00	NE	0	0

03-P03	402.52	403.46	0.94	0.00	20599	11	10
03-P03	403.46	404.35	0.88	0.17	20600	317	21
03-P03	404.35	405.14	0.79	0.03	20601	375	22
03-P03	405.14	406.15	1.01	0.00	20602	122	23
03-P03	406.15	407.37	1.22	1.13	20603	3610	50
03-P03	407.37	408.28	0.91	0.00	20604	56	10
03-P03	408.28	428.21	19.93	0.00	NE	0	0
03-P03	428.21	429.19	0.98	0.00	20605	77	13
03-P03	429.19	430.19	1.01	0.00	20606	194	19
03-P03	430.19	430.80	0.61	0.03	20607	210	18
03-P03	430.80	431.72	0.91	0.00	20608	43	18
03-P03	431.72	443.48	11.77	0.00	NE	0	0
03-P03	443.48	444.52	1.04	0.00	20609	108	33
03-P03	444.52	461.65	17.13	0.00	NE	0	0
03-P03	461.65	462.26	0.61	0.00	20610	26	18
03-P03	462.26	462.90	0.64	3.91	20611	12600	131
03-P03	462.90	463.91	1.01	0.03	20612	80	21
03-P03	463.91	465.03	1.13	0.00	20613	97	45
03-P03	465.03	465.80	0.76	0.48	20614	2480	73
03-P03	465.80	466.92	1.13	7.06	20615	16500	181
03-P03	466.92	467.41	0.49	7.37	20616	18000	1075
03-P03	467.41	468.02	0.61	0.17	20617	3250	81
03-P03	468.02	468.93	0.91	0.03	20618	158	21
03-P03	468.93	469.67	0.73	0.03	20619	145	37
03-P03	469.67	470.09	0.43	0.10	20620	180	24
03-P03	470.09	471.40	1.31	0.03	20621	198	30
03-P03	471.40	472.29	0.88	0.00	20622	69	21
03-P03	472.29	473.81	1.52	0.27	20623	865	26
03-P03	473.81	475.09	1.28	0.24	20624	387	20
03-P03	475.09	523.65	48.55	0.00	NE	0	0
03-P03	523.65	524.26	0.61	0.00	20625	394	15
03-P03	524.26	524.71	0.46	8.30	20626	12800	50
03-P03	524.71	525.17	0.46	0.10	20627	1455	31
03-P03	525.17	525.63	0.46	10.18	20628	11300	52
03-P03	525.63	526.24	0.61	0.00	20629	114	20
03-P03	526.24	535.90	9.66	0.00	NE	0	0
03-P03	535.90	536.94	1.04	0.00	20630	128	17
03-P03	536.94	537.97	1.04	1.30	20631	6680	87
03-P03	537.97	539.01	1.04	0.03	20632	267	33
03-P03	539.01	539.89	0.88	3.87	20633	11100	103
03-P03	539.89	540.81	0.91	0.00	20634	87	17
03-P03	540.81	547.30	6.49	0.00	NE	0	0
03-P03	547.30	548.12	0.82	0.00	20635	100	14
03-P03	548.12	552.66	4.54	0.00	NE	0	0
03-P03	552.66	553.27	0.61	1.10	20636	2350	28
03-P03	553.27	554.40	1.13	0.07	20637	859	51
03-P03	554.40	555.59	1.19	0.03	20638	396	127

198
199

03-P03	555.59	565.43	9.85	0.00	NE	0	0
03-P03	565.43	566.04	0.61	7.47	20639	6580	2800
03-P03	566.04	573.94	7.89	0.00	NE	0	0
03-P03	573.94	574.97	1.04	0.00	20640	128	95
03-P03	574.97	575.80	0.82	1.61	20641	1650	10000
03-P03	575.80	577.08	1.28	5.59	20642	9520	1500
03-P03	577.08	578.60	1.52	0.93	20643	954	251
03-P03	578.60	579.64	1.04	0.82	20644	1315	400
03-P03	579.64	580.37	0.73	2.64	20645	3580	58
03-P03	580.37	606.77	26.40	0.00	NE	0	0
03-P03	606.77	607.37	0.61	3.50	20646	2880	53
03-P03	607.37	616.46	9.08	0.00	NE	0	0
03-P03	616.46	617.25	0.79	4.18	20647	3490	59
03-P03	617.25	625.63	8.38	0.00	NE	0	0
03-P03	625.63	626.55	0.91	0.07	20648	226	27
03-P03	626.55	670.44	43.89	0.00	NE	0	0
03-P03	670.44	671.41	0.98	0.03	20649	263	59
03-P03	671.41	672.11	0.70	4.56	20650	9750	106
03-P03	672.11	672.79	0.67	0.10	20651	350	56
03-P03	672.79	673.64	0.85	0.00	20652	84	40
03-P03	673.64	694.12	20.48	0.00	NE	0	0
03-P03	694.12	694.85	0.73	0.65	20653	805	51
03-P03	694.85	700.83	5.97	0.00	NE	0	0
03-P03	700.83	701.44	0.61	0.65	20654	1200	27
03-P03	701.44	726.73	25.30	0.00	NE	0	0
03-P03	726.73	727.89	1.16	0.00	20655	38	29
03-P03	727.89	729.02	1.13	0.00	20656	90	83
03-P03	729.02	729.94	0.91	2.85	20657	6180	1320
03-P03	729.94	730.79	0.85	1.68	20658	3500	120
03-P03	730.79	731.46	0.67	0.03	20659	216	72
03-P03	731.46	732.28	0.82	0.00	20660	47	49
03-P03	732.28	732.92	0.64	0.00	20661	31	32
03-P03	732.92	733.99	1.07	0.00	20662	97	41
03-P03	733.99	735.12	1.13	0.00	20663	288	53
03-P03	735.12	736.06	0.94	0.00	20664	139	40
03-P03	736.06	740.54	4.48	0.00	NE	0	0
03-P03	740.54	741.27	0.73	0.14	20665	360	44
03-P03	741.27	742.49	1.22	0.00	20666	279	51
03-P03	742.49	743.01	0.52	0.00	20667	137	81
03-P03	743.01	743.93	0.91	0.00	20668	126	37
03-P03	743.93	744.72	0.79	0.00	20669	76	44
03-P03	744.72	762.27	17.56	0.00	NE	0	0
03-P03	762.27	762.61	0.34	0.00	20670	382	76
03-P03	762.61	768.71	6.10	0.00	NE	0	0
04-1707E1	192.69	193.70	1.01	0.14	20988	0	0
04-1707E1	193.70	194.22	0.52	0.00	20989	0	0
04-1707E1	194.22	194.98	0.76	0.07	20990	0	0

109
200

04-1707E1	194.98	195.96	0.98	0.07	20991	0	0
04-1707E1	195.96	196.29	0.34	0.00	20992	0	0
04-1707E1	196.29	196.87	0.58	0.03	20993	0	0
04-1707E1	196.87	197.02	0.15	0.00	20994	0	0
04-1707E1	197.02	197.51	0.49	0.10	20995	0	0
04-1707E1	197.51	198.85	1.34	0.03	20996	0	0
04-1707E1	198.85	199.19	0.34	0.10	20997	0	0
04-1707E1	199.19	199.52	0.34	0.10	20998	0	0
04-1707E1	199.52	200.41	0.88	0.99	20999	0.13	0.04
04-1707E1	200.41	201.93	1.52	0.21	21000	0	0
04-1707E1	201.93	202.14	0.21	6.21	109001	0.04	0.19
04-1707E1	202.14	202.30	0.15	0.55	109002	0	0
04-1707E1	202.30	203.03	0.73	13.47	109003	0.04	1.95
04-1707E1	203.03	203.91	0.88	0.96	109004	0	0
04-1707E1	203.91	205.44	1.52	0.31	109005	0	0
04-1707E1	205.44	205.98	0.55	0.10	109006	0	0
04-1707E1	205.98	207.02	1.04	0.21	109007	0	0
04-1707E1	207.02	208.15	1.13	0.03	109008	0	0
04-1707E1	208.15	209.46	1.31	0.03	109009	0	0
04-1707E1	209.46	210.92	1.46	3.60	109010	0.01	0.44
04-1707E1	210.92	211.90	0.98	7.95	109011	0.11	0.45
04-1707E1	211.90	213.42	1.52	4.66	109012	0.03	1.14
04-1707E1	213.42	214.91	1.49	0.58	109013	0	0
04-1707E1	214.91	216.68	1.77	0.17	109014	0	0
04-1707E1	216.68	217.90	1.22	0.00	109015	0	0
04-1707E1	217.90	219.43	1.52	0.00	109016	0	0
04-1707E1	219.43	220.55	1.13	0.00	109017	0	0
04-1707E1	220.55	221.68	1.13	0.03	109018	0	0
04-1707E1	221.68	222.05	0.37	0.07	109019	0	0
04-1707E1	222.05	222.35	0.30	0.03	109020	0	0
04-1707E1	222.35	223.72	1.37	0.00	109021	0	0
04-1707E2	212.45	213.97	1.52	0.00	109023	0	0
04-1707E2	213.97	215.40	1.43	0.00	109024	0	0
04-1707E2	215.40	216.99	1.58	0.00	109025	0	0
04-1707E2	216.99	218.51	1.52	0.03	109026	0	0
04-1707E2	218.51	220.13	1.62	0.21	109027	0	0
04-1707E2	220.13	220.98	0.85	0.17	109028	0	0
04-1707E2	220.98	221.59	0.61	0.14	109029	0	0
04-1707E2	221.59	222.08	0.49	0.31	109030	0	0
04-1707E2	222.08	222.20	0.12	18.62	109031	1.31	0.03
04-1707E2	222.20	223.36	1.16	12.51	109032	1.14	0.05
04-1707E2	223.36	224.27	0.91	4.49	109033	0.95	0.01
04-1707E2	224.27	224.73	0.46	12.75	109034	1.75	0.01
04-1707E2	224.73	226.04	1.31	0.93	109035	0	0
04-1707E2	226.04	226.71	0.67	0.14	109036	0	0
04-1707E2	226.71	227.69	0.98	0.17	109037	0	0
04-1707E2	227.69	228.57	0.88	2.85	109038	0.91	0.01

200
201

04-1707E2	228.57	229.39	0.82	14.57	109039	1.16	0.27
04-1707E2	229.39	230.70	1.31	0.10	109040	0	0
04-1707E2	230.70	232.23	1.52	0.07	109041	0	0
04-1707E2	232.23	233.48	1.25	0.03	109042	0	0
04-1707E2	233.48	234.54	1.07	0.03	109043	0	0
04-1707E2	234.54	235.15	0.61	0.03	109044	0	0
04-1707E2	235.15	235.61	0.46	0.03	109045	0	0
04-1707E2	235.61	236.59	0.98	0.10	109046	0	0
04-1707E2	236.59	237.59	1.01	38.64	109047	3.03	0.01
04-1707E2	237.59	239.24	1.65	0.89	109048	0	0
04-1707E2	239.24	239.85	0.61	0.14	109049	0	0
04-1707E2	239.85	240.49	0.64	0.07	109050	0	0
04-1707E2	240.49	241.10	0.61	0.07	109051	0	0
04-1707E2	241.10	243.84	2.74	0.07	109052	0	0
04-1707E2	243.84	245.36	1.52	0.07	109053	0	0
04-1707E2	245.36	246.64	1.28	0.99	109054	0	0
04-1707E2	246.64	247.16	0.52	8.16	109055	0.06	0.01
04-1707E2	247.16	247.77	0.61	0.27	109056	0	0
04-1707E2	247.77	248.35	0.58	25.82	109057	0.1	0.01
04-1707E2	248.35	248.72	0.37	13.95	109058	0.05	0.01
04-1707E2	248.72	250.24	1.52	0.41	109059	0	0
04-1707E2	250.24	251.46	1.22	0.03	109060	0	0
04-1707E2	251.46	251.92	0.46	0.10	109061	0	0
04-1707E2	251.92	252.50	0.58	0.10	109062	0	0
04-1707E2	252.50	253.84	1.34	0.17	109063	0	0
04-1707E2	253.84	254.11	0.27	23.52	109064	0.71	1.21
04-1707E2	254.11	255.64	1.52	1.13	109065	0.14	0.59
04-1707E2	255.64	256.76	1.13	0.10	109066	0	0
04-1707E2	256.76	258.01	1.25	0.03	109067	0	0
04-1707E2	258.01	258.62	0.61	0.03	109068	0	0
04-1707E2	258.62	260.09	1.46	0.03	109069	0	0
04-1707E2	260.09	260.91	0.82	0.00	109070	0	0
04-1707E2	260.91	262.43	1.52	0.00	109071	0	0
04-1707E2	262.43	263.96	1.52	0.00	109072	0	0
04-1707E2	263.96	264.87	0.91	0.00	109073	0	0
04-1737E1	66.75	68.28	1.52	0.00	20848	0	0
04-1737E1	68.28	69.89	1.62	0.00	20849	0	0
04-1737E1	69.89	71.60	1.71	0.00	20850	0	0
04-1737E1	71.60	72.97	1.37	0.00	20851	0	0
04-1737E1	72.97	74.52	1.55	0.00	20852	0	0
04-1737E1	74.52	75.93	1.40	0.00	20853	0	0
04-1737E1	75.93	77.42	1.49	0.34	20854	0	0
04-1737E1	107.90	109.42	1.52	0.03	20856	0	0
04-1737E1	109.42	110.95	1.52	0.00	20857	0	0
04-1737E1	110.95	112.47	1.52	0.00	20858	0	0
04-1737E1	112.47	149.50	1.52	0.00	20859	0	0
04-1737E1	149.50	150.88	1.37	0.00	20860	0	0

201
202

04-1737E1	150.88	151.94	1.07	0.03	20861	0	0
04-1737E1	151.94	153.01	1.07	0.00	20862	0	0
04-1737E1	178.77	179.83	1.07	0.00	20864	0	0
04-1737E1	179.83	181.20	1.37	0.00	20865	0	0
04-1737E1	181.20	181.97	0.76	0.00	20866	0	0
04-1737E1	181.97	183.03	1.07	0.03	20867	0	0
04-1737E1	183.03	184.25	1.22	20.43	20868	1.83	0.07
04-1737E1	184.25	185.32	1.07	30.65	20869	2.99	0.02
04-1737E1	185.32	186.54	1.22	0.07	20870	0	0
04-1737E1	186.54	187.45	0.91	0.07	20871	0	0
04-1737E1	187.45	188.98	1.52	18.38	20873	1.24	0.01
04-1737E1	188.98	190.41	1.43	24.51	20874	1.71	0.22
04-1737E1	190.41	190.80	0.40	29.14	20875	1.43	0.24
04-1737E1	190.80	192.02	1.22	4.46	20876	0.75	0.01
04-1737E1	192.02	193.40	1.37	0.07	20877	0	0
04-1737E1	193.40	194.46	1.07	0.07	20878	0	0
04-1737E1	194.46	195.38	0.91	0.03	20879	0	0
04-1737E1	195.38	196.60	1.22	0.03	20880	0	0
04-1737E1	196.60	198.27	1.68	0.00	20881	0	0
04-1737E1	198.27	199.34	1.07	0.00	20882	0	0
04-1737E1	199.34	200.25	0.91	2.54	20883	0.4	0.01
04-1737E1	200.25	201.47	1.22	12.48	20884	1.68	0.01
04-1737E1	201.47	202.69	1.22	2.88	20885	1.24	0.01
04-1737E1	202.69	204.06	1.37	1.06	20886	0.47	0.01
04-1737E1	204.06	206.04	1.98	1.20	20887	0.32	0.01
04-1737E1	206.04	207.05	1.01	4.53	20888	0.23	0.01
04-1737E1	207.05	207.57	0.52	4.01	20889	0.01	0.01
04-1737E1	207.57	208.18	0.61	0.03	20891	0	0
04-1737E1	208.18	209.70	1.52	1.99	20892	0.19	0.01
04-1737E1	209.70	210.92	1.22	0.03	20893	0	0
04-1737E1	210.92	212.66	1.74	0.03	20894	0	0
04-1737E1	212.66	213.76	1.10	0.14	20895	0	0
04-1737E1	213.76	215.59	1.83	0.07	20896	0	0
04-1737E1	215.59	216.77	1.19	0.17	20897	0	0
04-1737E1	216.77	218.24	1.46	0.03	20898	0	0
04-1737E1	218.24	219.64	1.40	0.03	20899	0	0
04-1737E1	219.64	219.91	0.27	0.14	20900	0	0
04-1737E1	219.91	221.13	1.22	0.03	20901	0	0
04-1737E1	221.13	222.87	1.74	0.03	20902	0	0
04-1737E1	222.87	223.88	1.01	0.03	20903	0	0
04-1737E1	223.88	224.64	0.76	0.07	20904	0	0
04-1737E1	224.64	225.55	0.91	0.07	20905	0	0
04-1737E1	225.55	227.08	1.52	0.10	20906	0	0
04-1737E1	227.08	227.53	0.46	0.21	20907	0	0
04-1737E1	227.53	228.30	0.76	0.38	20908	0	0
04-1737E1	228.30	229.45	1.16	0.21	20909	0	0
04-1737E1	229.45	231.19	1.74	0.10	20910	0	0

202
203

04-1737E1	231.19	232.26	1.07	3.84	20911	0.57	0
04-1737E1	232.26	233.78	1.52	28.83	20912	4.7	0.01
04-1737E1	233.78	234.45	0.67	10.25	20913	3.1	0
04-1737E1	234.45	235.37	0.91	1.95	20914	1.41	0
04-1737E1	235.37	236.98	1.62	0.58	20915	0	0
04-1737E1	236.98	238.35	1.37	0.99	20916	0.27	0
04-1737E1	238.35	239.54	1.19	0.03	20917	0	0
04-1737E1	239.54	241.10	1.55	0.03	20918	0	0
04-1737E1	241.10	242.62	1.52	0.03	20919	0	0
04-1737E1	242.62	243.23	0.61	0.07	20920	0	0
04-1737E1	260.60	261.52	0.91	0.24	20922	0	0
04-1737E1	261.52	263.04	1.52	0.10	20923	0	0
04-1737E1	263.04	263.65	0.61	0.07	20924	0	0
04-1737E1	263.65	264.20	0.55	0.07	20925	0	0
04-1737E1	264.20	264.69	0.49	0.00	20926	0	0
04-1737E1	264.69	266.21	1.52	0.03	20927	0	0
04-1737E1	266.21	266.85	0.64	0.07	20928	0	0
04-1737E1	266.85	267.13	0.27	1.51	20929	0.83	0.01
04-1737E1	267.13	267.89	0.76	1.17	20930	0.55	0.01
04-1737E1	267.89	268.53	0.64	0.03	20931	0	0
04-1737E1	268.53	269.44	0.91	0.03	20932	0	0
04-1737E1	269.44	270.81	1.37	0.00	20933	0	0
04-1737E1	270.81	272.25	1.43	0.00	20934	0	0
04-1737E1	272.25	272.74	0.49	0.00	20935	0	0
04-1737E1	272.74	273.50	0.76	0.00	20936	0	0
04-1737E1	273.50	273.68	0.18	0.00	20937	0	0
04-1737E1	273.68	274.69	1.01	0.17	20938	0	0
04-1737E1	274.69	275.05	0.37	1.54	20939	0.5	0.01
04-1737E1	275.05	276.27	1.22	0.00	20940	0	0
04-1737E1	276.27	277.83	1.55	0.00	20941	0	0
04-1737E1	277.83	278.28	0.46	0.00	20942	0	0
04-1737E1	278.28	279.20	0.91	0.00	20943	0	0
04-1737E1	279.20	279.50	0.30	0.00	20944	0	0
04-1737E1	279.50	280.87	1.37	0.00	20945	0	0
04-1737E1	280.87	282.15	1.28	0.00	20946	0	0
04-1737E1	282.15	283.46	1.31	0.00	20947	0	0
04-1737E1	283.46	284.99	1.52	0.17	20948	0	0
04-1737E1	284.99	286.51	1.52	0.03	20949	0	0
04-1737E1	286.51	287.43	0.91	0.00	20950	0	0
04-1737E2	222.99	223.42	0.43	0.07	20952	0	0
04-1737E2	223.42	224.82	1.40	2.98	20953	0.4	0.02
04-1737E2	224.82	226.34	1.52	0.03	20954	0	0
04-1737E2	226.34	227.93	1.58	0.03	20955	0	0
04-1737E2	227.93	229.45	1.52	0.07	20956	0	0
04-1737E2	229.45	230.58	1.13	0.03	20957	0	0
04-1737E2	230.58	231.34	0.76	0.00	20958	0	0
04-1737E2	231.34	232.87	1.52	0.00	20959	0	0

203
204

04-1737E2	232.87	233.17	0.30	1.54	20960	0.42	0
04-1737E2	233.17	233.66	0.49	0.03	20961	0	0
04-1737E2	233.66	233.96	0.30	30.93	20962	2.65	0.01
04-1737E2	233.96	235.49	1.52	0.51	20963	0	0
04-1737E2	235.49	237.01	1.52	0.48	20964	0	0
04-1737E2	237.01	238.54	1.52	0.14	20965	0	0
04-1737E2	238.54	240.43	1.89	0.00	20966	0	0
04-1737E2	240.43	240.73	0.30	0.03	20967	0	0
04-1737E2	240.73	241.58	0.85	0.00	20968	0	0
04-1737E2	241.58	243.11	1.52	0.03	20969	0	0
04-1737E2	243.11	244.54	1.43	0.03	20970	0	0
04-1737E2	244.54	246.07	1.52	0.00	20971	0	0
04-1737E2	246.07	247.28	1.22	0.00	20972	0	0
04-1737E2	247.28	248.81	1.52	0.00	20973	0	0
04-1737E2	248.81	250.27	1.46	0.00	20974	0	0
04-1737E2	250.27	251.76	1.49	0.00	20975	0	0
04-1737E2	251.76	253.14	1.37	0.27	20976	0	0
04-1737E2	253.14	254.42	1.28	0.00	20977	0	0
04-1737E2	254.42	255.27	0.85	0.00	20978	0	0
04-1737E2	255.27	255.88	0.61	0.00	20979	0	0
04-1737E2	255.88	256.64	0.76	0.03	20980	0	0
04-1737E2	256.64	256.92	0.27	8.91	20981	0.49	0
04-1737E2	256.92	257.83	0.91	2.23	20982	0.97	0.01
04-1737E2	257.83	259.08	1.25	14.37	20983	0.96	0.01
04-1737E2	259.08	260.60	1.52	0.03	20984	0	0
04-1737E2	260.60	262.13	1.52	0.03	20985	0	0
04-1737E2	262.13	263.65	1.52	0.00	20986	0	0
04-300SW1	178.64	179.62	0.98	0.89	109177	0	0
04-300SW1	179.62	180.75	1.13	1.23	109178	0.05	0.06
04-300SW1	180.75	181.72	0.98	11.21	109179	1.18	0.05
04-300SW1	181.72	183.18	1.46	0.03	109180	0	0
04-300SW2	80.35	80.62	0.27	0.24	109074	0	0
04-300SW2	80.62	82.30	1.68	0.03	109075	0	0
04-300SW2	82.30	83.21	0.91	0.69	109076	0	0
04-300SW2	107.72	108.72	1.01	0.45	109077	0	0
04-300SW2	108.72	110.28	1.55	0.00	109078	0	0
04-300SW2	110.28	111.40	1.13	3.98	109079	2.1	0.01
04-300SW2	111.40	112.26	0.85	4.56	109080	2.61	0.01
04-300SW2	112.26	113.39	1.13	0.00	109081	0	0
04-300SW2	113.39	114.00	0.61	0.03	109082	0	0
04-300SW2	180.20	181.20	1.01	0.03	109083	0	0
04-300SW2	181.20	181.66	0.46	5.49	109084	1.09	0.07
04-300SW2	181.66	182.58	0.91	0.21	109085	0	0
04-300SW2	182.58	183.49	0.91	0.03	109086	0	0
04-300SW2	183.49	185.01	1.52	0.00	109087	0	0
04-300SW2	185.01	186.48	1.46	0.00	109088	0	0
04-300SW2	186.48	187.70	1.22	0.00	109089	0	0

204
205

04-300SW2	187.70	188.88	1.19	0.00	109090	0	0
04-300SW2	188.88	190.41	1.52	0.00	109091	0	0
04-300SW2	190.41	190.90	0.49	0.00	109092	0	0
04-300SW2	190.90	191.17	0.27	0.03	109093	0	0
04-300SW2	191.17	192.91	1.74	0.03	109094	0	0
04-300SW2	192.91	193.09	0.18	0.03	109095	0	0
04-300SW2	193.09	194.61	1.52	0.03	109096	0	0
04-300SW2	194.61	195.16	0.55	0.10	109097	0	0
04-300SW2	195.16	196.75	1.58	0.14	109098	0	0
04-300SW2	196.75	197.66	0.91	0.03	109099	0	0
04-300SW2	197.66	199.80	2.13	0.03	109100	0	0
04-300SW2	199.80	201.02	1.22	0.31	109101	0	0
04-300SW2	201.02	202.54	1.52	0.00	109102	0	0
04-300SW2	202.54	203.91	1.37	0.00	109103	0	0
04-300SW2	203.91	205.44	1.52	0.00	109104	0	0
04-300SW2	205.44	206.53	1.10	0.10	109105	0	0
04-300SW2	206.53	206.96	0.43	10.11	109106	1.3	0.02
04-300SW2	206.96	207.84	0.88	0.03	109107	0	0
04-300SW2	207.84	208.94	1.10	28.63	109108	2.01	0.03
04-300SW2	208.94	210.59	1.65	0.55	109109	0	0
04-300SW2	210.59	212.11	1.52	0.03	109110	0	0
04-300SW3	77.63	78.94	1.31	0.00	109113	0	0
04-300SW3	78.94	80.47	1.52	0.00	109114	0	0
04-300SW3	80.47	80.92	0.46	0.03	109115	0	0
04-300SW3	80.92	81.53	0.61	0.03	109116	0	0
04-300SW3	81.53	83.06	1.52	0.00	109117	0	0
04-300SW3	83.06	84.43	1.37	0.00	109118	0	0
04-300SW3	122.99	124.21	1.22	0.14	109119	0	0
04-300SW3	124.21	125.64	1.43	0.03	109120	0	0
04-300SW3	125.64	126.25	0.61	2.13	109121	1.19	0.01
04-300SW3	131.52	133.05	1.52	1.37	109122	0.83	0.01
04-300SW3	133.05	134.02	0.98	0.00	109123	0	0
04-300SW3	134.02	135.82	1.80	0.00	109124	0	0
04-300SW3	135.82	136.37	0.55	0.17	109125	0	0
04-300SW3	136.37	137.65	1.28	10.66	109126	1.99	0.01
04-300SW3	137.65	138.87	1.22	0.41	109127	0	0
04-300SW3	138.87	140.24	1.37	0.03	109128	0	0
04-300SW3	140.24	140.60	0.37	0.00	109129	0	0
04-300SW3	140.60	142.16	1.55	0.00	109130	0	0
04-300SW3	142.16	143.80	1.65	0.00	109131	0	0
04-300SW3	143.80	144.29	0.49	0.03	109132	0	0
04-300SW3	144.29	145.08	0.79	0.03	109133	0	0
04-300SW3	145.08	145.39	0.30	0.34	109134	0	0
04-300SW3	145.39	146.00	0.61	0.00	109135	0	0
04-300SW3	194.71	196.38	1.68	0.10	109136	0	0
04-300SW3	196.38	196.87	0.49	13.27	109137	1.3	0.03
04-300SW3	196.87	197.48	0.61	1.75	109138	0.58	0.02

205
206

04-300SW3	197.48	199.03	1.55	10.32	109139	2.67	0.02
04-300SW3	199.03	199.80	0.76	0.10	109140	0	0
04-300SW3	199.80	201.11	1.31	2.13	109141	1.13	0.01
04-300SW3	201.11	201.63	0.52	2.88	109142	1.2	0.01
04-300SW3	201.63	202.57	0.94	7.30	109143	2.42	0.02
04-300SW3	202.57	203.61	1.04	16.97	109144	3.41	0.02
04-300SW3	203.61	204.58	0.98	0.07	109145	0	0
04-300SW3	204.58	205.44	0.85	0.55	109146	0	0
04-300SW3	205.44	206.75	1.31	0.00	109147	0	0
04-300SW3	206.75	208.18	1.43	0.00	109148	0	0
04-300SW3	208.18	209.55	1.37	0.00	109149	0	0
04-300SW3	209.55	211.23	1.68	0.00	109150	0	0
04-300SW3	211.23	212.54	1.31	0.00	109151	0	0
04-300SW3	212.54	213.36	0.82	0.00	109152	0	0
04-300SW3	213.36	214.73	1.37	0.00	109153	0	0
04-300SW3	214.73	215.31	0.58	0.00	109154	0	0
04-300SW3	215.31	216.71	1.40	0.00	109155	0	0
04-300SW3	216.71	217.93	1.22	0.03	109156	0	0
04-300SW3	217.93	219.52	1.58	0.03	109157	0	0
04-300SW3	219.52	221.19	1.68	0.00	109158	0	0
04-300SW3	221.19	222.81	1.62	0.03	109159	0	0
04-300SW3	222.81	223.42	0.61	0.31	109160	0	0
04-300SW3	223.42	223.88	0.46	0.34	109161	0	0
04-300SW3	223.88	224.58	0.70	4.83	109162	1.85	0.02
04-300SW3	224.58	225.86	1.28	0.00	109163	0	0
04-300SW3	225.86	227.26	1.40	22.42	109164	3.27	0.13
04-300SW3	227.26	227.96	0.70	1.99	109165	0.84	0.03
04-300SW3	227.96	228.84	0.88	22.22	109166	1.95	1.36
04-300SW3	228.84	230.00	1.16	34.15	109167	2.98	0.02
04-300SW3	230.00	230.73	0.73	5.83	109168	1.17	0.01
04-300SW3	230.73	232.26	1.52	0.10	109169	0	0
04-300SW3	232.26	233.84	1.58	0.00	109170	0	0
04-300SW3	233.84	235.00	1.16	0.00	109171	0	0
04-300SW3	235.00	236.22	1.22	0.00	109172	0	0
04-300SW3	236.22	236.71	0.49	0.14	109173	0	0
04-300SW3	236.71	237.44	0.73	0.00	109174	0	0
04-300SW3	237.44	238.96	1.52	0.00	109175	0	0
04-330SW1	91.74	93.27	1.52	0.00	20701	0	0
04-330SW1	93.27	94.79	1.52	0.00	20702	0	0
04-330SW1	94.79	96.32	1.52	0.00	20703	0	0
04-330SW1	163.07	164.21	1.14	0.00	20704	0	0
04-330SW1	164.21	165.35	1.14	0.07	20705	0	0
04-330SW1	165.35	165.89	0.53	0.03	20706	0	0
04-330SW1	165.89	166.60	0.72	0.03	20707	0	0
04-330SW1	166.60	166.73	0.12	0.07	20708	0	0
04-330SW1	166.73	167.18	0.46	0.00	20709	0	0
04-330SW1	167.18	168.25	1.07	1.06	20710	0.23	0.01

206
207

04-330SW1	168.25	168.78	0.53	10.01	20711	1.05	0.02
04-330SW1	168.78	170.54	1.75	0.93	20712	0	0
04-330SW1	170.54	172.21	1.68	0.38	20713	0	0
04-330SW1	172.21	172.36	0.15	1.27	20714	0.06	0.05
04-330SW1	172.36	172.82	0.46	30.03	20715	2.73	0.83
04-330SW1	172.82	173.74	0.91	35.45	20716	3.14	1.28
04-330SW1	173.74	174.19	0.46	9.94	20717	2.75	0.33
04-330SW1	174.19	174.50	0.30	80.61	20718	4.41	2.23
04-330SW1	174.50	175.11	0.61	2.13	20719	1.03	0.02
04-330SW1	175.11	175.56	0.46	0.14	20720	0	0
04-330SW1	175.56	176.02	0.46	0.03	20721	0	0
04-330SW1	176.02	177.09	1.07	0.10	20722	0	0
04-330SW2	167.34	168.86	1.52	0.00	20723	0	0
04-330SW2	168.86	170.93	2.07	2.85	20724	0.27	0.01
04-330SW2	170.93	171.60	0.67	13.68	20725	2.15	0.01
04-330SW2	171.60	173.13	1.52	0.03	20726	0	0
04-330SW2	173.13	173.81	0.69	0.00	20727	0	0
04-330SW2	173.81	175.26	1.45	0.00	20728	0	0
04-330SW2	175.26	176.78	1.52	0.00	20729	0	0
04-330SW2	176.78	178.31	1.52	0.00	20730	0	0
04-330SW2	178.31	179.83	1.52	0.00	20731	0	0
04-330SW2	179.83	181.36	1.52	0.00	20732	0	0
04-330SW2	181.36	181.84	0.49	0.00	20733	0	0
04-330SW2	181.84	183.42	1.57	0.00	20734	0	0
04-330SW2	183.42	183.79	0.38	0.03	20735	0	0
04-330SW2	183.79	184.56	0.76	0.03	20736	0	0
04-330SW2	184.56	186.08	1.52	0.24	20737	0	0
04-330SW2	186.08	187.18	1.10	0.00	20738	0	0
04-330SW2	187.18	187.67	0.49	0.38	20739	0	0
04-330SW2	187.67	188.98	1.31	0.27	20740	0	0
04-330SW2	188.98	190.58	1.60	0.10	20741	0	0
04-330SW2	190.58	190.80	0.23	0.03	20742	0	0
04-330SW2	190.80	191.96	1.16	0.17	20743	0	0
04-330SW2	191.96	192.57	0.61	2.33	20744	1.83	0.26
04-330SW2	192.57	193.09	0.52	0.65	20746	0	0
04-330SW2	193.09	194.22	1.13	0.24	20747	0	0
04-330SW2	194.22	195.56	1.34	16.22	20748	1.89	0.08
04-330SW2	195.56	197.08	1.52	44.37	20749	2.39	1.48
04-330SW2	197.08	198.42	1.34	39.74	20750	2.26	0.38
04-330SW2	198.42	199.40	0.98	0.55	20751	0	0
04-330SW2	199.40	200.62	1.22	45.67	20752	1.49	0.15
04-330SW2	200.62	202.27	1.65	35.35	20753	2.45	0.01
04-330SW2	202.27	203.30	1.04	0.17	20754	0	0
04-330SW2	203.30	203.76	0.46	1.61	20755	0.71	0.01
04-330SW2	203.76	205.59	1.83	0.03	20756	0	0
04-330SW2	205.59	206.35	0.76	0.03	20757	0	0
04-330SW2	206.35	207.39	1.04	0.00	20758	0	0

²⁰⁷
208

04-360SW1	157.95	159.47	1.52	0.00	20797	0	0
04-360SW1	159.47	161.00	1.52	0.00	20798	0	0
04-360SW1	161.00	162.21	1.22	0.00	20799	0	0
04-360SW1	162.21	162.76	0.55	0.03	20800	0	0
04-360SW1	162.76	163.68	0.91	5.18	20801	0.07	0.01
04-360SW1	163.68	164.56	0.88	0.10	20802	0	0
04-360SW1	164.56	165.66	1.10	0.00	20803	0	0
04-360SW1	165.66	167.34	1.68	0.00	20804	0	0
04-360SW1	167.34	168.55	1.22	0.00	20805	0	0
04-360SW1	168.55	170.08	1.52	0.00	20806	0	0
04-360SW1	170.08	170.87	0.79	0.00	20807	0	0
04-360SW1	170.87	171.60	0.73	0.03	20808	0	0
04-360SW1	171.60	173.00	1.40	0.00	20809	0	0
04-360SW1	173.00	174.96	1.95	0.00	20810	0	0
04-360SW1	174.96	175.75	0.79	0.00	20811	0	0
04-360SW1	175.75	177.24	1.49	0.00	20812	0	0
04-360SW1	177.24	177.88	0.64	0.03	20813	0	0
04-360SW1	177.88	179.10	1.22	0.03	20814	0	0
04-360SW1	179.10	180.05	0.94	31.34	20815	2.36	7.99
04-360SW1	180.05	181.57	1.52	4.80	20816	1.17	0.02
04-360SW1	181.57	182.39	0.82	9.29	20817	2.84	0.13
04-360SW1	182.39	183.95	1.55	28.53	20818	2.49	0.46
04-360SW1	183.95	184.47	0.52	13.95	20819	2.26	0.02
04-360SW1	184.47	185.93	1.46	5.11	20820	0.94	0.02
04-360SW1	185.93	187.38	1.45	1.51	20821	0.11	0.01
04-360SW1	187.38	188.90	1.52	1.65	20822	0.09	0.01
04-360SW1	188.90	189.74	0.84	14.23	20823	0.33	0.01
04-360SW1	189.74	190.50	0.76	0.31	20824	0	0
04-360SW1	190.50	191.69	1.19	12.75	20825	1.12	0.11
04-360SW1	191.69	192.69	1.01	18.38	20826	2.61	0.02
04-360SW1	192.69	193.43	0.73	17.62	20827	3.71	0.02
04-360SW1	193.43	194.95	1.52	0.75	20828	0	0
04-360SW1	194.95	195.99	1.04	0.10	20829	0	0
04-360SW1	195.99	197.51	1.52	0.00	20830	0	0
04-360SW1	197.51	198.73	1.22	0.00	20831	0	0
04-360SW1	198.73	199.40	0.67	0.00	20832	0	0
04-360SW1	199.40	200.86	1.46	0.03	20833	0	0
04-360SW1	200.86	201.17	0.30	0.03	20834	0	0
04-360SW1	201.17	202.39	1.22	0.00	20835	0	0
04-360SW1	202.39	202.84	0.46	0.00	20836	0	0
04-360SW1	202.84	204.22	1.37	0.07	20837	0	0
04-360SW1	204.22	205.40	1.19	0.00	20838	0	0
04-360SW1	225.25	227.08	1.83	0.03	20839	0	0
04-360SW1	227.08	228.60	1.52	0.00	20840	0	0
04-360SW1	228.60	230.12	1.52	0.00	20841	0	0
04-360SW1	230.12	231.34	1.22	0.00	20842	0	0
04-360SW1	231.34	232.87	1.52	0.00	20843	0	0

208
209

04-360SW1	232.87	234.18	1.31	0.00	20844	0	0
04-360SW1	234.18	235.70	1.52	0.00	20845	0	0
04-360SW1	235.70	237.44	1.74	0.00	20846	0	0
04-360SW2	137.77	139.29	1.52	0.00	20759	0	0
04-360SW2	139.29	140.54	1.25	0.00	20760	0	0
04-360SW2	140.54	142.04	1.49	0.03	20761	0	0
04-360SW2	142.04	143.87	1.83	0.03	20762	0	0
04-360SW2	143.87	145.39	1.52	0.00	20763	0	0
04-360SW2	145.39	146.67	1.28	0.00	20764	0	0
04-360SW2	146.67	147.83	1.16	0.00	20765	0	0
04-360SW2	147.83	149.35	1.52	0.07	20766	0	0
04-360SW2	157.58	159.04	1.46	0.10	20767	0	0
04-360SW2	159.04	160.48	1.43	0.27	20768	0	0
04-360SW2	160.48	161.39	0.91	0.07	20769	0	0
04-360SW2	161.39	162.76	1.37	0.03	20770	0	0
04-360SW2	185.11	186.75	1.65	0.24	20772	0	0
04-360SW2	186.75	188.15	1.40	0.07	20773	0	0
04-360SW2	188.15	189.68	1.52	0.00	20774	0	0
04-360SW2	189.68	191.20	1.52	1.37	20775	0.13	0.02
04-360SW2	191.20	192.42	1.22	62.26	20776	0.35	0.02
04-360SW2	192.42	193.91	1.49	0.10	20777	0	0
04-360SW2	193.91	195.38	1.46	21.33	20778	0.03	0.01
04-360SW2	195.38	197.24	1.86	0.03	20779	0	0
04-360SW2	197.24	198.24	1.01	0.03	20780	0	0
04-360SW2	198.24	199.52	1.28	2.47	20781	0.08	0.01
04-360SW2	199.52	201.02	1.49	9.50	20782	1.03	0.05
04-360SW2	201.02	201.87	0.85	1.71	20783	0.88	0.12
04-360SW2	201.87	203.39	1.52	0.10	20784	0	0
04-360SW2	203.39	203.52	0.12	0.17	20785	0	0
05-1676E1	121.92	122.53	0.61	0.03	109252	0	0
05-1676E1	122.53	123.44	0.91	0.27	109253	0	0
05-1676E1	123.44	124.66	1.22	0.00	109254	0	0
05-1676E1	124.66	126.19	1.52	0.00	109255	0	0
05-1676E1	126.19	127.04	0.85	0.00	109257	0	0
05-1676E1	167.03	168.55	1.52	0.00	109258	0	0
05-1676E1	168.55	170.08	1.52	0.00	109259	0	0
05-1676E1	170.08	171.60	1.52	0.03	109260	0	0
05-1676E1	171.60	173.13	1.52	0.00	109261	0	0
05-1676E1	173.13	173.89	0.76	0.00	109263	0	0
05-1676E1	173.89	174.80	0.91	0.03	109264	0	0
05-1676E1	174.80	176.33	1.52	0.00	109265	0	0
05-1676E1	176.33	178.00	1.68	0.03	109266	0	0
05-1676E1	178.00	178.61	0.61	0.03	109267	0	0
05-1676E1	178.61	180.14	1.52	0.03	109269	0	0
05-1676E1	180.14	181.05	0.91	0.00	109270	0	0
05-1676E1	181.05	181.97	0.91	0.00	109271	0	0
05-1676E1	181.97	183.03	1.07	0.07	109272	0	0

209
210

05-1676E1	183.03	184.40	1.37	0.62	109273	0	0
05-1676E1	184.40	185.32	0.91	0.93	109275	0	0
05-1676E1	185.32	185.62	0.30	0.45	109276	0	0
05-1676E1	185.62	187.15	1.52	4.08	109277	0	0
05-1676E1	187.15	187.88	0.73	14.43	109278	0	0
05-1676E1	187.88	188.21	0.34	3.22	109279	0	0
05-1676E1	188.21	188.52	0.30	22.01	109280	0	0
05-1676E1	188.52	189.37	0.85	33.02	109281	0	0
05-1676E1	189.37	189.68	0.30	10.25	109283	0	0
05-1676E1	189.68	191.20	1.52	7.58	109284	0	0
05-1676E1	191.20	192.73	1.52	0.21	109285	0	0
05-1676E1	192.73	194.25	1.52	0.10	109286	0	0
05-1676E1	194.25	195.56	1.31	0.07	109287	0	0
05-1676E1	195.56	195.68	0.12	0.24	109288	0	0
05-1676E1	195.68	195.86	0.18	0.10	109289	0	0
05-1676E1	195.86	195.99	0.12	0.07	109290	0	0
05-1676E1	195.99	197.21	1.22	0.00	109291	0	0
05-1676E1	197.21	198.27	1.07	0.03	109293	0	0
05-1676E1	198.27	199.95	1.68	0.00	109294	0	0
05-1676E1	199.95	201.69	1.74	0.00	109295	0	0
05-1676E1	201.69	203.00	1.31	0.00	109296	0	0
05-1676E2	113.60	115.03	1.43	0.00	109299	0	0
05-1676E2	115.03	116.43	1.40	0.00	109300	0	0
05-1676E2	116.43	117.96	1.52	0.00	109301	0	0
05-1676E2	117.96	119.48	1.52	0.00	109302	0	0
05-1676E2	119.48	121.01	1.52	0.00	109303	0	0
05-1676E2	121.01	121.62	0.61	0.00	109304	0	0
05-1676E2	185.29	186.81	1.52	0.00	109306	0	0
05-1676E2	186.81	188.34	1.52	0.00	109307	0	0
05-1676E2	188.34	189.56	1.22	0.00	109308	0	0
05-1676E2	189.56	190.99	1.43	0.07	109309	0	0
05-1676E2	190.99	192.51	1.52	0.00	109310	0	0
05-1676E2	192.51	194.04	1.52	0.00	109312	0	0
05-1676E2	194.04	195.56	1.52	0.03	109313	0	0
05-1676E2	195.56	196.41	0.85	0.03	109314	0	0
05-1676E2	196.41	197.94	1.52	0.03	109316	0	0
05-1676E2	197.94	199.55	1.62	0.10	109317	0	0
05-1676E2	199.55	200.35	0.79	0.10	109318	0	0
05-1676E2	200.35	202.05	1.71	0.14	109319	0	0
05-1676E2	202.05	202.39	0.34	0.07	109320	0	0
05-1676E2	202.39	202.84	0.46	0.14	109322	0	0
05-1676E2	202.84	203.15	0.30	0.10	109323	0	0
05-1676E2	203.15	204.34	1.19	0.72	109324	0	0
05-1676E2	204.34	204.52	0.18	6.48	109325	0	0
05-1676E2	204.52	205.28	0.76	0.31	109327	0	0
05-1676E2	205.28	205.56	0.27	6.93	109328	0	0
05-1676E2	205.56	207.69	2.13	1.92	109329	0	0

210
211

05-1676E2	207.69	207.87	0.18	0.27	109330	0	0
05-1676E2	207.87	208.48	0.61	0.14	109332	0	0
05-1676E2	208.48	210.01	1.52	0.07	109333	0	0
05-1676E2	210.01	210.77	0.76	20.71	109334	0	0
05-1676E2	210.77	212.29	1.52	6.14	109336	0	0
05-1676E2	212.29	213.27	0.98	3.84	109337	0	0
05-1676E2	213.27	214.18	0.91	0.48	109338	0	0
05-1676E2	214.18	215.71	1.52	0.07	109339	0	0
05-1676E2	215.71	217.02	1.31	0.03	109342	0	0
05-1676E2	217.02	218.54	1.52	0.00	109343	0	0
05-1676E2	218.54	219.76	1.22	0.03	109344	0	0
05-1676E2	219.76	220.37	0.61	0.03	109345	0	0
05-1676E2	220.37	221.89	1.52	0.03	109346	0	0
05-1676E2	221.89	223.11	1.22	0.00	109347	0	0
05-1676E2	223.11	224.61	1.49	0.00	109348	0	0
05-1676E2	224.61	226.13	1.52	0.00	109350	0	0
05-1676E2	226.13	227.66	1.52	0.00	109351	0	0
05-300SW4	186.32	188.28	1.95	0.00	109352	0	0
05-300SW4	188.28	189.80	1.52	0.00	109353	0	0
05-300SW4	189.80	191.11	1.31	0.00	109354	0	0
05-300SW4	191.11	192.42	1.31	0.00	109355	0	0
05-300SW4	192.42	193.03	0.61	0.03	109357	0	0
05-300SW4	193.03	194.01	0.98	0.03	109358	0	0
05-300SW4	194.01	194.77	0.76	0.03	109359	0	0
05-300SW4	194.77	195.16	0.40	0.21	109360	0	0
05-300SW4	195.16	195.99	0.82	2.88	109361	0	0
05-300SW4	195.99	197.21	1.22	0.03	109362	0	0
05-300SW4	197.21	198.73	1.52	0.03	109363	0	0
05-300SW4	198.73	200.13	1.40	0.00	109364	0	0
05-300SW4	200.13	200.86	0.73	13.41	109365	0	0
05-300SW4	200.86	203.00	2.13	0.24	109367	0	0
05-300SW4	203.00	204.22	1.22	0.03	109368	0	0
05-300SW4	204.22	204.83	0.61	4.01	109369	0	0
05-300SW4	204.83	205.89	1.07	0.27	109370	0	0
05-300SW4	205.89	206.96	1.07	3.39	109371	0	0
05-300SW4	206.96	208.33	1.37	0.03	109373	0	0
05-300SW4	208.33	209.55	1.22	0.31	109374	0	0
05-300SW4	209.55	210.92	1.37	1.41	109375	0	0
05-300SW4	210.92	212.14	1.22	1.61	109376	0	0
05-300SW4	212.14	212.84	0.70	0.03	109378	0	0
05-300SW4	212.84	213.36	0.52	8.09	109379	0	0
05-300SW4	213.36	214.12	0.76	0.07	109380	0	0
05-300SW4	214.12	214.58	0.46	0.27	109381	0	0
05-300SW4	214.58	215.49	0.91	0.03	109382	0	0
05-300SW4	215.49	216.26	0.76	0.00	109384	0	0
05-300SW4	216.26	216.71	0.46	0.00	109385	0	0
05-300SW4	216.71	217.93	1.22	0.00	109386	0	0

211
212

05-300SW4	217.93	218.18	0.24	0.03	109387	0	0
05-300SW4	218.18	219.46	1.28	0.00	109388	0	0
05-300SW4	219.46	220.80	1.34	0.00	109389	0	0
05-300SW4	220.80	221.38	0.58	0.31	109390	0	0
05-300SW4	221.38	222.20	0.82	0.03	109392	0	0
05-300SW4	222.20	223.72	1.52	0.00	109393	0	0
05-300SW4	223.72	225.25	1.52	0.00	109394	0	0
05-300SW4	225.25	226.77	1.52	0.14	109395	0	0
05-300SW4	226.77	228.30	1.52	0.00	109396	0	0
05-300SW4	228.30	228.78	0.49	0.00	109397	0	0
05-300SW4	228.78	229.39	0.61	0.03	109398	0	0
05-300SW4	229.39	230.92	1.52	0.00	109399	0	0
05-300SW4	230.92	231.22	0.30	0.00	109401	0	0
05-300SW4	231.22	232.04	0.82	0.03	109402	0	0
05-300SW4	232.04	233.57	1.52	0.00	109403	0	0
05-300SW4	233.57	234.39	0.82	0.00	109404	0	0
05-300SW4	234.39	235.31	0.91	0.00	109405	0	0
05-300SW4	235.31	236.83	1.52	0.00	109407	0	0
05-300SW4	236.83	237.13	0.30	0.10	109408	0	0
05-300SW4	237.13	238.90	1.77	0.00	109409	0	0
05-300SW4	238.90	239.09	0.18	0.00	109410	0	0
05-300SW4	239.09	240.49	1.40	0.00	109412	0	0
05-300SW4	240.49	240.94	0.46	0.00	109413	0	0
05-300SW4	240.94	242.01	1.07	0.00	109414	0	0
05-300SW4	242.01	242.93	0.91	0.00	109415	0	0
05-300SW4	242.93	243.54	0.61	0.00	109417	0	0
05-300SW4	243.54	245.06	1.52	0.00	109418	0	0
05-300SW4	245.06	245.67	0.61	0.00	109419	0	0
05-300SW4	245.67	246.13	0.46	0.00	109420	0	0
05-300SW4	246.13	246.89	0.76	0.00	109422	0	0
05-300SW4	246.89	247.38	0.49	0.00	109423	0	0
05-300SW4	247.38	248.47	1.10	0.03	109424	0	0
05-300SW4	248.47	248.96	0.49	0.03	109425	0	0
05-300SW4	248.96	249.54	0.58	0.00	109426	0	0
05-300SW4	249.54	250.06	0.52	0.03	109428	0	0
05-300SW4	250.06	250.97	0.91	0.00	109429	0	0
05-300SW4	250.97	251.40	0.43	0.00	109430	0	0
05-300SW4	251.40	252.22	0.82	0.00	109431	0	0
05-300SW4	252.22	252.62	0.40	0.00	109432	0	0
05-300SW4	252.62	253.84	1.22	0.03	109434	0	0
05-300SW4	253.84	254.33	0.49	0.03	109435	0	0
05-300SW4	254.33	254.63	0.30	0.17	109436	0	0
05-300SW4	254.63	255.24	0.61	1.30	109437	0	0
05-300SW4	255.24	255.85	0.61	0.03	109438	0	0
05-300SW4	255.73	256.34	0.61	0.58	109440	0	0
05-300SW4	256.34	256.95	0.61	3.33	109441	0	0
05-300SW4	256.95	257.56	0.61	3.26	109442	0	0

212
213

05-300SW4	257.56	258.17	0.61	0.03	109443	0	0
05-300SW4	258.17	258.78	0.61	7.75	109444	0	0
05-300SW4	258.78	259.23	0.46	0.48	109445	0	0
05-300SW4	259.23	259.99	0.76	4.11	109446	0	0
05-300SW4	259.99	260.82	0.82	0.75	109448	0	0
05-300SW4	260.82	261.34	0.52	0.21	109449	0	0
05-300SW4	261.34	261.82	0.49	10.90	109451	0	0
05-300SW4	261.82	263.35	1.52	42.21	109452	0	0
05-300SW4	263.35	263.65	0.30	52.80	109453	0	0
05-300SW4	263.65	264.41	0.76	1.92	109454	0	0
05-300SW4	264.41	265.02	0.61	0.24	109455	0	0
05-300SW4	265.02	265.72	0.70	11.55	109457	0	0
05-300SW4	265.72	267.25	1.52	11.25	109458	0	0
05-300SW4	267.25	267.92	0.67	9.57	109459	0	0
05-300SW4	267.92	269.05	1.13	13.54	109460	0	0
05-300SW4	269.05	269.75	0.70	0.86	109461	0	0
05-300SW4	269.75	270.97	1.22	0.10	109463	0	0
05-300SW4	270.97	271.64	0.67	0.03	109464	0	0
05-300SW4	271.64	273.04	1.40	0.03	109465	0	0
05-300SW4	273.04	273.28	0.24	0.00	109466	0	0
05-300SW4	273.28	273.89	0.61	0.00	109467	0	0
05-300SW4	273.89	275.11	1.22	0.00	109468	0	0
05-300SW4	275.11	276.58	1.46	0.03	109469	0	0
05-300SW4	276.58	276.82	0.24	0.03	109471	0	0
05-300SW4	276.82	277.12	0.30	0.03	109472	0	0
05-300SW4	277.12	277.34	0.21	0.03	109473	0	0
05-300SW4	277.34	277.98	0.64	0.00	109474	0	0
05-300SW4	277.98	278.74	0.76	0.00	109475	0	0
05-300SW4	278.74	280.11	1.37	0.00	109476	0	0
05-300SW4	280.11	281.00	0.88	0.00	109477	0	0
05-300SW5	205.44	206.75	1.31	0.00	109479	0	0
05-300SW5	206.75	207.11	0.37	0.03	109480	0	0
05-300SW5	207.11	208.64	1.52	0.03	109481	0	0
05-300SW5	208.64	210.01	1.37	0.03	109482	0	0
05-300SW5	210.01	211.53	1.52	0.00	109483	0	0
05-300SW5	211.53	212.45	0.91	0.00	109484	0	0
05-300SW5	212.45	212.90	0.46	0.00	109486	0	0
05-300SW5	212.90	213.21	0.30	0.03	109487	0	0
05-300SW5	213.21	214.12	0.91	0.03	109488	0	0
05-300SW5	214.12	214.34	0.21	0.03	109489	0	0
05-300SW5	214.34	215.80	1.46	0.10	109490	0	0
05-300SW5	215.80	216.71	0.91	16.66	109492	0	0
05-300SW5	216.71	217.78	1.07	7.03	109493	0	0
05-300SW5	217.78	218.30	0.52	24.10	109494	0	0
05-300SW5	218.30	219.15	0.85	0.45	109495	0	0
05-300SW5	219.15	219.91	0.76	0.48	109496	0	0
05-300SW5	219.91	220.68	0.76	0.07	109498	0	0

213
214

05-300SW5	220.68	221.28	0.61	0.00	109499	0	0
05-300SW5	221.28	221.44	0.15	0.00	109500	0	0
05-300SW5	221.44	222.50	1.07	0.00	109501	0	0
05-300SW5	222.50	224.03	1.52	0.00	109502	0	0
05-300SW5	224.03	224.64	0.61	0.00	109504	0	0
05-300SW5	224.64	226.16	1.52	0.00	109505	0	0
05-300SW5	226.16	227.69	1.52	0.00	109506	0	0
05-300SW5	227.69	229.21	1.52	0.00	109507	0	0
05-300SW5	229.21	230.73	1.52	0.00	109508	0	0
05-300SW5	230.73	232.26	1.52	0.00	109509	0	0
05-300SW5	232.26	233.78	1.52	0.03	109511	0	0
05-300SW5	233.78	235.31	1.52	0.03	109512	0	0
05-300SW5	235.31	236.46	1.16	0.03	109513	0	0
05-300SW5	236.46	236.77	0.30	3.19	109514	0	0
05-300SW5	236.77	238.05	1.28	0.03	109515	0	0
05-300SW5	238.05	239.33	1.28	0.03	109516	0	0
05-300SW5	239.33	239.63	0.30	0.03	109517	0	0
05-300SW5	239.63	240.49	0.85	0.00	109518	0	0
05-300SW5	240.49	241.10	0.61	2.30	109520	0	0
05-300SW5	241.10	241.71	0.61	5.62	109521	0	0
05-300SW5	241.71	242.41	0.70	18.75	109522	0	0
05-300SW5	242.41	243.02	0.61	26.91	109523	0	0
05-300SW5	243.02	243.99	0.98	18.89	109524	0	0
05-300SW5	243.99	245.06	1.07	45.81	109525	0	0
05-300SW5	245.06	245.67	0.61	18.03	109526	0	0
05-300SW5	245.67	246.74	1.07	9.94	109528	0	0
05-300SW5	246.74	247.80	1.07	0.07	109529	0	0
05-300SW5	247.80	248.87	1.07	0.07	109530	0	0
05-300SW5	248.87	249.78	0.91	0.10	109532	0	0
05-300SW5	249.78	250.85	1.07	0.69	109533	0	0
05-300SW5	250.85	251.31	0.46	0.10	109534	0	0
05-300SW5	251.31	251.73	0.43	0.03	109535	0	0
05-300SW5	251.73	252.37	0.64	0.03	109536	0	0
05-300SW5	252.37	252.74	0.37	0.07	109538	0	0
05-300SW5	252.74	254.14	1.40	0.00	109539	0	0
05-300SW5	254.14	254.81	0.67	0.00	109540	0	0
05-300SW5	254.81	256.34	1.52	0.00	109541	0	0
05-300SW5	256.34	257.71	1.37	0.00	109543	0	0
05-300SW5	257.71	258.78	1.07	0.00	109544	0	0
05-300SW5	258.78	260.09	1.31	0.03	109545	0	0
05-300SW6	215.34	216.41	1.07	0.00	109546	0	0
05-300SW6	216.41	217.17	0.76	0.00	109547	0	0
05-300SW6	217.17	217.93	0.76	0.00	109549	0	0
05-300SW6	217.93	219.46	1.52	0.00	109550	0	0
05-300SW6	219.46	220.98	1.52	0.00	109551	0	0
05-300SW6	220.98	221.44	0.46	0.00	109552	0	0
05-300SW6	221.44	222.20	0.76	0.00	109553	0	0

214
215

05-300SW6	222.20	223.11	0.91	0.00	109554	0	0
05-300SW6	223.11	224.64	1.52	0.00	109555	0	0
05-300SW6	224.64	225.86	1.22	0.00	109557	0	0
05-300SW6	225.86	227.38	1.52	0.00	109558	0	0
05-300SW6	227.38	228.90	1.52	0.00	109559	0	0
05-300SW6	228.90	230.43	1.52	0.00	109560	0	0
05-300SW6	230.43	231.95	1.52	0.00	109561	0	0
05-300SW6	231.95	233.54	1.58	0.00	109562	0	0
05-300SW6	233.54	234.70	1.16	11.69	109563	0	0
05-300SW6	234.70	236.22	1.52	14.61	109564	0	0
05-300SW6	236.22	236.68	0.46	22.49	109565	0	0
05-300SW6	236.68	237.13	0.46	12.99	109566	0	0
05-300SW6	237.13	237.59	0.46	23.31	109567	0	0
05-300SW6	237.59	238.66	1.07	33.02	109568	0	0
05-300SW6	238.81	239.57	0.76	8.67	109570	0	0
05-300SW6	239.57	240.79	1.22	13.65	109571	0	0
05-300SW6	240.79	242.32	1.52	0.51	109572	0	0
05-300SW6	242.32	243.84	1.52	0.82	109573	0	0
05-300SW6	243.84	245.36	1.52	0.14	109574	0	0
05-300SW6	245.36	246.89	1.52	0.27	109575	0	0
05-300SW6	246.89	248.41	1.52	0.82	109577	0	0
05-300SW6	248.41	249.33	0.91	0.00	109578	0	0
05-300SW6	249.33	250.09	0.76	5.62	109579	0	0
05-300SW6	250.09	250.39	0.30	0.00	109580	0	0
05-300SW6	250.39	251.76	1.37	0.24	109581	0	0
05-300SW6	251.76	252.98	1.22	0.03	109582	0	0
05-300SW6	252.98	253.59	0.61	0.34	109583	0	0
05-300SW6	253.59	255.12	1.52	0.00	109584	0	0
05-300SW6	255.12	256.64	1.52	0.03	109586	0	0
05-300SW6	256.64	258.17	1.52	0.03	109587	0	0
05-300SW6	258.17	259.54	1.37	0.03	109588	0	0
05-300SW6	259.54	260.91	1.37	0.03	109589	0	0
05-300SW6	260.91	261.82	0.91	5.18	109590	0	0
05-300SW6	261.82	262.43	0.61	40.11	109591	0	0
05-300SW6	262.43	263.04	0.61	24.31	109593	0	0
05-300SW6	263.04	263.65	0.61	23.90	109594	0	0
05-300SW6	263.65	264.05	0.40	30.69	109595	0	0
05-300SW6	264.05	264.87	0.82	12.55	109596	0	0
05-300SW6	264.87	266.09	1.22	10.70	109597	0	0
05-300SW6	266.09	267.46	1.37	0.38	109599	0	0
05-300SW6	267.46	267.92	0.46	0.38	109600	0	0
05-300SW6	267.92	269.44	1.52	0.03	109601	0	0
05-300SW6	269.44	270.36	0.91	0.03	109602	0	0
05-300SW6	327.05	327.36	0.30	0.14	109604	0	0
05-300SW6	327.36	328.57	1.22	0.00	109605	0	0
05-300SW6	328.57	329.43	0.85	0.51	109606	0	0
05-300SW6	329.43	330.10	0.67	0.03	109607	0	0

²¹⁵
216

05-300SW6	330.10	331.47	1.37	0.14	109608	0	0
05-300SW6	331.47	332.84	1.37	0.62	109610	0	0
05-300SW6	332.84	334.37	1.52	0.51	109611	0	0
05-300SW6	334.37	335.28	0.91	0.07	109612	0	0
05-300SW6	335.28	336.80	1.52	0.03	109613	0	0
05-300SW6	336.80	338.33	1.52	0.07	109614	0	0
05-300SW6	338.33	339.85	1.52	0.00	109615	0	0
05-300SW6	339.85	340.16	0.30	0.03	109616	0	0
05-300SW6	340.16	341.38	1.22	0.00	109617	0	0
05-300SW6	341.38	342.75	1.37	0.03	109618	0	0
05-300SW6	342.75	344.12	1.37	0.00	109619	0	0
05-300SW6	344.12	345.49	1.37	0.00	109620	0	0
05-300SW6	345.49	346.56	1.07	0.00	109621	0	0
05-300SW6	346.56	347.17	0.61	0.00	109622	0	0
05-330SW3	204.22	205.13	0.91	0.14	109788	0	0
05-330SW3	205.13	206.20	1.07	6.93	109789	0	0
05-330SW3	206.20	206.47	0.27	2.37	109790	0	0
05-330SW3	206.47	207.48	1.01	7.17	109791	0	0
05-330SW3	207.48	207.87	0.40	19.10	109792	0	0
05-330SW3	207.87	209.06	1.19	7.41	109793	0	0
05-330SW3	209.06	209.67	0.61	0.14	109794	0	0
05-330SW3	209.67	210.59	0.91	0.07	109796	0	0
05-330SW3	210.59	212.11	1.52	0.07	109797	0	0
05-330SW3	212.11	213.06	0.94	0.03	109798	0	0
05-330SW3	213.06	213.94	0.88	0.14	109799	0	0
05-330SW3	213.94	214.43	0.49	10.11	109801	0	0
05-330SW3	214.43	214.82	0.40	18.45	109802	0	0
05-330SW3	214.82	215.65	0.82	16.94	109803	0	0
05-330SW3	215.65	216.10	0.46	18.34	109804	0	0
05-330SW3	216.10	216.41	0.30	8.50	109806	0	0
05-330SW3	216.41	217.93	1.52	7.30	109807	0	0
05-330SW3	217.93	219.15	1.22	7.44	109808	0	0
05-330SW3	219.15	220.68	1.52	7.27	109809	0	0
05-330SW3	220.68	222.11	1.43	10.80	109810	0	0
05-330SW3	222.11	223.11	1.01	6.65	109812	0	0
05-330SW3	223.11	223.42	0.30	0.07	109813	0	0
05-330SW3	223.42	223.88	0.46	2.13	109814	0	0
05-330SW3	223.88	224.94	1.07	1.06	109815	0	0
05-330SW3	224.94	225.70	0.76	0.14	109816	0	0
05-330SW3	225.70	226.31	0.61	0.03	109818	0	0
05-330SW3	226.31	227.38	1.07	0.00	109819	0	0
05-330SW3	227.38	227.99	0.61	0.03	109820	0	0
05-330SW3	227.99	229.67	1.68	0.00	109821	0	0
05-330SW3	229.67	230.12	0.46	0.00	109822	0	0
05-330SW3	230.12	230.73	0.61	1.10	109823	0	0
05-330SW3	230.73	232.56	1.83	0.03	109824	0	0
05-330SW3	232.56	233.63	1.07	0.45	109825	0	0

216
217

05-330SW3	251.76	252.68	0.91	0.00	109827	0	0
05-330SW3	252.68	253.59	0.91	0.10	109828	0	0
05-330SW3	270.05	270.66	0.61	1.85	109829	0	0
05-330SW3	270.66	270.97	0.30	0.10	109830	0	0
05-330SW3	270.97	271.58	0.61	0.00	109831	0	0
05-330SW3	271.58	272.03	0.46	0.27	109832	0	0
05-330SW3	272.03	273.10	1.07	0.00	109834	0	0
05-330SW3	273.10	274.62	1.52	0.07	109835	0	0
05-330SW3	274.62	275.54	0.91	0.00	109836	0	0
05-330SW3	275.54	276.76	1.22	0.34	109837	0	0
05-330SW3	276.76	277.37	0.61	0.31	109838	0	0
05-330SW3	277.37	277.98	0.61	0.03	109839	0	0
05-330SW4	138.38	138.68	0.30	4.53	109676	0	0
05-330SW4	138.68	139.42	0.73	1.47	109677	0	0
05-330SW4	139.42	140.51	1.10	0.03	109678	0	0
05-330SW4	140.51	140.82	0.30	0.51	109679	0	0
05-330SW4	140.82	141.73	0.91	0.86	109680	0	0
05-330SW4	141.73	142.04	0.30	0.00	109681	0	0
05-330SW4	197.82	199.34	1.52	0.00	109682	0	0
05-330SW4	199.34	200.16	0.82	0.00	109683	0	0
05-330SW4	200.16	200.86	0.70	0.03	109684	0	0
05-330SW4	200.86	201.78	0.91	0.00	109686	0	0
05-330SW4	201.78	202.27	0.49	0.03	109687	0	0
05-330SW4	202.27	203.00	0.73	0.03	109688	0	0
05-330SW4	203.00	203.76	0.76	0.99	109689	0	0
05-330SW4	203.76	204.37	0.61	0.38	109690	0	0
05-330SW4	204.37	205.74	1.37	8.16	109691	0	0
05-330SW4	205.74	206.96	1.22	0.45	109692	0	0
05-330SW4	206.96	208.64	1.68	0.55	109694	0	0
05-330SW4	208.64	209.09	0.46	0.10	109695	0	0
05-330SW4	209.09	209.70	0.61	0.03	109696	0	0
05-330SW4	209.70	210.46	0.76	1.89	109697	0	0
05-330SW4	210.46	211.35	0.88	0.03	109698	0	0
05-330SW4	211.35	212.26	0.91	0.24	109699	0	0
05-330SW4	212.26	213.06	0.79	2.54	109700	0	0
05-330SW4	213.06	214.58	1.52	0.31	109701	0	0
05-330SW4	214.58	215.28	0.70	13.41	109702	0	0
05-330SW4	215.28	215.89	0.61	2.06	109703	0	0
05-330SW4	215.89	216.07	0.18	0.21	109704	0	0
05-330SW4	216.07	217.63	1.55	27.60	109705	0	0
05-330SW4	217.63	218.85	1.22	0.45	109706	0	0
05-330SW4	218.85	219.15	0.30	0.00	109708	0	0
05-330SW4	219.15	219.76	0.61	0.03	109709	0	0
05-330SW4	219.76	220.68	0.91	1.06	109710	0	0
05-330SW4	220.68	221.22	0.55	0.10	109711	0	0
05-330SW4	221.22	221.89	0.67	0.72	109713	0	0
05-330SW4	221.89	223.36	1.46	6.17	109714	0	0

05-330SW4	223.36	224.67	1.31	25.71	109715	0	0
05-330SW4	224.67	225.06	0.40	48.62	109716	0	0
05-330SW4	225.06	226.28	1.22	8.78	109717	0	0
05-330SW4	226.28	227.26	0.98	27.29	109718	0	0
05-330SW4	227.26	228.39	1.13	36.00	109720	0	0
05-330SW4	228.39	228.57	0.18	58.11	109721	0	0
05-330SW4	228.57	228.87	0.30	6.82	109722	0	0
05-330SW4	228.87	229.36	0.49	47.01	109723	0	0
05-330SW4	229.36	229.73	0.37	22.22	109724	0	0
05-330SW4	229.73	230.64	0.91	25.61	109725	0	0
05-330SW4	230.64	231.95	1.31	8.98	109726	0	0
05-330SW4	231.95	232.56	0.61	1.58	109727	0	0
05-330SW4	232.56	233.78	1.22	11.86	109728	0	0
05-330SW4	233.78	234.39	0.61	0.07	109730	0	0
05-330SW4	234.39	235.92	1.52	0.07	109731	0	0
05-330SW4	235.92	237.44	1.52	0.00	109732	0	0
05-330SW4	237.44	238.96	1.52	0.00	109733	0	0
05-330SW4	238.96	240.49	1.52	0.00	109734	0	0
05-330SW4	240.49	242.01	1.52	0.00	109735	0	0
05-330SW4	242.01	243.54	1.52	0.00	109736	0	0
05-330SW4	243.54	244.14	0.61	0.00	109737	0	0
05-330SW4	244.14	244.60	0.46	0.00	109738	0	0
05-330SW4	244.60	245.67	1.07	0.00	109739	0	0
05-330SW4	245.67	245.97	0.30	0.00	109740	0	0
05-330SW4	245.97	246.43	0.46	0.00	109741	0	0
05-330SW4	246.43	247.65	1.22	0.00	109743	0	0
05-330SW4	247.65	249.72	2.07	0.17	109744	0	0
05-330SW4	249.72	250.55	0.82	0.21	109745	0	0
05-330SW4	250.55	252.07	1.52	0.03	109746	0	0
05-330SW4	252.07	253.59	0.00	0.24	109748	0	0
05-330SW4	253.59	255.27	1.68	0.00	109749	0	0
05-330SW4	255.27	256.34	1.07	0.00	109750	0	0
05-330SW4	256.34	257.65	1.31	0.07	109751	0	0
05-330SW4	257.65	258.47	0.82	0.03	109753	0	0
05-330SW4	258.47	259.23	0.76	0.00	109754	0	0
05-330SW4	259.23	260.30	1.07	0.03	109755	0	0
05-330SW4	260.30	261.82	1.52	0.00	109756	0	0
05-330SW4	261.82	263.35	1.52	0.00	109757	0	0
05-330SW4	263.35	264.23	0.88	0.00	109758	0	0
05-330SW4	264.23	265.18	0.94	0.00	109759	0	0
05-330SW4	265.18	265.79	0.61	0.00	109760	0	0
05-330SW4	265.79	267.61	1.83	0.00	109761	0	0
05-330SW4	267.61	268.22	0.61	0.00	109762	0	0
05-330SW4	268.22	269.75	1.52	0.00	109763	0	0
05-330SW4	269.75	270.81	1.07	0.00	109764	0	0
05-330SW4	270.81	271.88	1.07	0.00	109765	0	0
05-330SW4	271.88	273.19	1.31	0.55	109766	0	0

²¹⁸
219

05-330SW4	273.19	274.02	0.82	0.00	109768	0	0
05-330SW4	274.02	275.54	1.52	0.00	109769	0	0
05-330SW4	275.54	276.64	1.10	0.00	109770	0	0
05-330SW4	276.64	278.16	1.52	0.00	109771	0	0
05-330SW4	278.16	279.68	1.52	0.00	109772	0	0
05-330SW4	279.68	281.21	1.52	0.00	109773	0	0
05-330SW4	281.21	282.85	1.65	0.00	109775	0	0
05-330SW4	282.85	283.77	0.91	0.03	109776	0	0
05-330SW4	283.77	285.87	2.10	0.27	109777	0	0
05-330SW4	285.87	286.82	0.94	0.00	109779	0	0
05-330SW4	286.82	288.34	1.52	0.00	109780	0	0
05-330SW4	288.34	289.86	1.52	0.00	109781	0	0
05-330SW4	289.86	291.08	1.22	0.00	109782	0	0
05-330SW4	291.08	292.61	1.52	0.00	109783	0	0
05-330SW4	292.61	293.83	1.22	0.00	109784	0	0
05-330SW4	293.83	295.35	1.52	0.00	109786	0	0
05-330SW5	133.14	133.81	0.67	0.00	109626	0	0
05-330SW5	133.81	135.21	1.40	0.00	109627	0	0
05-330SW5	135.21	135.70	0.49	0.00	109628	0	0
05-330SW5	135.70	136.86	1.16	0.03	109629	0	0
05-330SW5	226.10	226.77	0.67	0.00	109631	0	0
05-330SW5	226.77	227.84	1.07	0.00	109632	0	0
05-330SW5	227.84	229.21	1.37	0.00	109633	0	0
05-330SW5	229.21	230.12	0.91	0.00	109634	0	0
05-330SW5	230.12	230.89	0.76	0.65	109636	0	0
05-330SW5	230.89	231.22	0.34	3.98	109637	0	0
05-330SW5	231.22	232.11	0.88	0.03	109638	0	0
05-330SW5	232.11	233.17	1.07	8.67	109639	0	0
05-330SW5	233.17	234.39	1.22	10.94	109640	0	0
05-330SW5	234.39	235.61	1.22	10.80	109641	0	0
05-330SW5	235.61	236.07	0.46	5.55	109642	0	0
05-330SW5	236.07	236.52	0.46	12.34	109643	0	0
05-330SW5	236.52	237.44	0.91	12.10	109644	0	0
05-330SW5	237.44	238.66	1.22	1.61	109645	0	0
05-330SW5	238.66	239.45	0.79	4.18	109647	0	0
05-330SW5	239.45	240.27	0.82	0.07	109648	0	0
05-330SW5	240.27	240.49	0.21	0.03	109649	0	0
05-330SW5	240.49	241.07	0.58	4.70	109650	0	0
05-330SW5	241.07	241.40	0.34	2.85	109652	0	0
05-330SW5	241.40	241.68	0.27	7.75	109653	0	0
05-330SW5	241.68	242.01	0.34	0.58	109654	0	0
05-330SW5	242.01	242.47	0.46	2.43	109655	0	0
05-330SW5	242.47	243.17	0.70	1.82	109657	0	0
05-330SW5	243.17	243.54	0.37	6.99	109658	0	0
05-330SW5	243.54	244.91	1.37	30.62	109659	0	0
05-330SW5	244.91	246.28	1.37	13.61	109660	0	0
05-330SW5	246.28	247.50	1.22	0.14	109662	0	0

²¹⁹
220

05-330SW5	247.50	248.38	0.88	0.07	109663	0	0
05-330SW5	248.38	248.72	0.34	0.10	109664	0	0
05-330SW5	248.72	249.63	0.91	0.03	109665	0	0
05-330SW5	270.97	271.58	0.61	0.00	109667	0	0
05-330SW5	271.58	272.80	1.22	0.03	109668	0	0
05-330SW5	272.80	273.41	0.61	0.03	109669	0	0
05-330SW5	273.41	274.02	0.61	0.00	109670	0	0
05-330SW5	274.02	275.54	1.52	0.00	109671	0	0
05-330SW5	275.54	277.06	1.52	0.10	109672	0	0
05-330SW5	277.06	277.37	0.30	0.31	109673	0	0
05-330SW5	277.37	278.89	1.52	0.03	109674	0	0