7.2 Central Vein

The Central and/or C2 Vein extend for about 1,800 m , and occur along the western contact between the Fe and Mg rich tholeiites, dipping west between 50° and 75°. The Central Vein is four or five metres wide at the hinge, pinches down to cm scale veins and swells to 40 m down-dip. Below this point, the shear zone that hosts the vein continues, but the vein itself has either attenuated or is pinched out completely. The C2 and Stringer Zones may possibly be the same structure, whereby the C2 Vein in the Doris North area grades into the Stringer Zone at Doris Central. The Stringer and C2 Zones both lie west of the main Lakeshore Vein.

7.3 Doris Hinge

The Central and Lakeshore Veins are the most important of the veins, and possibly represent the limbs of a shallow northerly plunging anticline. The strongest gold grades occur within the Central Vein and Hinge Zone at the crest of the anticline. The Hinge has been eroded south of the surface exposure, and has been tested to the north by diamond drilling over a strike extent of 325 m. Quartz veins hosted in the closure vary in thickness from one or two metres at the north to 12 m true width to the south, near surface. The quartz veins in the hinge area are dismembered into a series of wedges by axial planar shears and crosscutting brittle faults.

8. EXPLORATION

8.1 BHP Exploration

In 1991 BHP began a reconnaissance sampling program over available properties in the Hope Bay belt. Mapping and sampling continued in 1992 on the Boston claims and Madrid claims and the first surface mineralization at Madrid was discovered. The first drill holes were completed at Boston and additional claims were staked at Boston to protect the Fe-carbonate shear.

In 1993, BHP staked all available Crown mineral title in the belt. At this time about 60% of the belt was closed to staking by the Tunngavik Federation of Nunavut (TFN) during the final stage of their land claims settlement. Successful drill programs continued at Boston and the exploration focus shifted to locating another deposit elsewhere in the belt. The first drill holes at Doris and Madrid were completed the following spring in 1995 and fieldwork continued at evaluate other areas throughout the belt until 1998. Underground development and drilling occurred at Boston in 1996-1997. In 1998, BHP suspended their activities and put the Hope Bay project on care and maintenance.

Between 1992 and 1998, BHP conducted an aggressive gold exploration program in the Hope Bay Belt. To the end of 1998, the following activities had been completed:

- 177,350 m of core drilling;
- 2,341 m of underground development at Boston (1996-1997);
- 16,761 tonnes of bulk samples at Boston (1996-1997);
- 17,183 km of airborne geophysics;
- 1,500 km of ground geophysics;
- 1:10,000 scale geological mapping over the entire belt;
- 23,872 surface samples including regional till samples;
- 6,127 m of overburden drilling;
- Comprehensive environmental baseline studies;
- Detailed studies addressing access, infrastructure, metallurgy and engineering.

In total, BHP spent of the order of C$85 million defining gold mineralization in the Hope Bay Belt.

8.2 Geophysics and Landsat Imagery

The majority of the outcrop in the belt has been mapped and sampled. The greatest potential for new discovery lies under cover within the shallow overburden filled valleys. Numerous breaks, flexures, faults and folds have been interpreted from the magnetics images. The following is a brief summary of geophysical activities completed in the Hope Bay Volcanic Belt (HBVB) to date. Complete lists of geophysical surveys are contained in Tables 8.2(a) and 8.2(b).

BHP has also purchased LandSat TM, RadarSat, and Shadowed Topography images for delineation of belt scale features. The Shadowed Topography image comprises summer and winter LandSat data to delineate EW features, and improve depth perception. The HBJV recently purchased updated Landsat images for the belt.

TABLE 8.2(a)
Ground Geophysical Surveys in the Hope Bay Volcanic Belt

Year	Claim/NTI Group	Anomaly/Grid	Survey
2000-01 1998	Tok 3 PJ	Madrid GM	Magnetics/IP HLEM
1997	Boston	South Boston	HLEM/Magnetics
1997	Tok	Doris Corridor	Magnetics/VLF
1997	Tok, Boston	Doris Lake, Spyder Lake	Marine Seismic
1997	Tok/Amarok	Corridor Seismic	Refraction Seismic
1997	Boston NTI	Boston W.	Magnetics
1996	Akungani	Kamik	Magnetics
1996	Tok	Wizard	Magnetics
1996	Madrid	N. Patch	Magnetics
1996	Amarok	Amarok	Magnetics/VLF
1996	Tok	Wolverine	Magnetics/VLF
1995	Koig	Discovery Bay	Magnetics/IP
1995	Koig	Engine	Magnetics
1995	Boston	Boston 18/19	Magnetics/VLF
1995	Tok	Patch Lake	HLEM/Magnetics/VLF
1995	Koig	Dupras	Magnetics
1995	Tok	Doris	Magnetics
1994	Koig	Discovery	Magnetics/VLF
1994	Tok	Doris	Magnetics
1994	Tok	Jeffe	Magnetics
1994	Madrid	Madrid	Magnetics/VLF/IP
1994	Amarok	Kamik	Magnetics/VLF
1993	Boston	Wally/Redback/Huntsman /Fickle-duck	Magnetics/VLF
1993	Madrid	Madrid	Magnetics/VLF/IP

TABLE 8.2(b)
Airborne Surveys in the Hope Bay Volcanic Belt and Elu Inlet Volcanic Belt

Year	Company	Surveys	Extent
1993	Dighem	Magnetics/EM	3,129 line-km
1994	Dighem	Magnetics/EM	2,924 line km
1995	Dighem	Magnetics/EM	4,673 line km
1998	High Sense	Magnetics, Magnetics/EM	6,750 line-km

9. DRILLING

9.1 BHP Drilling

BHP Minerals Canada Ltd began diamond core drilling in the Hope Bay during the summer of 1992. A lightweight, thin-wall BQ core drill rig (Hydracore-28), operated by Shearcroft Mining Exploration Services Ltd was the first drill mobilized into the belt. A BBS-25A NQ core drill operated by Conners Drilling Ltd was brought in later in the 1992 season.

In 1993, JT Thomas Diamond Drilling Ltd was awarded the drilling contract and utilized in two Longyear-38 drill rigs drilling NQ sized core. JT Thomas continued as drill contractor through the 1998 season operating two rigs in 1994, three in 1995-96, five in 1997, and seven diamond drill rigs in the 1998 winter/spring season. Drilling of HQ size core was introduced in 1994 to aid in delineation at the Boston deposit (larger core provides a more representative Au concentration) and was subsequently used at the Doris and Madrid deposits. A Longyear-44 was commissioned in 1996 for deep drilling at the Doris deposit. A gopher diamond drill rig (ABDGM size core) supplied by Shearcroft Mining Exploration Services Ltd was utilized during the 1996-98 season for reconnaissance.

In total there have been 691 surface diamond drill holes (excluding 10 abandoned holes that drilled less than 10 metres) drilling 482 NQ-sized core, 144 HQ-sized core, 44 ABDGM-sized core, and 21 BQ-sized core holes.

9.2 2000 HBJV Winter Surface Drill Program

The 2000 winter drill program consisted of drilling in three different areas: Doris North, Doris Central and South Patch. A total of 23,114m were drilled in 141 holes with NQ2 drill rods. The program was completed with four drills, although for a short period of time five drills were operating. The program started with three Longyear 38's and one Longyear 44. Later in the program two Major 2000's were mobilized and the Longyear 44 was demobilized from the Hope Bay Belt. Table 9.2 lists the number of holes and metres drilled for each area.

TABLE 9.2
2000 HBJV Winter Drill Program Summary

Area	Metres Drilled	Holes
Doris North	12,053	91
Doris Central	6,430	27
Doris Central – Phase 2	3,626	17
South Patch	1,005	6
Total	**23,114**	**141**

9.2.1 Doris North (Hinge and Connector)

The Doris North area was the primary target for the 2000 winter drill program. The goal of the drill program was to increase the density of drill data in order to upgrade the Inferred Resource to a higher confidence Indicated Resource. The largest Inferred Resource at Doris North was identified by BHP in the Hinge Zone. The planned drill hole locations were designed to provide 25 by 15 m intercepts through the Hinge Zone. Drilling confirmed the interpretation of the Hinge Zone and confirmed its continuity over at least 300 m. The strike of the Hinge Zone may extend up to 550 m, although this must be confirmed with more drilling. Many of the best intersections were from the Central Vein in the Hinge Zone. While high-grade intersections were obtained from the Lakeshore Vein, the distribution was erratic and these intersections were less common. Visible gold in the quartz veins was quite common in the Central Vein and Hinge Zone, slightly less so in the Lakeshore Vein.

The proposed program was also designed to test the west dipping Central Vein and the sub-vertical Lakeshore Vein. A number of vertical holes were planned to test for sub-horizontal splays that may have been associated with the Lakeshore Vein. The drilling succeeded in delineating the Hinge Zone on a minimum of 25 m spaced pierce points along the majority of its strike length. Prior to 2000, the Connector Zone was not well drilled; a few widely spaced holes returned some good intersections. The goal of the 2000 drill program in this area was to expand the Inferred Resource in the Doris Underground and Connector Zones.

9.2.2 Doris Central

Doris Central drilling was designed to test the Lakeshore Vein on a 50 by 50 m pattern, with the goal of expanding the Inferred Resource. After the Stringer Zone was discovered during the Phase 1 drill program, a second phase of drilling was initiated. The purpose of the Phase 2 program was to improve the drill hole spacing to 50 by 25 m and to better define the Stringer Zone. Drilling during 2000 was generally on 25 m centres along strike, and 25 to 50 m centres down dip, with decreasing drill density away from the core, high-grade zone at Doris Central.

The Stringer Zone was discovered at Doris Central. Although, Stringer Zone mineralization was encountered in 1997 drill holes, the intersections were narrow and located in close proximity to the Lakeshore Vein. In some holes the Stringer style mineralization was incorrectly identified as the Lakeshore Vein. The dilational zone is a zone centred where the Stringer Zone intersects the Lakeshore Vein.

9.3 2000 HBJV Fall Drill Program

A much smaller-scale program was implemented in early fall 2000 to test two areas not investigated during the winter program: Madrid and Doris North Extension. Thirteen holes and 2,625 m were drilled using NQ2 drill rods. A Major 2000 drill tested conceptual/geological targets defined by the summer surface exploration program in the Madrid area. A Longyear 38 tested similar conceptual targets at Doris North Extension.

The goal of this program was to identify new gold mineralization that could significantly increase the total ounces in the northern half of the Hope Bay belt. Drilling at Doris North Extension targeted the underground projection of a vein network identified in the summer mapping program. This program had also identified high-grade gold values in sulphidic argillites. Results from this drilling failed to identify any significant mineralization and suggests that the Doris vein system either exists at depths exceeding those of the latest drilling or that the Doris vein system occurs further east from the last occurrence found on section 15800N.

10. SAMPLING METHOD AND APPROACH

10.1 Core Handling

The drilling contractor placed drill core in labeled boxes with marking blocks showing depth down the hole. The core boxes were delivered to the geology core logging areas at the Boston and Windy camps for geotechnical logging, geologic logging, core photography and sampling. The following procedure is rigorously complied with:

- Geotechnical technicians check that all footage blocks are in the proper order. (We work in metric, but the drillers work in feet).
- Geotechnical technicians fit the ends of the broken core back together without changing the "up-hole" direction.
- Geotechnical technicians convert the blocks to metres (drillers work in feet), and write the depths in metres on each block
- Geotechnical technicians measure the length of core (without gaps left by pushing the pieces together) between each set of blocks.

10.2 Geotechnical Logging

Core was laid out on a table in the boxes and measured to ensure blocking had been done properly. Downhole depths were indicated in feet on the core blocks, technicians converted depths to metric and marked the depth on the other side of the core blocks.

Geotechnical technicians record RQD (Rock Quality Designation) over the entire hole. RQD is a measure of the strength of each rock type, based on the density of natural fractures within it and is expressed as a percentage of the actual core lengths. In specific areas geotechnical logging was completed according to the Laubscher method of rock mass rating. This application was maintained to stay as consistent as possible with previous work completed by BHP. Cameron Clayton, a rock mechanics engineer with Golder and Associates Ltd., visited early in the program to observe the geotechnical logging procedures and provide technical assistance to HBJV geologic technicians.

Initially, all holes at Boston had geotechnical logging through mineralized zones plus 20 m uphole and downhole from the boundaries of the mineralization. As a result of the visit by Cameron Clayton, the amount of geotechnical logging was further reduced. The deposit was divided into 10 m thick vertical east-west slices, consistent with the mine geology cross sections, and detailed geotechnical logging was only done on holes drilled within every second 10 m thick slice. RQD and recovery were recorded for drill holes that did not require detailed geotechnical core logging.

At Windy, most of the holes were geotechnically logged for the entire length of the hole. Following Cameron Clayton's visit, certain holes were selected for geotechnical logging based on the distribution of holes that had already been logged. Also, only the mineralized and altered zones were logged plus 20 m intervals above and below the zones.

10.3 Core Photography

Digital photos were taken of all drill core and a camera mount has been constructed above each core logging table to ensure uniformity of photo. Procedures for photography are described below:

- Align the core boxes with marks on the table.
- Place a bar documenting the hole #, box numbers (from and to) and drill interval in metres (from and to) where it will appear at the top of the photo. This bar has a colour card attached to it.
- Make sure the colour card is clean.
- Wet the core before taking the photograph.
- All core photos are to be taken using the highest resolution possible with the digital camera.

10.4 Core Logging

Once the core has been geoteched it is ready to be logged. Logging will focus on the collection of geological and mineralogical parameters, which have a demonstrated relationship to gold mineralization. Log sheet and sample form templates are located in an Excel file named "2001_LOG_TEMPLATE.XLS". Each project (Doris, Boston, Madrid) has an individual log sheet within this file. The geotech and sample forms are the same for all projects. A drill hole summary template is located in Word document 2001_SUMMARY_TEMPLATE. Changes made to the templates on site must be approved by the Project Manager and then relayed to the Database Manager.

The geologist will fill in the log sheet using standard codes for lithology, alteration, veins and mineralization. The Project Manager must approve any new codes before they can be used, and this information conveyed to the Database Manager. For numerical values, ranges are not permitted. If required, a range may be indicated in a note. Notes may be written across all columns in the log, since there is no specific column dedicated to notes.

The Drill Hole Header Sheet should be filled out with collar coordinates, hole azimuth, dip, etc. For holes with acid tests, the information should be entered in the appropriate place. For holes that are surveyed by Maxibor, an indication that Maxibor was used should be included on the Header Sheet, and the Maxibor file should be e-mailed to Vancouver (data@skycomip.com).

Core logging and sampling methods were very similar to those previously employed by BHP except that logging by HBJV geologists was more quantitative than qualitative. For example, HBJV geologists made estimates of the percentages of quartz and pyrite, where previously BHP geologists had used terms such as "Strong", "Medium" and "Weak".

10.5 Core Sampling

Sampling was done using minimum sample lengths of 30 cm and maximum sample length of 1.0 m. Where possible, samples were defined by geological boundaries or characteristics such as sulphide abundance, alteration or vein intensity. All sampled core was sawn, and one half placed in labelled plastic sample bags. The other half of the core was returned to the core box and is kept at site.

10.6 Core Storage

At Boston, core boxes were placed on pallets, one drill hole per pallet, with the boxes in order, first box on the bottom and last box on top. The pallets were bound with steel strapping and stored on the muck pad. Initially the pallets were stacked rows, so that any hole from the 2000 program could be accessed quickly by fork-lift. Later the pallets were moved off the muck pad to provide additional room. Re-organization of the pallets will be required before the drill holes will be readily accessible again.

Core storage will be an ongoing problem at Boston, due to the limited space available on the muck pad and the high cost of building and maintaining core racks. Once the crusher and sampling tower are removed it might be possible to store core in the building that housed them, although this space is also valuable for storing other materials.

At Windy drill core was previously stored in "Coreland" located north of the Windy Camp. The area was filled with core from drill programs dating from 1994 though 1998. It was necessary to start a new Coreland for the 2000 drill core. The area chosen was located on the ridge above the camp. Initially, core was moved to the new Coreland with snowmobiles and skimmers, but this was an extremely time consuming process requiring up two or three snowmobiles and six to eight people. After the snow was melted, the core was bundled and flown by the helicopter to the new Coreland.

11. SAMPLE PREPARATION AND SECURITY

11.1 Sample Assays

Drill core from Hope Bay Project was analyzed at TSL Laboratories in Saskatoon, Saskatchewan. TSL is a recognized Canadian Laboratory that uses standard fire assay techniques. HBJV and Roscoe Postle personnel made a series of visits to TSL and reviewed analytical procedures.

The sample preparation used was to crush the entire sample to 70% passing 2 mm. From this a 1,000 g split was pulverized with a ring and puck mill to 95% passing 106 microns. The routine gold assay method was on a 29.2 g sample using fire assay with a gravimetric finish. Although BHP used an atomic absorption finish the upper limit of accuracy is only 10 g Au/t. Use of the gravimetric finish was found to give a much better comparison against the metallic screened results.

Samples observed to contain visible gold were sent directly for screened metallics assay. Initially samples that contained greater than 5 g Au/t on an initial fire assay were automatically re-assayed with a screened metallics assay. As more analysis became available the comparison between gravimetric and screen metallic indicated that metallic assays replicated screen metallics at values in excess of 20 g Au/t. In 2001 the threshold at which assays were submitted for screened metallics was increased to 10 g Au/t.

Assay results were received from TSL, first as digital Microsoft Excel files via e-mail, followed, several days later, by a signed hard-copy certificate via regular mail. These digital files were compiled into a Microsoft Access database. Separate tables within the database were maintained for fire assays and for metallics assays. The "final value" used in various reports and calculations, was the metallics assay when available, otherwise the value was an average of the fire assays.

When the signed certificates arrived, they were checked against the Access database tables. A second data integrity check, performed by another person, in which one out of five certificates, chosen randomly, was used to verify the digital data. No errors were detected. The only difference between the hard-copy certificates and the database is the substitution of a value of –0.03 to represent below-detection-limit assays, reported on paper as <0.03 g/t Au. This substitution is made to allow the assay fields within the database to remain as numerical values, rather than text

12. DATA CORROBORATION

Drill logs were written out by hand in the field. The hand-written sheets were faxed to the HBJV Vancouver office where a data-entry clerk keyed each drill log into an Access database. Separate database files were maintained for each drill hole, these were compiled into a master Access database after an initial round of verification. The data entry system implemented pick lists and validation rules in order to limit entries to codes that had been approved by the project managers. The Database Manager checked all sample numbers and sample intervals against the faxed copies of the hand-written sample sheets. The data tables from each drill log were exported into an Excel file. The Excel file was e-mailed to camp, proofread and corrected digitally by the logging geologist, and the corrected file e-mailed back to the HBJV Vancouver office. The verified data was imported back into the drill-hole-specific Access database and from there compiled into the master Access database.

Approximately once per week, the compilation of corrected drill-hole data was exported into an Excel file, e-mailed to camp and imported into Gemcom. Gemcom validations were run in order to detect further errors in the data such as overlapping or missing intervals. Corrections were made to the hole-specific Access databases and then the corrected data re-imported into the compiled Access database. This process was repeated until no errors were detected.

Several geologists experimented with logging directly onto laptop computers using Microsoft Excel. For these logs, the data was manipulated to maintain consistency with the Access database. Data was separated into tables on separate worksheets, codes added to link sub-tables such as alteration and veins to the appropriate lithological unit, notes shifted so that they were associated with the appropriate interval, and blank intervals deleted. Once manipulated, the data was imported into the Access database, and subjected to the validation procedure outlined above.

12.1 Quality Control/Quality Assurance of Assays

For every 20 samples submitted for assay, one blank and one standard were inserted into the sample stream. Blanks were either pieces of diabase dyke or samples of carbonate dyke rock. Pulverized standards were obtained from ALS Chemex and IME Laboratories. Table 12.1 identifies the estimated grade and one standard deviation for each of these standards.

TABLE 12.1
List of Standard used at Windy and Boston, 2001

Standard	Expected Value	Standard Deviation
IME00L	3.06	0.18
IME00H	7.83	0.38
IME01QX	3.21	0.34
IME01RX	7.56	0.39
IME01ZX	11.69	0.49
CH01SIN	1.97	0.21
CH01COS	5.85	0.37
CH01TAN	13.25	0.67

If the assayed value of a standard was greater than two standard deviations from the mean expected result for that particular standard, the HBJV requested the complete re-assay of an entire batch of samples.

These results are reported under an extensive quality control program supervised by Dean McDonald, P.Geo. Ph.D., Exploration Manager with Miramar Mining Corporation, who is an appropriately qualified person as defined by National Instrument 43-101. To further ensure the integrity of exploration results, the HBJV had Roscoe Postle & Associates independently audit quality control and quality assurance ("QA/QC") programs in place at the Hope Bay project. See News Release 00-06 dated April 11, 2000 for details on the program. This QA/QC program includes on site control of core samples and a program of duplicate, check, and blank assaying, including check assaying at a separate laboratory. Roscoe Postle found that the quality of these QA/QC programs exceeded industry standards. Dr. McDonald has corroborated the data, including sampling, analytical and test data, on which the above information is based.

13. ADJACENT PROPERTIES

The Hope Bay belt includes a number of other mineral deposits that are not addressed in this report: the balance of the Doris deposit (including Doris Connector and Doris Central, and all of the Madrid and Boston deposits.

Mineralization contained in these deposits is detailed in the mineral resource estimates discussed below and the geologic setting is outlined above. However, these deposits do not affect the results of thus Study as the development of the Doris Hinge Zone is addressed on a stand alone basis, and these adjacent deposits are therefore not material to the results of this Study.

14. MINERAL PROCESSING AND METALLURGICAL TESTING

14.1 Summary Of Metallurgical Testing

Laboratory studies were performed on a number of mineral samples obtained from the Doris Deposit, at the Hope Bay Gold Project. The samples were obtained from three areas, consisting of the Doris Central - Stringer Zone (CSZ), Doris Central – Lakeshore Vein (CLV), and Doris North.

The use of gravity recovery followed by cyanidation of gravity tailing or cyanidation of flotation concentrate provided for a promising treatment procedure;

- The gravity concentrate typically recovered 25% to 50% of the gold present, depending on the composite sample tested and grind conditions used.
- The primary grind can be relatively coarse (80% passing 149 microns), which will impact the size of the mill as well as the power required. Gold recovery improved at finer grind sizes and this may be justified pending economic evaluation.
- Overall gold recoveries using gravity and cyanidation ranged from 89% to 98.5% depending on the composite sample tested and conditions used.

- The gold recovery to the flotation rougher concentrate is good and only this concentrate needs to be reground prior to cyanidation, rather than the whole ore.
- The overall recovery of the gold by gravity and cyanidation of reground flotation concentrate ranged from 93% to 98%.

The test program indicates that the Doris material responds well to conventional gravity, flotation and cyanidation procedures for recovery of gold. For further details see Appendix E – Metallurgical Test Program by PRA.

14.2 Summary Process Description

The gold extraction process concentrates the incoming ore using gravity and sulphide flotation techniques. Only the concentrate is then intensively leached using cyanide. The gold in solution is electrically plated out from the cyanide solution and the resulting cathodes are then smelted to form gold dore bar. The cyanide in the treated concentrate and effluents is destroyed prior to discharge of the tailings to the tailings pond. The details of the process are as follows:

- Run of mine (ROM) ore is crushed in a two stage crushing circuit to a final product size of –13 mm. Crushed ore is stored on a stockpile and fed continuously to the processing plant by front-end loader.
- Crushed ore is ground in a closed circuit wet ball mill, with the entire mill discharge being fed to a continuous in-line pressure jig for recovery of free coarse and medium sized gold particles. By treating the entire mill discharge, recovery of nuggetty or flaky gold is maximised since it is recovered before it is broken up into finer particles or trapped in the circulating load. Mass of jig concentrate will be approximately 5% of the mass of new feed to the plant, and gold recovery is expected to be 60% of new feed. This concentrate is fed to an in-line leach reactor (ILR).

Jig tailings are classified in a cyclone cluster. Cyclone underflow is treated in a single flash flotation cell for recovery of remaining coarse gold, free fine gold and fast floating sulphides, principally pyrite. Mass of flash flotation concentrate will be approximately 4% of the mass of new feed to the plant, and gold recovery is expected to be 22% of new feed. This concentrate discharges to a concentrate regrind circuit. Flash flotation tailings are recycled to the ball mill.

Cyclone overflow at a size of 80%, -125 microns is treated in a scavenger flotation circuit consisting of four tank cells in series. Each cell is fitted with froth crowder rings to maximise recovery of lower grade froth expected in this part of the circuit. Mass of scavenger flotation concentrate will be approximately 2% of the mass of new feed to the plant, and gold recovery is expected to be approximately 15% of new feed. This concentrate discharges to a concentrate regrind circuit. Scavenger tailings gravity flow to the final tailings sump.

The concentrate regrind circuit treats 1.6 t/h of concentrate from the flash and scavenger flotation circuits. It consists of a vertical grinding mill in closed circuit with a classifying cyclone. Concentrate is ground to an 80% passing size of 45 microns. The purpose of this process step is to liberate more of the fine gold prior to leaching in the ILR.

Slurry from the regrind circuit cyclone overflow is combined with concentrate from the in-line pressure jig in the ILR feed cone, where it is dewatered to 75% solids w/w before gravity feeding into the ILR. In the ILR, gold is leached with spent electrolyte from electrowinning, supplemented by sodium cyanide solution. Sodium hydroxide is added in this step to control the pulp to a pH of 12. Gold recovery to solution is approximately 98%.

Slurry leaving the ILR is thickened in a settling cone. Clear overflow solution from the settling cone proceeds to gold electrowinning. Settling cone underflow at 60% solids content is filtered on a belt filter to reduce the moisture content to 15%. A three stage counter current wash ensures high recovery of cyanide soluble gold and free cyanide to the filtrate, with a washing efficiency in excess of 99%. Filtrate is pumped to electrowinning for recovery of gold.

Electrowinning is performed in two cells in series to maximise maximises gold recovery. Gold recovery over electrowinning is approximately 98%. Spent electrolyte from electrowinning is split roughly 60:40 between a recycle to ILR feed and a bleed to cyanide detoxification.

In electrowinning gold is plated onto steel wool cathodes. Loaded cathodes are removed periodically from the cells using a hoist and replaced with fresh cathodes. The loaded cathodes are washed and dried in a calcining oven where the steel wool of the cathodes is oxidised. The calcine is smelted in a barring furnace with a mixture of pre-weighed fluxes. Molten gold with other metallic impurities is poured into dore moulds, weighed and stored in the safe awaiting transport off site for refining.

Leach residue filter cake is repulped with electrowinning bleed solution and gravity launders into the cyanide destruction contactor, where it is contacted with Caro's Acid, a stoichiometric mix of sulphuric acid and hydrogen peroxide. Free cyanide and WAD cyanide complexes in solution are rapidly destroyed to a level of less than 5 ppm total cyanide. The discharge from this contacting tank flows into the final tails hopper, where it is combined with scavenger flotation tails to a final pulp density of 36% solids, and is pumped to a storage dam located some 3 km away. The two tailings pipelines (one duty/one standby) are heat traced to prevent freezing during the cold weather season. The tailings are considered environmentally acceptable, as cyanide contacted pulp that is to be diluted down by an order of magnitude by flotation tails slurry that has not been in contact with cyanide. Therefore, total cyanide in final tails should routinely be around 0.5 ppm, and never higher than 1 ppm, in accordance with current international guidelines. As the sulphides component in the tailings is low, and the tailings are to be stored under water cover, it is expected that there is minimal risk of sulphide degradation to acid products occurring. Provision has been made for emergency automatic diversion of tails pulp to plastic lined emergency ponds in the event of a pipe rupture further down the line or intermittent high cyanide levels in tails pulp leaving the plant.

Water supply for the plant is to be taken from a barren lake located 3.0 km from the proposed plant site. A high head borehole pump located in a decant well in the lake, accessed by a jetty or causeway, will allow all year round drawing of water from the lake as the intake to the pump will be located below any ice cap that forms. The water is pumped through a heat-traced pipe to the process water tank, with a takeoff to potable water treatment.

15. MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

15.1 Mineral Resource Estimates

The Hope Bay Mineral resource estimates for the Hope Bay belt have been updated to incorporate the results of all holes drilled during 2001 and are summarized below (Table 15.1). Further details, including a specific breakdown of measured and indicated mineral resources and inferred mineral resources for individual deposits, cut-off grades and other assumptions are outlined in the tables as set out in Appendix E.

TABLE 15.1

Summary of Hope Bay Project Mineral Resource Estimates to December 31, 2001

Catery/Deposit	Tonnes (000's)	Gold Grade (g/t)	Contained Gold (000's oz)
Measured & Indicated Resources			
Boston	1,386	15.4	687
Doris	887	21.5	614
Madrid	1,090	10.3	363
Sub-total Measured & Indicated Resources	**3,363**	**15.4**	**1,664**
Additional Inferred Resources*:			
Boston	2,574	10.9	901
Doris	1,679	15.0	811
Madrid	2,460	11.8	935
Sub-total Additional Inferred Resources*	**6,713**	**12.3**	**2,648**

*Inferred resources are in addition to measured and indicated resources.

All resource estimates have been prepared by the Hope Bay Joint Venture staff in accordance with Canadian regulatory requirements set out in National Instrument 43-101 and reviewed by Dean McDonald, P. Geo. Ph.D., Exploration Manager for Miramar Mining Corporation. Resource estimation models for the Boston, Doris (excluding the Doris Hinge and Doris Central zones) and Madrid (excluding Naartok and Suluk) were estimated utilizing a two dimensional polygonal approach. The Doris Hinge, Doris Central, Naartok and Suluk deposits were block modeled using ordinary kriging methods, whereas other zones applied inverse distance methods. Capping and cut off grades were applied as set out in the attached tables. Measured resources were estimated only in the Boston B2 Zone where the resource blocks have been undercut. Indicated resources for all the deposits generally lie within 25 m of a drill hole within detail drilled areas and inferred resources generally lie no more than 50 metres from a drill hole. Independent resource consultant Geostat Systems Inc. of Montreal has audited these estimates.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mineability, selectivity, mining loss and dilution. These mineral resource estimates include inferred mineral resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that these inferred resources will be converted to measured and indicated categories through further drilling, or into mineral reserves once economic considerations are applied

15.1 Scope of Mine Planning Work

The mine plan developed in this study is based on three of the Hope Bay deposits located between sections 15,200N and 15,800N:

- Doris Hinge resources
- Lakeshore Vein resources
- Central Vein resources

Resources of all classifications are considered in the mine plan including measured, indicated and inferred. Under a qualification that includes "inferred", it does not serve the study objectives to include too much detail in terms of development and stope layout.

The scope of the mine planning work includes the development of conceptual open pit and underground mining scenarios and scoping level cost estimates for the extraction of the economic portions of these deposits.

15.2 Mine Planning Procedure

The following procedure was used in the mine planning:

- Hope Bay Joint Venture prepared Gemcom block models for the three deposits
- Plans and sections were plotted through the deposits
- Cut-off grades were estimated
- Mineable shapes were hand drawn on the sections, outlining continuous areas above cut-off grade
- A minimum mining thickness of 2.2 m was used
- The mineable shapes were entered into Gemcom and constructed in to excavation solids
- The in-situ tonnes and grades in the solids were estimated
- The solids were checked and adjusted as necessary to avoid excessive dilution
- A thickness of 0.4 m was added to the hanging wall and footwall to simulate external dilution
- Mining recovery factors were applied, yielding the "study resources"
- Open pit mining was assessed using the Whittle Optimizer
- For underground mining, conceptual plans were made for access and mining method
- Production rates were estimated
- Information packages were provided to mining contractors to allow them to prepare estimates of costs and manpower, and to confirm production rates
- A project production schedule was developed

15.3 Context of Mine Planning Work

Based on work done by Hope Bay Joint Venture, the mineralization can be characterized as follows.

General:

- Mining quartz veins easily identified visually
- Hanging wall and footwall rocks are altered mafic volcanics
- Competent rock mass significantly enhanced by permafrost
- Overlain by strongly foliated rocks, structure parallel to strike
- No major faulting identified that would significantly displace mineralization

- There is potential for significant short range variability in geometry; strike, dip and thickness

Doris Hinge:

- Geometry is a plunging fold-nose, refer to typical cross section
- High gold grades
- Offsetting post mineralization faults, mainly oriented West to NW, occur about every 10 m within the Hinge, with off sets of about 1 m

Doris Central & Lakeshore Vein:

- Steep and narrow geometry
- Variable thickness and gold content in Lakeshore vein

15.4 Mining Options

Mining options are described in the following sections for Doris Hinge Open Pit, Doris Hinge Underground, and Lakeshore and Central Underground.

15.4.1 Doris Hinge Open Pit

The Doris Hinge open pit is designed to recover the resources of the high-grade Hinge Zone which is a north plunging fold nose lying close to surface, located just north of Doris Lake. Refer to Figure 15.4.1(a), "Mining Site Plan".

The Whittle program was used to create an optimized pit shell. Input costs and parameters are shown below in Table 15.4.1(a) (note that underground costs are higher than estimated in the underground section as they include an element for capital and higher uncertainty for underground operations).

TABLE 15.4.1(a)

Optimization Parameters for Doris Hinge Open Pit Design

Optimization Parameter	Selected Value
Open pit mining	CN $4.50/tonne
Extra for ore selectivity	CN $2.00/tonne
Underground mining	CN $60.00/tonne
Processing	CN $24.00/tonne
General & Administration	CN $25.00/tonne
Total open pit	CN $51.00/tonne
Total underground	CN $109.00/tonne
Overall pit wall slope	50°
Gold price	US $275.00/ounce
Exchange	0.65
Mill recovery	95%

Underground costs are included in the input parameters to allow the program to determine an economic changeover point to underground mining.

An open pit cut-off grade of 3.95 g/t Au has been used, based on the above inputs. Details are included in Appendix A, "Underground Mining Detailed Calculations".

For the purposes of this preliminary assessment, a pit design has not been completed, nor has the pit access ramp been designed into the pit walls. Tonnages of the study resources and waste rock have been based on the economic pit shell. Permafrost conditions will enhance wall stability. Refer to Figures 15.4.1(b), (c) and (d) showing a plan and sections of the Hinge Pit. The Hinge Pit data is given in Table 15.4.1(b).

TABLE 15.4.1(b)

Hinge Pit Data

Parameter	Value
Waste rock amount	375,000 tonnes
Study resources	82,500 tonnes at 12.1 g/t Au
Including: Dilution	10%
Including: Mining recovery	95%
Strip ratio	4.5
Total pit depth	40 m
Number of benches	8 x 5 m

Nuna Logistics prepared a cost estimate for open pit mining and the earthworks required for the project. The earthworks included an airstrip, roads and tailings impoundment wall. Nuna determined that the required waste tonnage for the earthworks was 486,000 tonnes, more than can be supplied from within the open pit shell.

For this reason it has been assumed that an additional 125,000 tonnes of waste will be excavated from the open pit. An unknown quantity of this extra stripping may be accounted for in the pit design. Incorporating a haulage ramp in the pit design would likely increase stripping requirements, but a design has not been completed.

Future work can be aimed at achieving a balance between waste requirements for earthworks, and the stripping requirements of the open pit.

The production rate has been estimated at 915 tonnes per day by Nuna. Split benching is envisaged for recovery of the study resources.

Equipment requirements have been specified by Nuna. One cat 988 loader and four Cat 773 50 tonne trucks will be capable of moving 190,000 tonnes per month. Waste will be hauled from the pit to construction areas. A Cat 235 excavator is planned to provide selective mining of ore grade equipment.

Nuna's complete equipment list and cost estimate are included in Appendix B, "Nuna Logistics Cost Estimate".

15.4.2 Doris Hinge Underground

The mining concept is to start from the bottom of the Doris Open Pit once it is completed, and to advance north at a minus grade following the plunge of the Doris Hinge study resources. Refer to Figure 15.4.1(a), "Mining Site Plan".

The study resources are mainly the high grade Doris Hinge zone from 15,300N to 15,800N. Cross sections were prepared at a 25 m spacing showing Doris Hinge, Lakeshore Vein, and Central Vein, as well as the block model grades for each of these zones.

Mining shapes were outlined on each section including some internal dilution. Irregular resource shapes were made more regular so they could be projected from section to section and made into a 3D solid in Gemcom. Maximum mining grades of –12 to –15% were respected in setting floor grades. The mining shapes were also controlled partly by the estimated cut-off grade of 7 g/t Au.

The Gemcom program estimated the resources listed in Table 15.4.2(a) within the 3D solid created from the mining shapes:

TABLE 15.4.2(a)
Estimated Resources For Underground Mining

Resource Area	Value
Central Vein	11,100 tonnes
Hinge	249,500 tonnes
Lakeshore Vein	15,200 tonnes
Waste rock at zero grade (21.1% internal dilution)	73,900 tonnes
Total resource	349,700 tonnes at 21 g/t Au

To estimate the study resources, factors were assessed for external dilution and mining recovery.

Some additional external dilution was modeled because the initial mining shapes assumed, on most sections, that mining would perfectly follow the hanging wall contact. This is not possible in practice. It is also known that there are offsetting faults about every 10 m along strike in this area. The offsets are fortunately small (<1m) but will add dilution.
To model the external dilution, the 3D mining solid was expanded 0.4 m on all sides. This effectively added 4.7% dilution (with a very low grade caused by the downward expansion of the sill into Lakeshore and Central) bringing total dilution to 25.8%. The external dilution percentage is low because the 0.4 m expansion is relatively small compared to the typical dimensions of the hinge mining shapes.

Mining recovery was estimated at 95%. Room and pillar mining is assumed, and this high recovery is based on achieving some pillar recovery on final retreat made possible by the good conditions (rock mass and permafrost) and relatively small spans.

The results of the above factors produce a study resource of 368,100 tonnes at 19.8 g/t Au. A small overlap with the open pit was corrected, resulting in the final study resource estimate of 358,600 tonnes at 19.9 g/t Au.

Generally, mine access is from the bottom of the Doris Hinge open pit. The face is in ore and can be advanced north. However, due to the steep initial plunge of this zone, this access must start near the sill at 15,300N and end up at the back (hanging wall contact) at 15,375N. Successive benches can be taken, never exceeding –15% (which for short distances is a conservative constraint). Refer to Figures 15.4.2(a), (b) and (c) showing a plan and sections of the Hinge Underground mining.

Mining cannot be completed in this zone, however, without a short waste access ramp from pit bottom (15,275N, 4910E, -17Elev) to section 15,350N at –39Elev. This ramp overcomes the problem of the steep initial plunge of the Hinge Zone.

The waste access ramp is sized at 5 x 5 m, driven at –15%. Drifting productivity is expected to be 1.8 m per 12 hr man-shift. Once set up, two crews of 2 miners can drive the ramp in 3 weeks.

The mining area is quite long in the strike direction at 500 m, and a secondary escape route is required. This will be provided by driving a ventilation/escape raise at section 15,650N, detailed in Table 15.4.2(b).

TABLE 15.4.2(b)
Raise Detail

Description	Value
Bottom	15650N, 5083E, –35 Elevation
Surface	15650N, 5100E, +11 Elevation
Length/dip	60 m at 50°
Dimensions	2.4 m x 2.4 m

The raise can be driven by open raising and a manway must be constructed within the raise.

If cost effective, additional ventilation raises can be driven to reduce the dependency on auxiliary fans and ducting. The approximate raise length to surface is 60 m, regardless of position along strike. One such raise is included in the cost estimate.

The mining method is room and pillar, modeled on the Nanisivik operational procedures. Good ground conditions are expected in permafrost. The mining depth is very shallow and no backfilling is planned for ground support.

The mining will be done with a development jumbo and this will consist of a combination of drifting, slashing and benching. This method will provide the flexibility to follow the changes in the geometry of the zone.
The stope will be opened up to vertical heights ranging from 17 to 22 meters. As benching progresses from top down, the back and shoulders will be pattern bolted and screened where necessary. Pattern bolting will be installed in the upper portions of the walls. Bolts are assumed to be 1.8 m by 46 mm split set bolts driven by the jumbo. It is assumed that the lower walls are not bolted because:

- They are within easy reach of maintenance scaling.
- Exposure time is less and there will be no loss of the surface permafrost.

Pillars have not been detailed but will be left as required to reduce spans on the advance. Pillars will be recovered during the final retreat.

Additional definition diamond drilling and test hole drilling with the jumbo will be done from within the stope as mining progresses. The additional data can be collected with no significant interference to mining.

Equipment will include a 2 or 3 boom jumbo, 6 yards scoop, 2 yard scoop (for narrow areas), a boom and basket configuration bolting jumbo, and a 23 tonne underground haulage truck. Ore will be hauled to a pit bottom stockpile for pick up by surface mining equipment.

Ventilation requirements are estimated at 90,000 cfm. Intake air will be down the ventilation/escape raise from surface. This direction is best since the natural tendency will be for the heated air from the diesels to flow up plunge along the back and out of the stope. This is a low pressure ventilation application, and the main fan should be 50 HP or less. The fan can be mounted on surface or underground (but here it will be more subject to blast damage).

Auxiliary ventilation within the northern end of the stope can be provided by a 48 inch, 100 HP fan connected to 48 inch ducting.

15.4.3 Lakeshore and Central Underground

The mining concept is to mine the higher-grade portions of the Central Vein and Lakeshore Vein that lie below (down dip) the Doris Hinge underground stope. Refer to Figure 15.4.1(a), “Mining Site Plan”.

Access ramps for the Lakeshore and Central mining will be driven from the Hinge stope. Pillar recovery in the Hinge stope can be started, but not completed until the Lakeshore and Central Vein access ramps are no longer needed.

Sections were prepared in the Doris Hinge area from 15,200N to 15,800N at a 25 m spacing showing Doris Hinge, Lakeshore Vein, and Central Vein, as well as the block model grades for each of these zones.

Mining shapes were outlined on cross sections considering an in-situ cut-off grade of 9 g/t Au. Four significant areas were identified above cut-off, referred to as shapes A, B, C and D. The approximate limits of these four areas are shown in Table 15.4.3(a).

TABLE 15.4.3(a)
Mining Shapes Data for Lakeshore and Central Veins (Section from 15,200N to 15,800N), using an in-situ cut-off grade of 9g/t Au

Number	Vein	Width (m)	Upper Elevation	Lower Elevation	Section (from)	Section (to)
A	Lakeshore	3 – 5	60 m	140 m	715N	770N
B	Central	2.2 – 5	50 m	70 m	400N	425N
C	Lakeshore	3.5	45 m	75 m	500N	n/a
D	Lakeshore	5 – 7	30 m	55 m	250N	275N

Gemcom 3D solids were created representing these areas. Refer to Figure 15.4.3, “Lakeshore & Central Mining Solids”. The in-situ mineral resources within these shapes are shown in Table 15.4.3(b) below.

TABLE 15.4.3(b)

Mining Shapes In-situ Resources (not expanded) for Lakeshore and Central Veins (Section from 15,200N to 15,800N), using an in-situ cut-off grade of 9g/t Au

Mining Shape	Vein	Diluted (kt)	Au (g/t)	Included Waste (kt)	% Dilution
A	Lakeshore	40.7	10.0	4.9	12%
B	Central	8.4	31.1	2.3	27%
C	Lakeshore	6.7	14.2	0.1	1%
D	Lakeshore	6.2	12.7	0.1	2%
Total		62.0	13.5	7.4	12%

To estimate the mineable portion of the in-situ resources, the following procedure was used:

- The shapes were expanded by 0.4 m to add external dilution to the average 12% internal dilution.
- Concepts were developed for mining methods and access ramps, drifts, cross cuts and raises. No drawings were created, just concepts and development estimates.
- Costs per tonne were estimated for access (waste development) and mining.
- All-in cut-off grades were estimated for each shape. Capital costs for waste development were included.
- The diluted grade of each shape was compared against its cut-off grade to determine if it were economically viable.
- The economic shapes were included in the mine plan. These were B, C and D.
- A mining recovery factor of 95% was applied.

The study resources are 21,500 tonnes at 19.4 g/t Au as shown in Table 15.4.3(c) below.

TABLE 15.4.3(c)
Lakeshore and Central Study Resources (Cut-off grade with capital and mineability assessment)

Shape	Diluted Resource		Total Cost* $/t	Diluted cut-off grade g/t Au	Study Resources		Notes
	Diluted ktonnes	Grade g/t Au			Recovered (95%) ktonnes	Grade g/t Au	
A	48.0	8.7	130	10.1	-	-	Not in plan
B	8.4	31.1	145	11.2	8.0	31.1	In plan
C	7.8	12.4	147	11.4	7.4	12.4	In plan
D	6.4	12.6	142	11.0	6.1	12.6	In plan
Total					21.5	19.4	

*** Total cost includes access plus $50 mining, $24 milling & $25 G & A.**

Generally, mine access is from the bottom of the mined out Doris Hinge stope. The final pillar recovery in the Hinge stope cannot be completed until mining is finished in the Lakeshore/Central areas. This affects the timing of the last 10% of the Hinge stope tonnage.

The waste access ramps and cross cuts can be driven during the mining of the Hinge stope. It may be possible to place some of the development waste in the mined out north end of the Hinge stope. Shapes B, C and D can be readied for production before the Hinge stope is mined out, thus ensuring steady production until all of the study resources have been extracted.

The mining method planned for the three mineable shapes is sublevel retreat. Each shape is quite small and will require only one or two sublevels in the zone, with uphole drilling to recover the resources overhead. Slots will be opened by inverse raises or open raises. Good ground conditions are expected in permafrost. The mining depth is very shallow and no backfilling is planned for ground support.

Equipment will include a 2 or 3 boom jumbo, 6 yards scoop, 2 yard scoop (for narrow areas), a longhole drill and a 26 tonne underground haulage truck. Ore will be hauled from the stope draw point up ramp to the Hinge stope, and from there to a pit bottom stockpile for pick up by surface mining equipment.

Ventilation will be achieved by auxiliary fans and ducting, or optionally by driving open raises from the stope lower levels up to the mined out portion of the Hinge stope.

15.4.4 Other Potential Mining Areas

Two additional Hope Bay deposits that could be incorporated into the mining schedule in the future are Doris North and Doris Central. These deposits are located under Doris Lake. Refer to Figure 15.4.1(a), "Mining Site Plan". Development of a production plan for these deposits is dependent on the success of the initial mining described in this report.

A conceptual plan indicates that these deposits can be accessed with no additional surface infrastructure. An access ramp would be driven south from the bottom of the Hinge Open Pit, continuing below Doris Lake, to reach these deposits. Study resources would be hauled underground by truck to the bottom of the Hinge Pit, and then handled in the same manner as the Hinge Underground production described above.

15.5 Manpower

15.5.1 Open Pit Mining

Nuna has estimated a crew of 38 men to complete the earthworks and open pit mining.

15.5.2 Underground Mining

Underground mining activity begins in month 15 (November 2004) when the shipping season allows the underground contractor to mobilize to site. It continues until month 36. Only modest fluctuations are expected in the underground work force during this time. Mining personnel on site during a typical month are shown in the table below. Note that there are an equal number of workers on "days off" at any given time due to the fly in/fly out schedule.

TABLE 15.5.2

Typical Underground Mining Manpower On Site

Group	Sub-Group	Number
Contractor	Diamond drilling	2
	Supervision	2
	Production miners	6
	Development miners	2
	Mechanics	4
	Nippers	2
	Surveyors	2
	Sub-total contractor	20
Owner	Engineering	4
	Geology	3
	Sub-total owner	7
Camp Man-days/month		713

Mine operations are planned on a continuous basis with two 12 hour shifts per day. The estimate of total man-days on site for underground mining personnel over the project duration is 13,900.

15.6 Production Execution and Schedule

The production schedule is shown in the table below.

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HOPE BAY
Doris North Trial Operation
Ore Production Schedule

											YEAR ONE						
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	Year 1
	Sep	Oct	Nov	Dec	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
MINED																	
monthly g/t						12.1	12.1	12.1							19.9	19.9	16.2
monthly tonnes						27,500	27,500	27,500	-	-	-				18,000	18,600	119,100
month average tpd						917	917	917	-	-	-				600	600	
					31	28	31	30	31	30	31	31	30	31	30	31	
										YEAR TWO							
					17	18	19	20	21	22	23	24	25	26	27	28	Year 2
					Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
MINED																	
monthly g/t					19.9	19.9	19.9	19.9	19.9	19.9	19.9	19.9	19.9	19.9	19.9	19.9	19.9
monthly tonnes					18,600	16,800	18,600	18,000	18,600	18,000	18,600	18,600	18,000	18,600	18,000	18,600	219,000
month average tpd					600	600	600	600	600	600	600	600	600	600	600	600	
										YEAR THREE							
					29	30	31	32	33	34	35	36	37	38	39	40	Year 3
					Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
MINED					from year 1												
monthly g/t					19.9	19.9	19.9	19.9	19.9	19.8	19.4	19.4					19.8
monthly tonnes					18,600	16,800	18,600	18,000	18,600	20,100	9,300	4,500	-	-	-	-	124,500
month average tpd					600	600	600	600	600	670	300	300	-	-	-	-	

The production schedule is based on continuous, year round mining operations, seven days per week, averaging 600 tonnes per day. The total study resources scheduled are 462,600 tonnes at an average grade of 18.5 g/t gold.

The schedule is designed around the constraints imposed by the northern location; the summer shipping season for mobilizing equipment and supplies to Roberts Bay by barge, and the availability of winter roads on site in the winter until the all weather road is constructed. Mining and milling activities extend over a period of 3 years.

The open pit is somewhat lower in grade at 12.1 g/t Au, but must be mined first because underground mining is dependent on a pit bottom access. Open pit waste stripping begins in month 5 and ore production begins in month 6. The open pit ore will be stockpiled since the mill is not scheduled to start until month 16. The open pit waste totals approximately 500,000 tonnes and all of it will be used in the construction of an airstrip, apron, roads and tailings containment dam.

Underground mining begins in month 15 with a portal being collared in a face of ore at pit bottom. The underground mining rate at 600 tpd, matches the processing rate. The underground ore, being of higher grade, will be processed first while the stockpiled open pit ore will be processed last.

The schedule has been simplified to show the Hinge production (19.9 g/t) coming completely ahead of Lakeshore/Central production (19.4 g/t). This simplification is possible because the underground grades are nearly the same. In practice the Lakeshore/Central stopes would be developed and mined starting a few months sooner than shown. This is because they only support a lower mining rate of about 300 tpd and should be blended with Hinge production. Also, from a planning perspective, there are no constraints on starting them earlier. This simplification will not have any significant impact on results.

15.7 Cost Estimate

15.7.1 Operating Costs

A summary of the operating costs are listed in Table 15.71.1.

TABLE 15.7.1
Operating Cost Summary

Operating Cost Breakdown	Unit Cost/tonne
Engineering and Geology	$ 3.44
Mining – Underground	$ 45.99
Open Pit Mining	$ 36.76
Processing	$ 36.83
General and Administration	$ 18.39
Safety, Security and Environmental	$ 2.41

15.7.1.1 Engineering and Geology

Engineering and Geology costs have been estimated by SRK Consulting with input from Miramar Mining Corporation. Costs within this area include mine engineering, owner mine supervision, mine geology, definition drilling and miscellaneous supplies. Definition drilling requirements shown below are based on discussions between SRK and Hope Bay Joint Venture geology personnel.

TABLE 15.7.1.1(a)
Definition Drilling Summary

Area	Meters	Costs
Doris Hinge	2,484	$ 141,905
Lakeshore & Central	1,235	$ 63,093
Totals	3,719	$ 204,998
Average unit costs	$55.12	$ 0.54/tonne

The drilling costs are based on a budget quote from Advanced Drilling. The basic coring charge is $42.25 per meter for BQ thinwall. Provisions are included for mobilization, some standby time and acid tests.

A Kubota tractor mounted Gopher drill is proposed to provide good mobility within the stope. The total meters of drilling shown will provide for coverage ranging from 10 x 10 m to 15 x 10 m in the mining areas.

The estimated costs for mine engineering and geology are presented in the table below.

TABLE 15.7.1.1(b)
Engineering and Geology Salaries

Staff	Number on payroll	Annual salaries	Payroll burden 34%	Cost per year
Mining engineer	2	$ 109,800	$ 36,870	$ 146,670
Geologist	3	$ 167,500	$ 56,190	$ 223,690
Mining Supervision	2	$ 166,270	$ 53,809	$ 220,079
Total				$ 590,439
Average cost per tonne				$ 2.69

Technical personnel on site are roughly half the number shown as "on payroll". One experienced mine engineer and one owner mine supervisor is required on site each day shift. Geology personnel must be available to the mine on both shifts to mark up faces and check muck piles.

In addition to drilling and technical staff, an allowance of $0.21 per tonne of ore mined has been included to cover technical staff field supplies etc.

The survey crew will be provided by the contractor, but will work closely with the owners technical staff.

15.7.1.2 Open pit mining

Open pit operating costs have been estimated by Nuna Logistics. SRK Consulting provided conceptual drawings and quantities of ore and waste on each bench, and Nuna estimated the operating costs for the Hinge open pit.

Nuna's cost estimate did not specifically separate earthworks costs from open pit operating costs, but discussions with Nuna's estimator indicate that $4.50 per tonne of material mined is a good estimate of the open pit portion of their project costs. Per tonne operating costs for ore have been increased by $1.50 to account for grade control activities. Based on the pit plan designed a total of 663,000 tonnes must be mined of which 82,500 tonnes is ore to be processed.

15.7.1.3 Underground mining

Underground operating costs are discussed in this section. Underground operating costs have been estimated by SRK Consulting with some input from Procon Mining and Hope Bay Joint Venture personnel (Procon reviewed conceptual drawings and quantities for the underground mining). Cost estimates are based on the tonnages and mining methods shown in the table below.

TABLE 15.7.1.3(a)
Underground Mining Tonnages

Area	Method	Percentage	Tonnage
Hinge Underground	Drifting	20%	71,720
	Slashing	30%	107,580
	Benching	50%	179,300
	Sub-total	100%	358,600
Lakeshore & Central Veins	Sublevel retreat	100%	21,500
TOTAL underground			380,100

Underground operating costs are estimated at $45.99 per tonne of ore as shown in the table below.

TABLE 15.7.1.3(b)
Underground Mining Costs

Mining Cost Breakdown	Unit Cost/tonne
Extraction	$ 39.11
Ground support	$ 2.42
Underground trucking	$ 1.73
Surface services	$ 2.73
Total mining	$ 45.99

Extraction and ground support operating costs have been estimated for Doris Hinge mining and these costs have been applied to mining of Lakeshore and Central vein areas even though the mining method will be different. A sublevel retreat method of open stoping will be used in the latter areas as differentiated from the benching method used in the Doris Hinge stope. This simplification is justified because of the level of study, the inferred category of some of the resources, and the small tonnage being mined in the Lakeshore and Central area, of less than 6% of the total.

The Hinge extraction cost includes drilling, blasting, mucking to a remuck bay or underground truck, installing services and nipping in supplies. It is an "all in" cost including supervision, mining, maintenance and surveying (Ground support is excluded and estimated separately). It is based on a contractor's drifting rate converted into a cost per tonne as shown below.

TABLE 15.7.1.3(c)
Extraction Unit Cost (unit costs are based on a contractors drifting cost)

Component	Value
Drift width	5.0 m
Drift height	5.0 m
Drift ore density	2.7 t/m3
Total drift tonnage per meter	67.5 tonnes/m
Contractor cost per meter (all in)	$ 3,000
Drifting cost per tonne	$ 44.44
Slashing cost per tonne @ 75%	$ 33.33
Benching cost per tonne @ 75%	$ 33.33

Slashing and benching costs are only 75% of the drifting cost due to less drilling, less explosives, better breakage and faster cycle. The above shown unit costs for extraction have been applied to the respective Hinge tonnages as shown below. A cost allowance has been added for some cost plus work or standby charges.

TABLE 15.7.1.3(d)
Total Mining Cost for Hinge Extraction

Method	Hinge u/g tonnes	Unit cost	Cost
Drifting	$ 71,720	$ 44.44	$ 3,187,556
Slashing	$ 107,580	$ 33.33	$ 3,586,000
Benching	$ 179,300	$ 33.33	$ 5,976,667
Sub-total	$ 358,600		$ 12,750,000
Add: Cost plus/stand by @ 10%			$ 1,275,000
Total			$ 14,025,000
Average cost per tonne			$39.11

The Hinge unit extraction cost is estimated at $39.11 per tonne. As discussed previously, this cost has also been applied to Lakeshore and Central vein mining.

Ground support costs are based on screening the back and shoulders of the Hinge stope, and pattern bolting (1.5 x 1.5 m) the upper portion of the walls. This is required due to the "man entry" mining method and the increasing height of the stope back as benching progresses down.

Ground control supplies are assumed to include 1.8 m split set bolts, sheets of weld mesh screen, and 0.6 m mechanical bolts to help pin the screen. The contractor's cost per installed split set bolt is $45 each and screening costs $22 per square meter. A cost summary is shown below.

TABLE 15.7.1.3(e)
Hinge Ground Support Summary

Location	Area	Screen	No. bolts	Cost
Back	12,550 m^2	Yes	8,715	$ 668,288
Walls	10,000 m^2	No	4,444	$ 200,000
Total				$ 868,288
Average cost per tonne				$ 2.42

Underground trucking costs have been estimated for the Hinge stope and the stoping planned for Lakeshore and Central veins. A 26 ton truck is assumed since there is currently one of these units at the Boston site. Haulage cycle times have been estimated from the stopes to the pit bottom where a transfer stockpile will be maintained.

The truck operating cost is estimated at $76 per hour exclusive of operator. A trucking summary is shown below.

TABLE 15.7.1.3(f)
Underground Trucking Summary

Area	Tonnes	One way distance (m)	Cycle time (min)	Cost per tonne
Doris Hinge	358,600	440	15.6	$ 1.71
Lakeshore & Central	21,500	465	17.8	$ 1.96
Total	380,100	-	-	$ 1.73

Costs have been based on trucking all of the ore tonnes. Some of the ore tonnes located near the pit bottom will likely be trammed by scoop with no requirements for trucking. It is assumed that these tonnes will be offset by the development waste tonnes that will likely require trucking. Development waste tonnage at 27,800 tonnes represents only 7% of total material to be moved.

Surface services, shown in the table below, amount to $2.73 per tonne of ore and include mine power and surface waste handling.

TABLE 15.7.1.3(g)

Surface Services Summary

Component	Cost/tonne
Underground Services	$ 0.27
Electricity	$ 2.46
Total	$ 2.73

Mine electrical power costs represent power for the main ventilation fans, some heating, pumps and the surface maintenance shop.

15.7.1.4 Processing Costs

Process operating costs are summarized in this section. Detailed discussion of these costs are contained within Bateman's report as referenced in Appendix C.

TABLE 15.7.1.4

Processing Cost Summary

Operating Cost Breakdown	Unit Cost/tonne
Labour	$ 9.12
Reagents and Consumables	$ 6.67
Power	$ 10.76
Maintenance and Mobile Equipment	$ 9.03
Assay Lab	$ 1.09
Miscellaneous	$ 0.16
Total	$ 36.83

15.7.1.5 General and Administration Costs

General and Administration costs have been estimated by the Hope Bay Joint Venture and have been verified by SRK Consulting. Costs within this area include plant buildings, general support including accounting, IS, purchasing and warehousing, camp costs, employee and mine contactor transportation, outside refining and miscellaneous.

TABLE 15.7.1.5
General and Administration Cost Summary

Operating Cost Breakdown	Unit Cost/tonne
Plant Buildings	$ 0.82
General Support	$ 3.86
Camp Costs	$ 3.82
Employee Transportation	$ 5.46
Outside Refining	$ 2.02
Miscellaneous	$ 2.41
Total	$ 18.39

Plant Buildings includes supplemental plant heating (main plant heating source is heat exchangers on the diesel generators) plus an allowance for general repairs.

General support is primarily related to labour costs associated with accounting services, purchasing and warehousing, surface labour, and general clerical functions. The total number of personnel in this category are 13.

Camp costs are derived from a per diem charge of $42 (estimate provided by Nuna Logistics) applied to an annualized number of camp man days of 19,893. This includes all owner employees, mining contractor employees plus visitors.

Employee transportation is based on an average airfare cost of $975 per employee per month. This is based upon a mix of employee travel costs to Yellowknife and Cambridge Bay. Employees include both owner and mining contractor personnel.

Outside refining costs are based upon actual refining and transportation costs per ounce as experienced by Miramar's Con Mine.

Miscellaneous costs include property insurance, communications and miscellaneous supplies.

15.7.1.6 Safety, Security and Environmental

These costs have been estimated by the Hope Bay Joint Venture and have been verified by SRK Consulting. Costs within this area include labour and supplies for onsite safety services and environmental monitoring and compliance requirements. Safety services will be provided by two full time camp nurses supplemented by trained safety personnel within the mine and mill operating groups. Environmental services are performed by one senior environmental coordinator and two technicians. Costs including labour and miscellaneous supplies have been estimated at $2.41 per tonne.

15.7.2 Capital Costs

15.7.2.1 Site Earthworks and Infrastructure

Site Earthworks and Infrastructure costs shown below have been estimated by Nuna Logistics with assistance from SRK Consulting. SRK Consulting developed plans for the site layout including roads, airstrip, tailings dam etc. and provided quantities to Nuna Logistics for cost development. Mobilization costs have been estimated based on a total of 3,300 tonnes of freight (in and out) barged between Hay River, NT and site. Per tonne barge rates have been estimated at $648 per tonne (NTCL). Contractor standby charges of $45,000 per month have been included during the period equipment remains idle awaiting demobilization.

TABLE 15.7.2.1
Site Earthworks and Infrastructure

Capital Costs	Cost
Mobilization and Demob	$ 2,138,000
Contractor Standby Charges	$ 315,000
Earthworks (lump sum)	$ 2,231,000
Sub-total	$ 4,684,000

15.7.2.2 Underground mining

Underground capital costs shown below have been estimated by SRK Consulting with some review by Procon Mining.

TABLE 15.7.2.2
Underground Mining Costs

Capital Costs	Cost
Mobilization	$ 308,000
Portal	$ 40,000
Mine access (waste development	$ 1,719,000
Sub-total	$ 2,067,000

The mobilization cost is based on Procon Mining performing the work since they have a small fleet of mining equipment at the Boston site. The allowance for mobilization which includes mining supplies, additional equipment and manpower has been discussed with Procon since they are experienced in moving men and equipment to the site.

The portal costs reflect one pit bottom portal to be collared in the bottom of the open pit.

It is assumed that all waste development will be capitalized since it all represents main underground access to support production. None of this waste development is required before the start of production. It can be driven concurrently with ore production and the mining cost schedule reflects this.

15.7.2.3 Process Plant

Process Plant capital costs shown below have been estimated by Bateman Engineering. Details of the estimate are contained within the Bateman study referenced. Costs include all plant equipment, engineering, construction and support costs.

TABLE 15.7.2.3
Process Plant Costs

Civils	270,375
Mechanical Equipment including conveyors & tanks	3,373,918
Platework & Liners included in 002	-
Structural Steelwork	455,931
Piping & Valves excl tailings & overland pipes	468,750
buildings, container office - allowance	103,841
Electrical including PLC, MCC	640,462
Instrumentation	586,737
Tails Lines and Water Supply	873,266
Desom Ventilation	403,350
Mech Eqt, Plw, elec, struct, & Instrum. Erect mobilise	16,667
First fill - allowance	96,364
Electrical mobilise	16,276
Instrumentation mobilise	4,167
Trial assemble, Strip plant and load into containers	83,333
Transportation - CIF to Hope Bay - 15% of FOB value	1,006,856
Commissioning Spares - allowance 1% of mech, elect, 67% inst.	40,256
Commissioning labour	44,258
Vendor assist during Constr & Comm-Allow 1%	36,344
12 month's Op Spares - allow 4% of mech & elect, 67% inst.	161,023
DIRECT FIELD COSTS	**8,687,290**
Sprung Steel Dome	1,913,771
Transportation to Hope Bay	43,906
TOTAL DIRECT FIELD COSTS	**10,644,967**
Home Office & Indirect Field Costs	2,233,391
TOTAL NET COST	**12,878,358**
Fee - 10% of Total DFC	1,064,497
Fee - "Technical know-how" 15% of Tot HO costs	335,009
Insurance - CAR & marine - allowance	133,333
Contingency @ 12% of TNC	1,545,403
TOTAL OTHER COSTS	**3,078,242**
OVERALL PROJECT COST	**15,956,599**

15.7.2.4 Other

Other costs have been developed by the Hope Bay Joint Venture and have been verified by SRK Consulting.

TABLE 15.7.2.4
Other Costs

Capital Costs	Cost
Mobile and Miscellaneous Equipment	$ 750,000
Systems and Communication	$ 1,000,000
Start up Labour	$ 1,238,000
Camp Costs	$ 175,000
Fuel Farm	$ 562,000
Miscellaneous	$ 250,000
Sub-total	$ 3,975,000

Mobile and Miscellaneous equipment is intended for light vehicles, a fuel truck, miscellaneous supply trucks and small shop tools

Systems and Communications covers site Information systems, phones, mine planning systems and associated hardware. Costs are based on quotes received by Con operations and recent Miramar experience.

Start up labour has been estimated as two full months of operating labour and support for the purposes of hiring and training.

Camp costs are the estimated costs to move the Boston man camp to the Doris site (60 km) and to upgrade the kitchen facility.

Fuel farm costs are lease costs for a six million liter tank farm for the project duration.

Miscellaneous costs primarily cover site geotechnical drilling, tailings dam engineering and other small engineering studies.

15.8 Economics

A detailed economic analysis was completed based upon the developed study resources, capital and operating costs as discussed in the sections above.

This analysis assumes the following schedule of events

- Ongoing feasibility and permitting activities support a Q3 2003 production decision
- Open Pit and site earthworks contractor mobilizes equipment and materials in Q3 2003
- Actual site construction and open pit mining begins in Q1 2004
- Process plant construction begins in late Q1 2004 with delivery to the site in Q3 2004
- Underground mining and process plant operations begin in Q4 2004

Table 15.8.1 below summarizes the economics of the Doris North Trial Operation. The detailed model is presented in Appendix E, along with sensitivity tables for gold price and $US:$C exchange, grade and recovery, and capital and operating costs.

15.8.1 Key Assumptions for the Economic Model

The following parameters have been used in the development of the economic model:

- All $ are stated in $Canadian unless otherwise specified
- Base gold price has been set at $US 280 per ounce
- Exchange Rate has been set at 1.570 $US:$C
- Capital Costs do not include feasibility or permitting costs
- All open pit costs are treated as an operating expense. Although costs are incurred prior to mill start up they are deferred against the ore stockpile created and charged to expense as the stockpiled ore is processes.
- Royalties payable are set at the minimum 1.8% of gross gold revenues

15.8.2 Bonding and Closure Assumptions for the Economic Model

Bonding and closure costs were developed by HBJV and reviewed by SRK Consulting. Although the actual bonding requirement will be established as part of the permitting process this study assumes a bond of $5 million will be required prior to the commencement of operations. It is expected that as part of the permitting process both the Boston and Windy campsites will be reclaimed. The rationale for this is:

- The camp, fuel, fuel farm, and miscellaneous equipment at Boston are to be used at the Doris North plant site
- There is 8,933 tonnes of 17.7 g/t ore stockpiled at Boston that will be processed in the mill
- All belt exploration will be conducted from the new operating site (Windy camp is redundant)
- Cleanup of these sites will release approximately $3.9 million in existing securities associated with disturbance at Boston
- Costs associated with the closure of Boston and Doris campsites has been estimated at $650,000. This includes ore haulage to the Doris North site, waste haul underground at Boston, garbage haul to the Doris North site, camp and tank farm tear down and haulage to the Doris North site.

As this study deals only with the mining and processing of Doris North resources, project closure costs have been developed. These costs would be incurred in the year following cessation of operations. Factors included include:

- Employee severance, as per Nunavut Labour Standards
- Scarifying roads, airstrip etc. and revegetation of disturbed areas
- Doris North underground closure
- Building teardown and removal

Closure costs have been estimated at $2.9 million. Offsetting this, a salvage value of $3.2 million is included.

TABLE 15.8.1
Summary Results

HOPE BAY PROJECT - DORIS NORTH								
SUMMARY RESULTS:			*2003*	*2004*	*2005*	*2006*	*2007*	*TOTAL*
Cash Operating Cost per Ounce	**$Cdn**		**$ -**	**$ 212**	**$ 197**	**$ 154**	**$ -**	**$ 179**
	$US		**$ -**	**$ 135**	**$ 126**	**$ 98**	**$ -**	**$ 114**
All-in Cash Operating Cost per Oz	**$Cdn**		**$ -**	**$ 1,218**	**$ 203**	**$ 142**	**$ -**	**$ 277**
	$US		**$ -**	**$ 776**	**$ 129**	**$ 91**	**$ -**	**$ 177**
Direct Operating Cost Per Tonne	**$Cdn**		**$ -**	**$ 67**	**$ 107**	**$ 139**	**$ -**	**$ 105**
	$US		**$ -**	**$ 43**	**$ 68**	**$ 88**	**$ -**	**$ 67**
MINE PRODUCTION								
Doris Hinge - North	Tonnes		-	119,100	219,000	124,500	-	462600
Ore Milled	Tonnes		-	55,200	219,000	197,400	-	471600
Grade	gpt		-	14.00	18.46	19.74	-	18.48
Recovery	%		0.0%	97.0%	97.0%	97.0%	0.0%	97.0%
Total Gold Recovered	**Oz**		**-**	**24,103**	**126,105**	**121,516**	**-**	**271,724**
Market Prices:								
Gold (Spot) $US		$280	280.00	280.00	280.00	280.00	280.00	280.00
Exchange Rate $US:$C		1.570	0.637	0.637	0.637	0.637	0.637	0.64
Sales Revenues:								
Gold (Spot)	$Cdn		-	10,595,788	55,435,580	53,418,615	-	119,449,983
Total Sales Revenue:	$Cdn		-	10,595,788	55,435,580	53,418,615	-	119,449,983
Cost of Goods Sold								
Mine Operations	$Cdn		-	4,916,347	10,826,895	6,225,799	-	21,969,040
Processing Operations			-	2,017,103	8,066,554	7,294,367	-	17,378,024
Surface Maintenance			-	45,100	180,400	165,367	-	390,867
Administration			-	970,186	4,374,014	3,612,763	-	8,956,962
Royalties		1.8%	-	190,724	997,840	961,535	-	2,150,100
Other / Amortization			313,731	4,071,945	11,639,647	11,476,646	(833,322)	26,668,646
Total Property Costs			313,731	12,211,405	36,085,350	29,736,475	(833,322)	77,513,639
Net Income	$Cdn		(313,731)	(1,615,617)	19,350,230	23,682,140	833,322	41,936,344
CASH FLOW								
Net Income	$Cdn		(313,731)	(1,615,617)	19,350,230	23,682,140	833,322	41,936,344
Non-cash			313,731	3,421,945	11,639,647	11,309,967	-	26,685,290
PP&E Capital			(3,119,000)	(21,400,590)	(2,165,700)	-	-	(26,685,290)
Reclamation/Cash Bonds			-	(1,150,000)	-	-	4,000,000	2,850,000
NET CASH FLOW	**$Cdn**		**(3,119,000)**	**(20,744,262)**	**28,824,176**	**34,992,107**	**4,833,322**	**44,786,344**
CUMULATIVE NET CASH	$Cdn		(3,119,000)	(23,863,262)	4,960,914	39,953,022	44,786,344	
NET PRESENT VALUE		0.4%	(3,004,304)	(19,439,200)	25,657,307	30,114,096	3,886,802	37,214,701
CUMULATIVE NPV		5.0%	(3,004,304)	(22,443,503)	3,213,803	33,327,899	37,214,701	
IRR		85.2%						

16. OTHER RELEVANT DATA AND INFORMATION

Crown mineral claims are subject to standard royalties under the provisions of the Canadian Mining Regulations ("CMR"). For each fiscal year, royalties shall be paid to the Crown by the owner or operator of every mine on Crown lands on the value of the output of the mine during that fiscal year, in an amount equal to the lesser of:

- 13% of the value of the output of the mine; and
- the amount calculated in accordance a sliding scale of 5% for value output of less than $5 million increasing to 14% for value output in excess of $45 million.

The value of the output is net of allowable deductions, including the following:the costs, incurred during the fiscal year, of sorting, valuing, marketing and selling the minerals or mineral-bearing substances produced from the mine;

- the costs, incurred during the fiscal year, of insurance, storage, handling and transportation to the smelter, treatment plant or refinery or to market of, and any duties payable in respect of, the minerals or mineral-bearing substances produced from the mine;
- the costs, incurred during the fiscal year, of mining and processing ore or mineral-bearing substances from the mine or of reprocessing tailings from the mine;
- the costs, incurred during the fiscal year, of repair and maintenance at the mine;
- general and indirect costs incurred during the fiscal year for property, employees or operations at the mine that are not otherwise allocated to operating costs;
- exploration costs incurred during the fiscal year by an owner of the mine
- a depreciation allowance, not exceeding the undeducted balance of the depreciable assets at the end of the fiscal year of the mine;
- a development allowance, determined by the operator, not exceeding the undeducted balance at the end of the fiscal year of the mine of
    - undeducted exploration costs incurred prior to the date of commencement of production
    - all costs incurred for the purposes of bringing the mine into production, less certain adjustments

- a qualifying environmental trust contribution allowance, determined by the operator, not exceeding the undeducted balance at the end of the fiscal year of amounts contributed to the qualifying environmental trust in respect of the mine; and
- if ore or mineral-bearing substances are processed by the operator of the mine prior to sale, an annual processing allowance equal to the lesser of
  - 8 per cent of the original cost of processing assets owned by the operator at the end of the fiscal year of the mine, and
  - 65 per cent of the value of the output of the mine, after deduction of the amounts referred to in the paragraphs above.

In respect of Inuit owned lands, Nunavut Tunngavik Inc. ("NTI") has reserved a royalty equal to 12% of the net profit derived from the operations as determined in accordance with the terms set out in the production lease. The royalty will be calculated based on gross revenues adjusted after the following considerations:

- Gross revenues shall include all proceeds from the sales of minerals, whether or not sold or paid for, sales of assets, insurance proceeds, withdrawals from an approved environmental trust and other sources of revenues but excluding any gains or losses from hedging.
- Allowable deductions, to the extent that they have not previously been deducted either under the NTI agreement or under CMR, including but not limited to the following:
  - All reasonable direct expenditures paid by the lessee in carrying out operations, including operating, preproduction and capital costs necessary to carry out such operations, both before and after the commencement of commercial production
  - All reasonable direct capital costs paid by the lessee after the commencement of commercial production for the purpose of improving, modifying or replacing any assets situated on the production lease and reasonably required to carry out operations;
  - Payment into an approved and qualifying environmental trust for the reclamation of the production lease area;
  - All prior payments under the royalty agreement;
  - Prior exploration costs within the production lease area, such amounts not to exceed the lessee's purchase price for the property.
- No allowance or deductions for items including the following:

- Corporate or general expenses not related to the carrying out of the operations;
- Amortization, depletion or depreciation, the royalty being determined on a cash basis;
- Off site facilities or costs not directly related to the carrying out of operations;
- Financing or taxes or costs deducted elsewhere under the NTI agreement or CMR.

Notwithstanding the available deductions, the aggregate amount of available deductions shall not exceed 85% of gross revenues with the effect of setting a minimum royalty payable of 1.8% pf gross revenues.

17. INTERPRETATION AND CONCLUSIONS

Based upon the work completed by SRK Consulting, Bateman Engineering, Nuna Logistics and the Hope Bay Joint Venture, development of the Doris North trial operation has the potential to generate significant cash flow while providing a wealth of operating experience. The performance of this trial operation could be instrumental in developing the full potential of the Hope Bay project.

The relevant features derived from this study which support this conclusion are:

- Project location
  - Good access as the site is within 3.5 km of tidewater
- Grade of the Doris North resources
  - High grade ores with a diluted grade of 18.9 g/t
- Geometry of the resource
  - Relatively simple lending itself to productive lower cost room and pillar mining techniques
- Near surface location of the resources
  - Shallow ore body provides both open pit potential and minimal development to access underground resources
- Readily available source of construction materials from the waste rock from the pit operation
  - Use of modular process plant construction which allows for most construction to take place away from site, this results in improved scheduling and reduced costs
- Low capital costs

- modular nature of the plant and use of contractor services

18. RECOMMENDATIONS

The Hope Bay Joint Venture, having determined that pursuit of a trial operation on the Doris North resources has the potential to meet internal economic thresholds, while providing invaluable operating experience is recommending pursuit of a detailed feasibility to support a construction decision. To this end the Joint Venture partners will seek approval from their respective Boards for funding to support the following program related to this study:

- An additional 7,200 m of infill drilling designed to confirm the resource numbers, upgrade the resources to the measured and indicated categories, and to facilitate detailed mine planning and ore release schedules
- Additional metallurgical studies to further optimize the process flow sheet
- Additional engineering work encompassing infrastructure, mine and process design to support a feasibility study
- Commencement of permitting activities to support construction and operations

If funding is made available, it is the Joint Venture's expectations that a feasibility study could be completed by the fall of 2002.

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