New Polaris Gold Mines Ltd.

Canarc Resource Corp.

New Polaris Project

Metallurgical Research Report

Atlin Mining Division
NTS 104K/12
N 58 42' lat., W 133 37' long.

by

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1.0 Introduction

1.1 Location

The New Polaris Gold Mine (formerly Polaris –Taku Gold Mines) is located in the Atlin mining district approximately 140 kilometres south of the town of Atlin, British Columbia, and approximately 60 kilometres east of Juneau, Alaska. The index map (figure 1.1) indicates the relative location of the property.

The property is located at approximately 133 37' W longitude and 58 42' N latitude on the west shore of the Tulsequah River, approximately 6 miles north of its' confluence with the Taku River.

1.2 Access

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

The property is accessible only by air. The most proximal airstrips of significant scale are located in Juneau and Atlin. The nearest road access terminates 7 kilometres south of Atlin, and 18 kilometres southeast of Juneau. Historically the property was serviced by a barge landing on the west bank of the Taku River just downstream of its' conflurence with the Tulsequah River, however heavy silt deposition in the Taku Inlet over the last fifty years now limits river traffic to small jet boats capable of carrying passengers only.

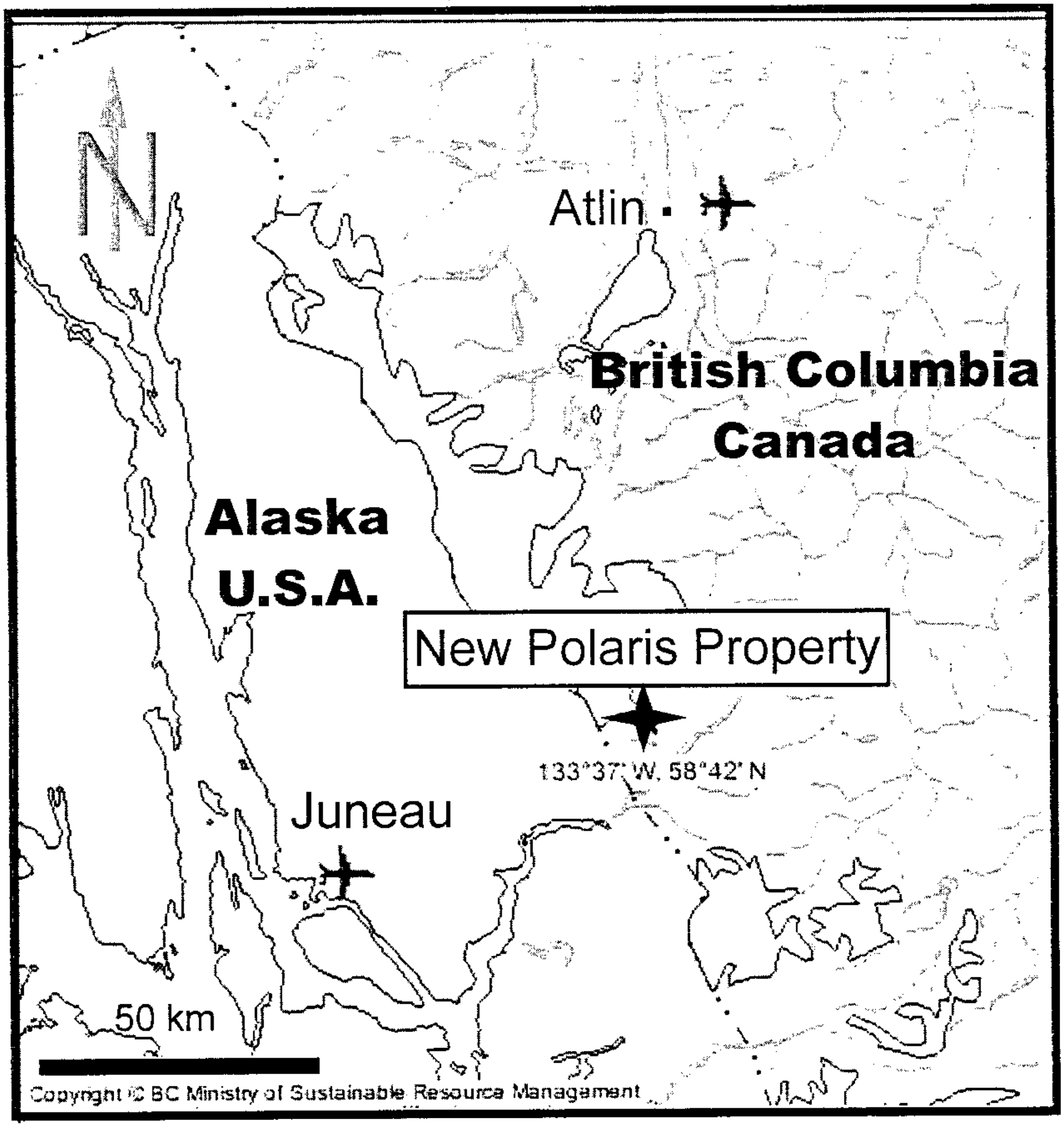
Two airstrips service the property. A 400 metre gravel strip is located on the property and the second is located approximately 4.5 kilometres to the south. The historic road access to this strip has fallen into disrepair, with side-channels of the Tulsequah River cutting it off at a few points.. The length and condition of the second strip is undetermined. The present operators have not used this strip in the past ten years. Being proximal to the river it is subject to annual erosion, and therefore a careful study would be necessitated to determine the overall suitability for use. In the past this strip was long enough for larger aircraft such as the DC 3 and DC 4. The first strip is suitable for use by fully loaded Cessna's, Beaver's, and larger capacity aircraft with STOL capability such as the Short's Skyvan.

On the property there are numerous roads and trails. These have been left from previous mining activities, and have been in some cases modified. The roads are suitable for use by small pick-up or all-terrain vehicle. The trails are suitable for foot passage only.

Figure 1.2 indicates the approximate location of the roads and airstrips about the property.

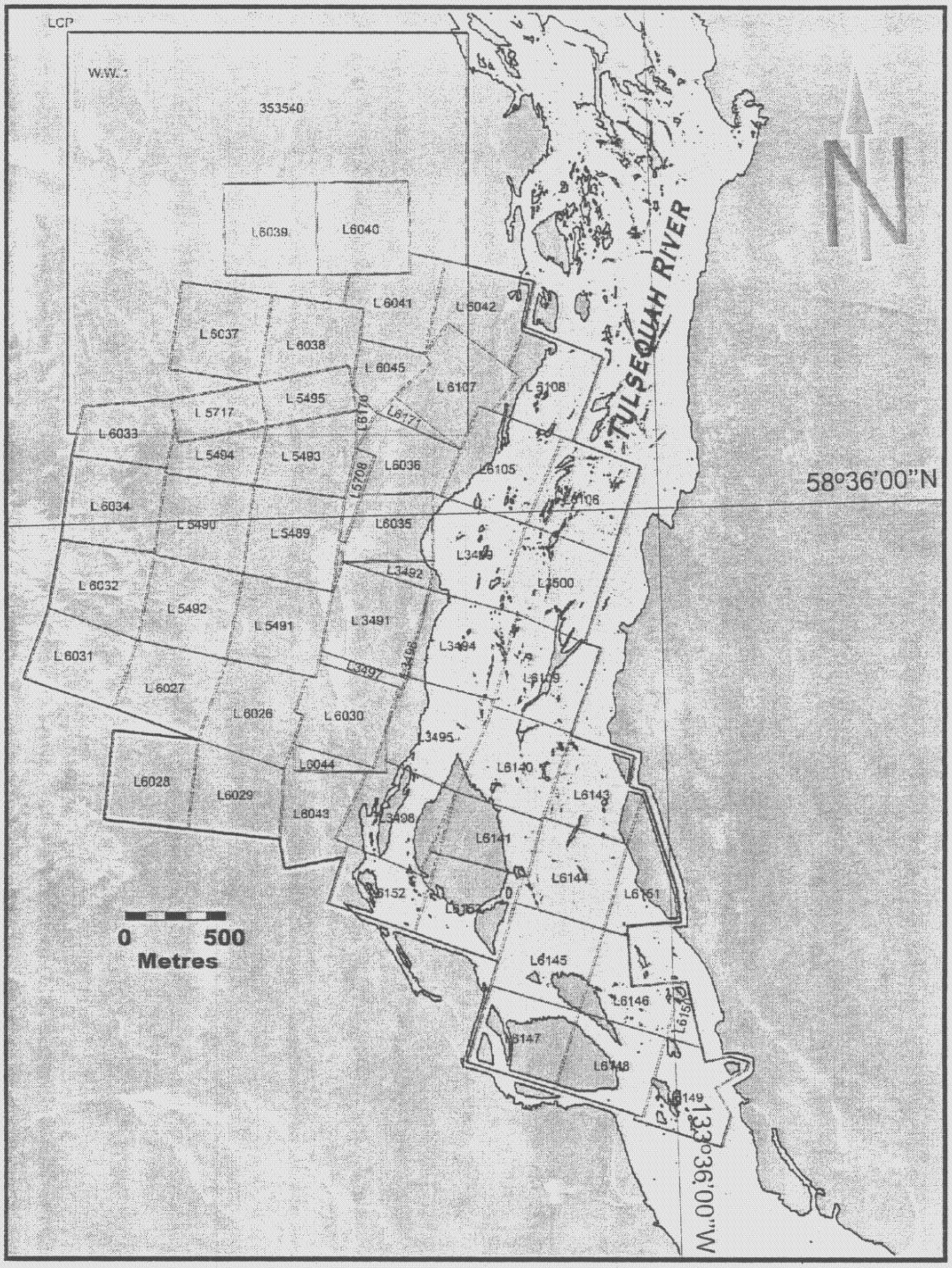
1.3 Climate

The climate is very characteristic of this section of the British Columbia coast, with heavy rainfall prevailing during the late summer and fall months, and comparatively heavy snowfall, interspersed with rain during the winter. The annual precipitation is approximately seventy-five inches of which twenty-eight inches occurs as rainfall. The snow seldom accumulates to a depth greater than five feet on the level. Winter temperatures are not severe and rarely fall below 10 degrees below zero Fahrenheit. Summer temperatures in July average 60°F with daytime temperatures reaching the high 80's on occasion. The vegetation is typical of northern temperature rain forest, consisting primarily of fir, hemlock, spruce and cedar forest on the hillsides and aspen and alder groves in the river valley.



Canarc Resource Corp
New Polaris Project
Project Location Map

Figure 1.1



NTS 104K/12
Atlin Mining Division

Canarc Resource Corp
New Polaris Project
Claim Map

Figure 1.2

1.4 Physiography

Extensive glaciation of recent age has been the dominant factor in topographic development. The Taku and Tulsequah Rivers, which dissect the area, provide its most striking features, with their broad valleys 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 ten miles 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 (Jokülhlaup) during the spring thaw every year, flooding the Tulsequah and Taku valleys below for three to five days.

Rugged relief characterizes the project site as alpine glaciation has been the pincised U-shaped valleys and . Located in the Coast Mountain Range, the topography of the area ranges between sea level and 2600 metres. The elevation of the New Polaris property ranges from 10 metres asl. on the Tulsequah River valley floor to 730 metres asl. on the eastern flanks of Whitewater Mountain..

Recent alpine glaciation typifies The physiography an area, which has been influenced by recent glaciation. The Taku and Tulsequah Rivers are broad till-filled valleys commonly more than 1.5 kilometres wide. The vertical extent of the till is unknown, however past seismic work and surface diamond drilling indicate thicknesses increasing away from the margins of the valley floor to a maximum of approximately 240 metres at its centre.

Outcrops on the property are scarce and represent only 5% of the total surface area.

1.5 Vegetation

Vegetation on the property consists of heavily forested slopes and floodplains. The forest was cutover approximately 50 years ago to provide lumber for the mining and residential requirements of the day. Today the forest is composed of cottonwoods, alders and brush in the near floodplain area, and spruce, hemlock and red on the mountain slopes to elevation of approximately 3500'. Above this elevation alpine vegetation (sedge, heather) prevails. This secondary growth is mature enough to provide a valuable resource for current mining purposes.

1.6 Wildlife

Wildlife observed in the area includes moose, black bear, grizzly bear, mountain goat, wolf and wolverine along with other small mammals. Trumpeter swans, bald eagles, rock ptarmigan and grouse are indigenous to the area. The disruption of flow in the Tulsequah River by the seasonal glacial outbursts provides for a negligible commercial fishery, however, the Whitewater and Shazah creeks, and pools in the Tulsequah River flood plain have been identified as spawning and rearing habitat for Steelhead, Dolly Varden char, and all five species of Pacific salmon.

2.0 Property 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 percent.

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 by 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.

An Edwards roaster and a cyanide plant to produce bullion were installed and tested in1949 in order to improve recovery and reduce shipping cost of concentrates to the Tacoma smelter. The addition of the roaster 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 bought out Rembrandt's interest in 1994 and has continued exploration up to the present. The Canarc's subsidiary New Polaris Gold Mines (formerly Golden Angus Mines Ltd.) currently operates the property.

Some pieces of major equipment were removed from the Polaris-Taku site since closure (crushers and mills) and no further mining has taken place on the property. Most of the buildings have been removed. A few of the townsite houses and the mechanical shop are in reasonably good condition and are currently being used for exploration purposes.

Although the underground equipment has been removed, the hoist, sheaves and conveyances are still in place. The "AJ" level was reopened in 1990. Ground conditions are excellent and there is excellent airflow throughout. The shaft is in excellent condition and required very little repair to facilitate the 1997 underground program. The "Polaris" adit collar was also re-established at this time and used as the primary access for that program.

3.0 Geology

3.1 Regional Geology

The Polaris-Taku mine lies within the Intermontane Province of the Western Cordillera approximately 4.5 kilometres from its western contact with the Coast Plutonic Complex. This portion of the Intermontane belt is predominantly composed of the lowermost sections of the Stikine Terrane.

The Whitewater Suite represents the oldest rocks in the area, possibly early Palaeozoic in age, and dominates the geology on the western edge of the property. It consists primarily of strongly metamorphosed and deformed quartzite and quartz-rich graphitic schist with interlayers of mafic, ultramafics, marble, and gneiss. The Whitewater Suite grades to the east into the less metamorphosed metavolcanic and metasedimentary rocks of the Mount Stapler Suite. Continuing east across the property, the Mount Stapler Suite is in fault contact with the similar yet less deformed mid to upper greenschist facies metamorphosed rocks of the Mount Eaton Suite.

This group of rocks hosts the deposit and composes the northeastern third of the property. It dominates the geology on the opposite (eastern) bank of the Tulsequah River where it hosts the Tulsequah Chief and Big Bull volcanogenic massive sulphide Cu-Pb-Zn-Au-Ag deposits.

3.2 Regional Structural Geology

The structural trend in the area is north-northwest to south-southeast, parallel to the alignment of the Late Cretaceous-Tertiary Coast Plutonic Complex which dominates the geology immediately west of the B.C.-Alaska international border. The older rocks have been intensely folded, sheared and deformed into broad doubly plunging symmetrical folds with large amplitudes.

The Mount Stapler and Mount Eaton Suites are separated by the Llewellyn fault which is a regionally significant north-south structure having a long history of movement. The most recent movement has been dextral (west side to the north). Slightly north of the Tulsequah Chief deposit on the east bank of the Tulsequah River, the Llewellyn fault is truncated and offset to the west onto the Polaris-Taku property by the east-west oriented Chief Cross fault.

3.3 Property Geology

The Mount Eaton Suite is a package of weakly metamorphosed volcanic rocks within the middle to upper Palaeozoic Stikine assemblage, which constitutes the basement of the Stikine Terrane. The Polaris-Taku portion of this suite is composed predominantly of basaltic to andesitic augite-phyric volcaniclastics and associated intrusives with lesser amounts of limestone, serpentinised ultramafics, and gabbro. These volcaniclastic and sedimentary units are northwest to north striking, vertical to steeply dipping and range in character from laminated ash to coarse tuffbreccia. Radical lateral facies changes typical of a dynamic depositional environment preclude reliable correlation of individual units across significant distances and thus hamper accurate interpretation of large-scale folding and fault offsets.

3.4 Property Structural Geology

All of the strata within the property have been subjected to compression, rotation, and subsequent extension. Figure 3.1 shows the prominent structural orientation, which is characterised by folds

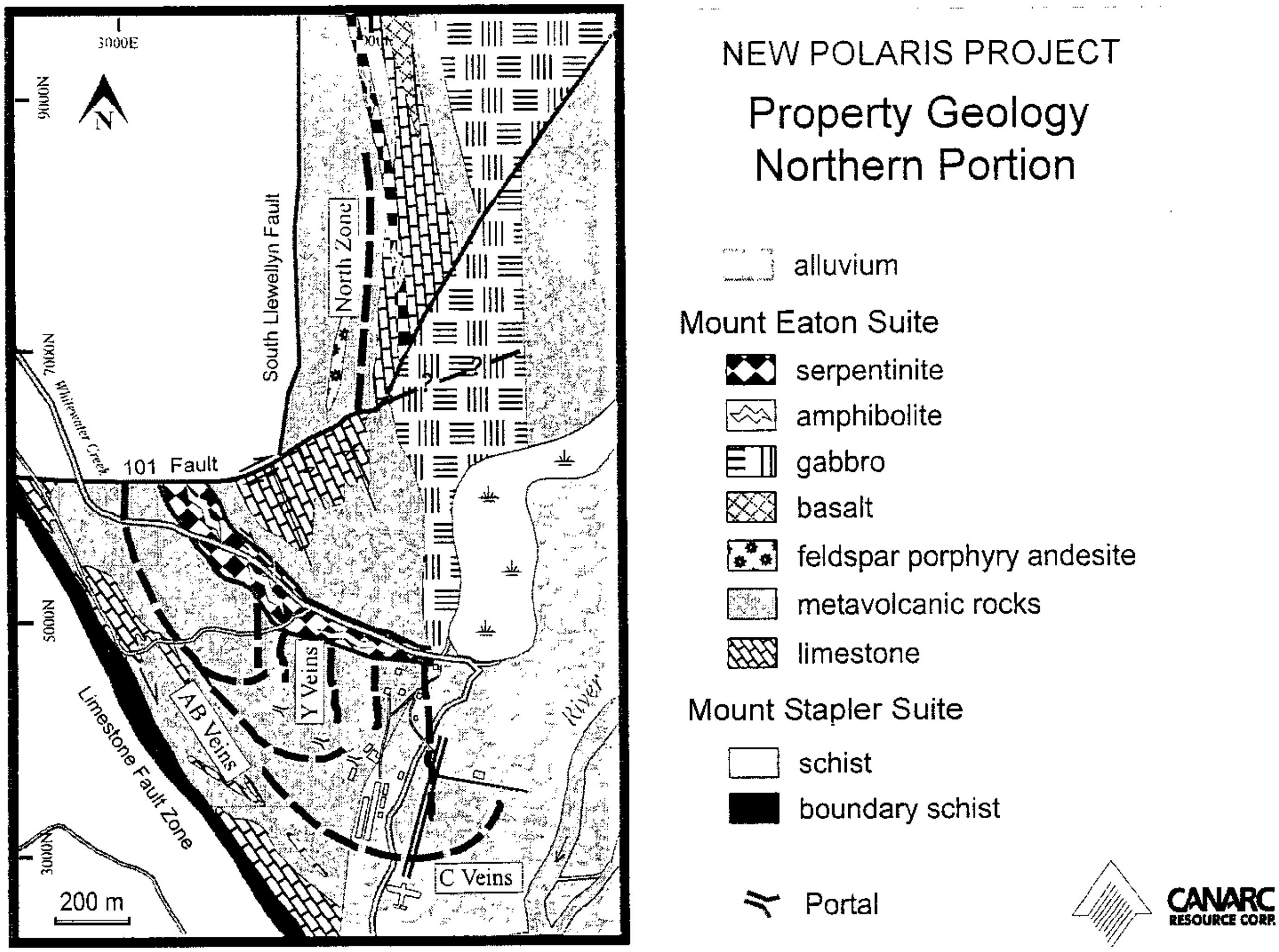


Figure 3.1

trending, northwest southeast and plunging to the southeast. The plunge of folds appears to be variable though generally shallow. Pervasive, weak to moderate, bedding-parallel flattening across the property is suggested both by the absence of oblique fabrics and by local strongly foliated zones having the same attitude as bedding within weakly foliated units. Small-scale isoclinal and intra-folial folds strike north-north-westerly and plunge moderately to the north. This is typical across most of the Mount Eaton suite. Numerous faults are found on the property, the more significant of which are summarised below.

The possible extension of the Llewellyn fault, termed the South Llewellyn fault, continues south from the Chief Cross fault along mine grid co-ordinate 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 on the opposite side of Whitewater Creek. This northwest-southeast oriented 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 by the Whitewater Creek Schist Zone, a zone of schistose chlorite-amphibolite-serpentinite altered andesitic to ultramafic rocks 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, dip approximately 45' to the northeast, and are subparallel 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 shows no displacement, as it apparently parallels the A-B vein system. Between the No. 1 and No. 5 faults, the subparallel Nos. 2, 3 and 4 faults have been mapped in the upper levels. Displacement along these three faults is poorly defined, although movement of up to 30 feet is observed. Nos. 2, 3 and 4 faults appear to converge into a single fault and to weaken with depth.

3.5 <u>Deposit Geometry</u>

The property geology, figure 3.2, and mine workings, figure 3.3, diagrams show the general structure of the veins. The distribution outlines the wedge shape, the predominant orientations and continuity of the zones, and the overall plunge of the system to the southeast. An early interpretation of the structure shows that various veins appear to meet and form "junction arcs" where both thickness and grade improve.

The most prominent vein orientations are: northwest striking and southwest dipping, the "A-B" veins; north striking and east dipping, the 'Y" veins; and the less extensive but economically important east to northeast striking and south dipping zones at the intersection of the previously mentioned vein sets. Recent workers interpreted these zones initially as "junction arcs". Historically they were known individually as the 25 vein, the 1-3-5 veins, and the deep, similarity oriented component of the current resource, the "C" vein. Up to 75% of total past production came from within 300 feet to either side of these junction arc centre lines. The recently discovered north zone bears many similarities to the AB zone and is interpreted as its fault offset northward continuation.

The arcuate nature of the deposit geometry within the mine wedge lends itself to an interpretation as a fold structure. However, this interpretation is not supported by both the intersection relationships between the ore structures and regional foliation, and textural differences between the individual vein sets.

More detailed analysis of the vein sets (Rhys, 1992) suggests they developed as a conjugate shear system during northeast to southwest compression. Respectively, the AB and Y vein systems represent the sinistral and dextral shears, while the Junction Arc and C vein systems represent the tensional component of the conjugate system.

3.6 <u>Mineralisation</u>

Mineralisation of the Polaris-Taku deposit bears strong similarities to many Archean Lode gold deposits such as the arsenical gold camp of Red Lake, Ontario, and the deposits of the Wiluna Belt in Western Australia.

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. Age dating by 'Ar / "Ar on sericite/fuchsite alteration associated with mineralisation provides an early tertiary date of 63.44 Ma. Gold is associated primarily with arsenopyrite and to a lesser extent with pyrite. Arsenopyrite is very fine-grained (< 1 mm) and acicular with a mode of occurrence commonly referred to as the "replacement type". It is observed as pervasive and patchy disseminations in altered wall rocks proximal to, or as breccia fragments within, quartz-ankerite veined shear zones.

Stibnite has no apparent influence on gold grade as it is found only in what are interpreted to be post-mineralisation quartz veinlets. It is also fine to very fine grained, frequently occurring as styolitic stringers within quartz.

The quartz-ankerite veining associated with the ore is itself not mineralised. Evidence from drillcore and historic observations confirm that with an increase in the proportion of veining comes a corresponding dilution of the ore zone. Ankerite veining may represent a slightly later veining event, which exploited the same structurally prepared fluid pathways as those which may have carried the earlier silica-sulphide-gold mineralisation. This is displayed quite well by the abundant breccia veining throughout the deposit where angular wall-rock fragments, both altered and unaltered, mineralised and unmineralised, are suspended in an ankerite-quartz vein matrix. Sulphide rich ribbon veinlets may represent mineralised host rocks, which have undergone subsequent flattening/shearing and veining.
Gold is distributed in three weakly defined populations, 0 to 0. 12, 0.12 to 0. 52, and greater than 0.52 oz Au/t.

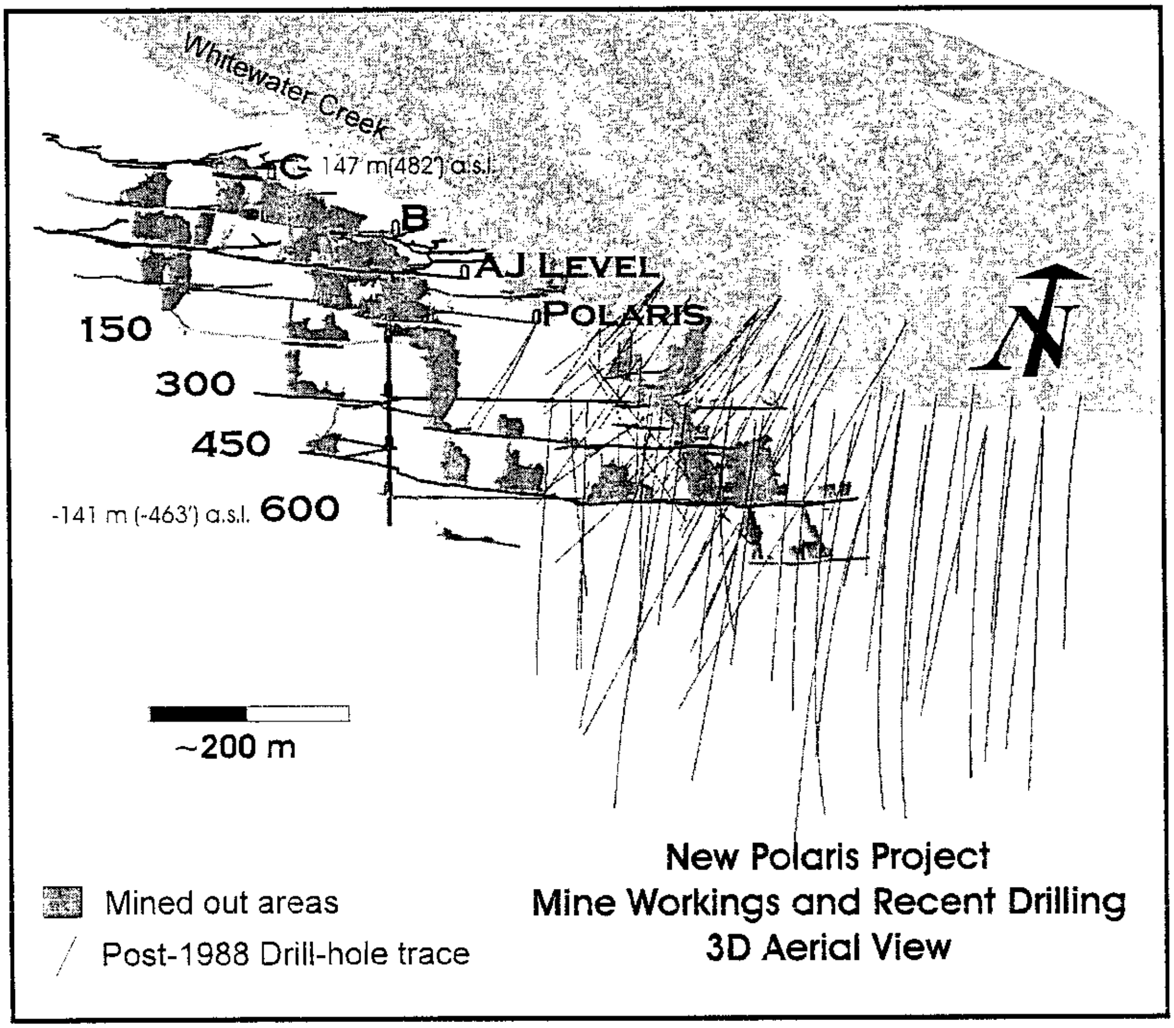


Figure 3.2

4.0 Exploration History

4.1 Exploration Results, 1988-1997

Recent exploration drilling consists of 143,992 feet in 202 holes drilled since 1988. Individual annual footages are provide in table 4.1. A general distribution of this drilling can be seen in figure 3.2. 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.

Diamond drilling from underground workings in 1996 was focused from the AJ level and targeted both the AB and Y vein systems.

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.

Table 4.1

SUMMARY OF EXPLORATION DRILLING TO 1997

Year	Zone #	# of Holes	Footage
1988	Y VEIN	8	3373
1989	Y VEIN	19	13378
1990	C VEIN	10	9391
1991	Y VEIN	4	4205
1991	C VEIN	7	6729
1992	Y VEIN	5	5262
1992	C VEIN	18	15662
1993	C VEIN	8	4270
1994	C VEIN	9	7729
1994	Y VEIN	5	5044
1994	NORTH ZONE	16	4403
1995	NORTH ZONE	14	11596
1995	C VEIN	6	13338
1996	Underground	24	10514
1997	Underground	49	29098
	Total	**202**	**143992**

4.2 Mineral Resources

A review of previous resource calculations has been made with the goal of identifying the probable order of magnitude of "reserves" that may be defined over time. While very little of these resources can be defined as proven mineable reserves at this time. There is sufficient available data to qualify them as probable and possible resources.

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 mineable 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.

Adtec Mining Consultants (1972) recalculated 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) recalculated the remaining "reserves" within the mine workings. These were defined to be in the order of 223,000 tons at 0.32 oz Au/SDT (diluted) based on a 0.15 oz/t cut-off and a minimum mining width of 4 feet. These reserves were subdivided into 151,000 tons of "assured" and 72,000 tons of "reasonably assured" reserves.

Beacon Hill recalculated 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. Au/SDT with 132,210 tons at 0.33 oz./t classed as probable and 112,210 tons at 0.32 classed as possible. Table 4.2 summarises their calculations. 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 Au/SDT (possible) which, added to their previously calculated reserves, brought the overall reserve potential up to 1,450,000 SDT @ 0.38 oz. Au/SDT (diluted) above the lowest worked level of the mine (600 level at elev. –462 feet Below Sea Level 'BSL').

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 oz./ton based on a geostatistical approach using a cut-off grade of 0.25 oz/ton. These reserves were divided into 333,000 tons @ 0.437 oz./t (probable) and 1,892,000 tons @ 0.432 oz./ton (possible). The calculation discounted much of the reserves around the old workings and did not include dilution and minimum mining width provisions. These calculations were based on both old and new drilling and extended the resource base down to roughly 1200 feet BSL.

Table 4.2

BEACON HILL RESERVES (1988) WITHIN MINE WORKINGS

PROBABLE RESOURCES					**POSSIBLE RESOURCES**			
Level	**In-Situ**		**Diluted**		**In-Situ**		**Diluted**	
	Tons (SDT)	**Grade** (oz /SDT)	**Tons** (SDT)	**Grade** (oz /SDT)	**Tons** (SDT)	**Grade** (oz /SDT	**Tons** (SDT)	**Grade** (oz/SDT)
	Above Polaris Adit							
Canyon	8,120	0.5	10,650	0.38	2,380	0.47	3,340	0.33
C	9,700	0.31	11,840	0.25	5,170	0.33	6,700	0.25
B	16,930	0.36	10,120	0.3	16,930	0.36	20,120	0.3
AJ	6,020	0.28	8,470	0.2	6,630	0.29	9,210	0.21
Polaris	12,670	0.37	16,720	0.28	10,450	0.36	14,080	0.27
Sub								
Total	53,440	0.37	67,800	0.29	41,560	0.35	53,450	0.27
	Below Polaris Adit							
150	310	0.52	570	0.28	400	0.52	740	0.28
300	19010	0.51	23830	0.4	14640	0.51	18870	0.39
450	120600	0.46	27080	0.35	18910	0.45	25080	0.34
600	10050	0.51	12930	0.4	11050	0.51	.14,070	0.4
Sub								
Total	50170	0.5	64410	0.39	45000	0.48	58760	0.37
TOTAL	*103,610*	*0.43*	*132,210*	*0.33*	*85,560*	*0.42*	*112,210*	*0.32*

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 calculated by Beacon Hill in 1989, into their overall estimate of a total (diluted) mineral resource of 1,600,000 tons at 0.46 oz. Au/SDT. Their calculations were based upon a minimum mining width of 5 feet or 15 % dilution and a cut-off grade of 0.25 oz/ton. The improvement in grade stems from the inclusion of new deeper holes that extend the known mineralzation to a depth of 1200 feet BSL and exclusion of lower grade material previously included in the Montgomery estimate.

Giroux was further contracted to provide reserve updates throughout 1992 and in February 1995 he recalculated 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 oz/t) has exhibited possible significant widths in the order of 22 feet. Giroux has included calculations for this zone, which for purposes of this review have been excluded due to grade. The results of his recalculation show that the "C" Vein discovered just prior to mine closure represents a significant new addition to the resource base. He has calculated a total of 85,700 tons grading 0.426 oz/ton (probable) and 595,000 tons grading 0.425 oz/ton (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 calculated by Giroux to date are summarised on Table 4.2. His calculations were in situ based on a 0.25 oz/ton cut-off and did not include dilution provisions as shown below.

In order to summarise the variety of RESOURCE CALCULATIONS identified above; the Beacon Hill calculation of residual reserves within and around the workings were totalled. To this total, the geostatistical resource calculations 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 SDT @ 0.359 oz. Au/SDT (Probable), and 2,614,210 SDT @ 0.363 oz. Au/SDT (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 overbreak during mining as well as overmining. The "AB" veins in-situ grade, as calculated by Giroux, already contains internal dilution from a parallel dike. To this total, an overall additional dilution of 15 % is considered appropriate. The "C" vein should not experience much dilution since it is generally thought to be fairly thick however it has been diluted 15% to allow for its relatively flat slope in places.

Table 4.3

Polaris Takus Geostatistical Resources								
	PROBABLE RESOURCES				POSSIBLE RESOURCES			
	In-Situ		Diluted		In-Situ		Diluted	
	Tons	Grade	Tons	Grade	Tons	Grade	Tons	Grade
Zone	(SDI)	(oz/SDT)	(SDI)	(oz/SDI)	(SDI)	(oz/SDT)	(SDI)	(oz/SDI)
GIROUX (1995)								
Y Zone	210,000	0.461	262,500	0.369	987,000	0.469	1,234,000	0.375
AB Zone	78,000	0.403	89,700	0.35	508,000	0.387	584,000	0.337
C Zone	85,700	0.426	98,500	0.37	595,000	0.425	684,000	0.37
Sub-total	373,000	0.441	450,700	0.365	2,090,000	0.437	2,502,000	0.365
BEACON HILL (1988)								
Upper Levels	53,440	0.37	67,800	0.29	41,560	0.35	53,450	0.27
Lower Levels	50,170	0.5	64,410	0.39	45,000	0.48	58,760	0.37
Sub-total	103.610	0.43	132,210	0.33	85,560	0.42	112.210	0.32
TOTAL	**476,610**	**0.439**	**582,910**	**0.359**	**2,175,560**	**0.436**	**2,614,210**	**0.363**

5.0 Metallurgical Testing

5.1 Sample Acquisition

Resource Development Inc. of Wheat Ridge Colorado received appoximately 150 kilograms of broken, unoxidized material from the AJ Level. Obtained with a rock hammer, chisel, and sledge hammer, The material was recovered from dry, unoxidised, 1.5 metre diameter boulders beneath a draw point identified on figure 5.1. Further care was taken to keep the material dry and it was immediately placed into 5 gallon metal pails and sealed tightly with a metal lid. These pails were then delivered to RDI's facility. Deepak Malhotra, President of RDI, managed the testing from this point forward.

5.2 Sample Preparation

The samples in each can were 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 2 kg samples and stored in plastic bags. A representative portion was further split, pulverized, split, and a protion of the pulverized material submitted fro chemical analyses at Florin Analytical Laboratory, Reno, Nevada. The results of this analysis are provided in table 6.1.

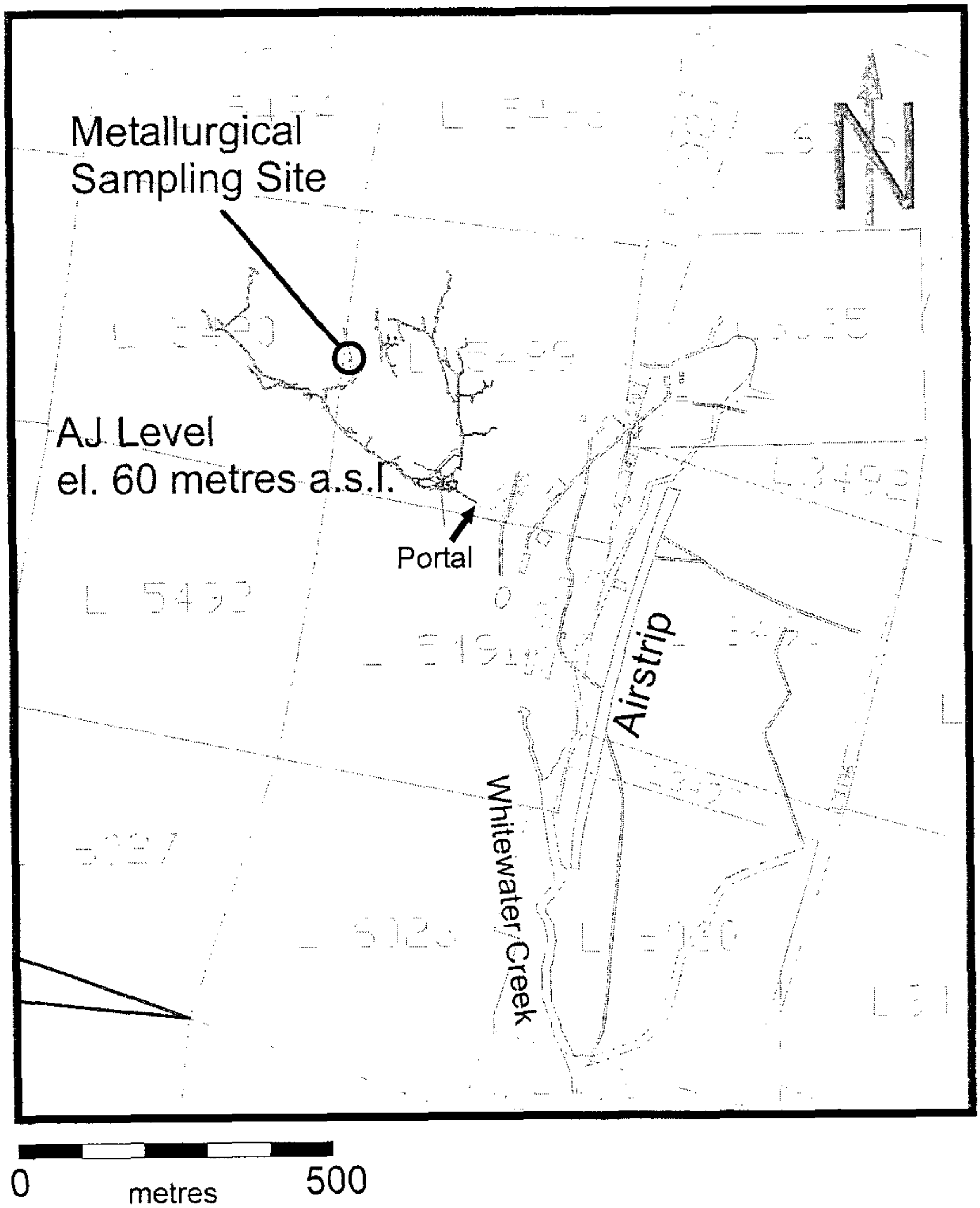
One kilogram of the sample was used in each test.

5.3 Leach testing

A one kilogram sample of material was wet-ground in a rod mill to 80% passing through 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 for 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 roasted in a muffle furnace at 425^{O}C for 4 hours, cooled, and a thief sample removed for assay. The dried residue was roasted in a muffle furnace at 625^{O}C for 4 hours, cooled, and a thief sample removed for assay. The roasted material was slurried to approximately 40% solids and the pH adjusted to approximately 11 and leached with sodium cyanide at a level of approximately 1.0g/l for 48 hours (Leach G). The residue from the fourth cyanide leaching was filtered, washed, re-filtered, dried and a thief sample removed for assay.

5.4 Flotation Test 1

The results from the first froth flotation test are provided in table 5.1.



Canarc Resource Corp
New Polaris Project
Metallurgical Sample Location Map

Figure 5.1

Table 5.1: Results from flotation test 1

Flotation Test 1:
Procedure

Operations	Conditions: min	Solids %	pH-Start	pH-End	Reagents, g/mt of flotation feed: PAX	MIBC
Rod mill grind	60	50		7.9		
Conditioning	1	26	8.0	8.1	100	20
Ro Flotation -1	5	26	8.1	8.3		
Conditioning	5	23	8.3	8.3	50	15
Ro Flotation -2	5	23	8.3	8.3		
Conditioning	1	22	8.3	8.3	50	15
Ro Flotation -3	5	22	8.3	8.2		
Total Reagent Used, g/mt of feed					200	50
Solution Concentration					1%	0.005

Results

Products	Weight gr	%	Chemical Analysis: Au g/mt	Ag g/mt	As %	Percent Distribution: Au	Ag	As
Feed (analyzed)	1000		19.48		2.84			
Feed (calculated)	1011	100.0	18.22		2.26	100.0		100.0
Ro Froth 1	127.9	12.7	83.78		10.35	58.2		58.1
Ro Froth 2	49.3	4.9	65.75		8.01	17.6		17.3
Ro Froth 3	31.9	3.2	45.11		5.78	7.8		8.1
Ro Tail 3	801.9	79.3	3.77		0.47	16.4		16.5

Observations: Grind in steel mill. Flotation in a 3.2 litre cell at 1600 rpm.

Rougher Tailing 1	883.1	87.35	8.72		1.08	41.8		41.9
Rougher Tailing 2	833.8	82.47	5.35		0.67	24.2		24.6
Ro Froth 1,2,3	209.1	20.70	73.63		9.10	83.6		83.5

5.5 Flotation Test 2

In the second test, Na_2S was added to the grinding mill. Results are provided in table 5.2.

Table 5.2: Results from flotation test 2

Flotation Test 2: Procedure

	Conditions				Reagents, g/mt of flotation feed			
Operations	min	Solids %	pH-Start	pH-End	PAX	$Na_2S.9H_2O$	Cu_2SO_4	MIBC
Rod mill grind	60	50				500		
Conditioning	1.0	26	8.6	8.8	100			20
Ro Flotation -1	5.0	26	8.8	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		250	10
Ro Flotation -3	5.0	23	9.0	8.4				
Total Reagent Used, g/mt of feed					180	500	250	45
Solution Concentration					1%	100%	5%	0.005

Results

Products	Weight		Chemical Analysis			Percent Distribution		
	gr	%	Au g/mt	Ag g/mt	As %	Au	Ag	As
Feed (analyzed)	1000		19.48		2.84	100.0	100.0	100.0
Feed (calculated)	1009.8	100.0	18.22	12.67	2.29			
Ro Froth 1	143.1	14.2	112.89	46.83	14.30	87.8	52.4	88.3
Ro Froth 2	18	1.8	32.92	33.90	3.84	3.2	4.8	3.0
Ro Froth 3	22.1	2.2	23.46	22.73	2.73	2.8	3.9	2.6
Ro Tail 3	826.6	81.9	1.37	6.02	0.17	6.2	38.9	6.1

Observations: Grind in steel mill. Flotation in a 3.2 litre cell at 1600 rpm.

Rougher Tailing 1	866.7	85.83	2.59	7.03	0.31	12.2	47.6	11.7
Rougher Tailing 2	848.7	84.05	1.95	6.46	0.24	9.0	42.8	8.7
Ro Froth 1,2,3	183.2	18.10	94.24	42.65	11.88	93.8	61.1	93.9

5.6 Flotation Test 3

In test 3 the conditioning pH was reduced with sulfuric acid, and copper sulfate was added to the third Condition/Flotation stage. Results are provided in table 5.3.

Table 5.3: Results from flotation test 3

Flotation Test 3: Procedure

Operations	Conditions: min	Solids %	pH-Start	pH-End	Reagents, g/mt of flotation feed: H_2SO_4	PAX	$Na_2S.9H_2O$	Cu_2SO_4	MIBC
Rod mill grind	60.0	50		8.2					
Conditioning	1.0	26	5.8	5.2	1853				
Conditioning	2.0	26	5.5	6.0		100			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	10
Ro Flotation -3	5.0	22	6.8						
Total Reagent Used, g/mt of feed					2316	180		250	55
Solution Concentration					97.5%	1%		5%	0.005

Results Products	Weight gr	%	Chemical Analysis: Au g/mt	Ag g/mt	As %	Percent Distribution: Au	Ag	As
Feed (analyzed)	1000.0		19.48		2.84			
Feed (calculated)	1005.4	100.0	18.84	16.41	2.45	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	3.09	15.57	0.43	13.0	74.9	13.9
Observations: Grind in steel mill. Flotation in a 3.2 litre cell at 1600 rpm.								
Rougher Tailing 1	856.1	85.15	5.47	14.98	0.72	247.0	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	21.00	78.08	19.59	10.05	87.0	25.1	86.1

5.7 Interpretation

As outlined in table 5.4, the standard flotation scheme recovered 20.7% of the weight, 83.6% of gold and 83.5% of arsenic in a rougher concentrate assaying 73.6% gpt Au. Sulfidizing prior to flotation improved both gold recovery and concentrate grade. The rougher concentrate recovered 18.1% of weight, 93.8% of gold and 93.9% of arsenic. The concentrate grade was 94.24 gpt Au.

Table 5.4 Summary of Flotation Test Results

Test No.	Process	Recovery %			Concentrate Grade, (grams per tonne Au)
		Wt.	Au	As	
1	PAX	20.7	83.6	83.5	73.63
2	PAX/MIBC/Na_2S/$CuSO_4$	18.1	93.8	93.9	94.24
3	Lower pH/ PAX/MIBC/ $CuSO_4$	21.0	87.0	86.1	78.08

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Watts, Griffis, and McOuat Ltd (1992): A Report on the Polaris-Taku Gold Project for Canarc Resource Corp., internal company report.

Section 7.0 Statement of Qualifications

I, James Gregory Moors, of 3375 Ontario St., Vancouver, British Columbia hereby certify that:

1. I am a geologist under the employ of Canarc Resource Corp., and have from November 1993 to April 1995, and from July 2002 till present been responsible for mineral exploration at the New Polaris Gold Mines site.
2. I am a graduate of University of Waterloo, with a B.Sc. Honours Earth Science degree (1989).
3. I am a Registered Professional Geoscientist with the Association of Professional Engineers and Geoscientists of British Columbia.
4. I have practised as a Geoscientist in British Columbia since 1991.
5. This report was prepared by me

James Gregory Moors

Dated this 23rd day January 2003.

VANCOUVER, BRITISH COLUMBIA

8.0 Itemised Statement of Costs

	units	Amount	rate	exchange	total
Labour					
James Moors	days	2	$222.00	1	$444.00
John Reed	days	2	$175.00	1	$350.00
Transportation					
Kluane Helicopters	hours	3.8	$1,649.80	1	*$6,269.24* (x26%) 1,649.80
Metallurgical testing					
RDI	US$		$3,500.00	1.549	$5,421.50
				TOTAL:	**$7,865.30**

KLUANE

CHARTER TICKET: KH 1104

HELICOPTERS A DIVISION OF 528470 ALBERTA LIMITED
P.O. BOX 2128, HAINES JUNCTION, YUKON TERRITORY, CANADA Y0B 1L0
TELEPHONE: (403) 634-2224 • FAX: (403) 634-2226

CHARTERER: CANARC

ADDRESS:

DATE:	A/C TYPE	A/C REG.	PURCHASE ORDER #	FORESTRY TICKET No.
Aug 23	AS350B2.	C-GKHS		

FLIGHT DESCRIPTION	TIME UP	TIME DOWN	HOURS	RATE	SUB-TOTAL
HAINES JUNCTION - ATLIN	1500	1600	1.0	1495	
ATLIN - TULSEQUAH.	900	9:50	0.8		
Tulsequah - ATLIN	1600	1635	0.6		
Aug 27th					
ATLIN - TULSEQUAH. - ATLIN	1500	1730	1.4		
Drop off 2persons					
Pickup. at later date -					
			3.8	1495	5681.00

CONTRACT No.	CONTRACT DAYS	MINIMUM HOURS	DAILY MINIMUMS	FUEL:	588.24
CHARTERER FUEL:	COMPANY FUEL:			OIL:	
DRUMS:	GALLONS $ /GAL.			MEALS:	
GALLONS	LITRES: 684 $ .86 /LTR.			LODGING:	
LITRES:					6269.24
				G.S.T. REG. 132709809	438.85
CHARTERER AUTHORIZATION:	PILOT:			TOTAL: $ 6708.09	

SIGNING OF THIS TICKET BY AUTHORIZED REPRESENTATIVES CONSTITUTES THE RIGHT BY THE CARRIER TO CHARGE 3% PER MONTH INTEREST ON ALL ACCOUNTS OVER 30 DAYS.

RESOURCE DEVELOPMENT INC.
11475 WEST I-70 FRONTAGE ROAD N.
WHEAT RIDGE, CO 80033
PHONE (303) 422-1176
FAX (303) 424-8580

Invoice

DATE	INVOICE #
12/17/2002	02-12-15

BILL TO:

Canarc Resource Corp.
Suite 800-850 West Hastings St.
Vancouver, B.C.
Canada V6C 1E1
Att: Mr. Bradford J. Cooke

DESCRIPTION	AMOUNT
Polaris Taku Project	
Sample Preparation	500.00
Diagnostic Leach	3,500.00
Laboratory Grind Studies	1,000.00
3 Flotation Tests	2,150.00
Shipment of Samples (see attachment)	99.27

Payment is due in 15 days
1.5% per month will be charged on past-due invoices

TOTAL $7,249.27