

# Mining Industry Questionnaire to Accompany Water License Applications to the Mackenzie Valley Land and Water Board



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*Regulating the use of land and waters and the deposit of waste, and enabling residents to participate in the management of resources to provide optimum benefit to the residents of the settlement areas and of the Mackenzie Valley and to all Canadians.*

## **October 2003**

The purpose of this questionnaire is to solicit supplemental information from an applicant to support his/her application for a water license (or renewal). It is anticipated that the completion of this questionnaire will reduce delays arising from the Board having to solicit additional information after an application has already been submitted. This information will also be useful during the pre-screening of your application, which must be undertaken prior to development and approval of a water license to determine if the project needs to be referred to the Environmental Impact Review Board.

The applicant should complete the questionnaire to the best of his/her ability, recognizing that some questions may not be relevant to the project under consideration. For questions that do not relate to his/her operation, the applicant is requested to indicate "N/A" (Not Applicable).

If any questions arise while completing the questionnaire, the applicant may wish to contact the Mackenzie Valley Land and Water Board at (867) 669-0506.

### **PLEASE PRINT OR TYPE YOUR RESPONSES**

**NOTE** If space is insufficient for any of the responses on this questionnaire, use the back of the sheet or an attachment.

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## SECTION 1 – GENERAL

Date: November 5, 2008

1.1 Applicant Fortune Minerals Limited (“Fortune”)  
Suite 1902, 140 Fullarton Street, London, Ontario N6A 5P2  
(519) 858-8188

Property Name: NICO Project<sup>1</sup>  
Closest Community: Whati, Northwest Territories  
Latitude/Longitude: 63° 33’ North & 116° 45’ West

1.2 Environmental Contact: Dr. Rick Schryer, Director of Regulatory and Environmental Affairs  
(306) 477-2114 /cell (306) 230-3019

1.3 Indicate the status of the mine and/or mill on the date of application. (check the appropriate space)

	Mine	Mill
Design	X	X
Under construction		
In operation		
Suspended		
Abandoned		

1.4 If a change in the status of the mine or mill is expected, indicate the nature and anticipated date of such change.

The underground advanced exploration program was completed in 2007. This program, undertaken in 2006 and 2007, provided a bulk sample for metallurgical and environmental characterization. Metallurgical characterization was by a pilot plant located at SGS Lakefield in Lakefield, ON that commenced in November 2007 and is ongoing. The ongoing work is related to optimization of the processes that produce bismuth and cobalt cathode metal.

The existing mine workings will be expanded to allow for mining at production rates after approval is received. In addition, the new surface open pit mine is expected to become operational 12 to 18 months after approval is received. Construction of processing and tailings/mine rock management areas is proposed for the commencement of pre-strip mining in the fall of 2010. Attachment 1 contains a list of acronyms, a list of units and a glossary of terms.

1.5 Indicate the ~~present or~~ proposed mine/mill operating schedule.

Table 1.5-1 Proposed Mine/Mill Operating Schedule

	Mine	Mill
Hours per day	24	24
Days per week	7	7
Weeks per year	52	52
Shift periods	underground <sup>2</sup> 2x10 hr	2x12 hr
	open pit 2x12 hr	2x12 hr
Number of employees	141	84

Pre-stripping of the open pit area is expected to last 12 months, and requires two 12-hour shifts per day.

<sup>1</sup> Alternative names for the mine are being considered.

<sup>2</sup> Years 1 -2.

**1.6 Attach a detailed map drawn to scale showing the relative locations of the (proposed) mine, mill, water treatment facilities, sewage and solid waste facilities, and tailings areas. The plan should include the water intake and pump house, fuel and chemical storage facilities, any existing and proposed concentrate, ore and waste rock storage piles, any existing and proposed drainage controls, piping distribution systems, gas, electric and water utility route locations, and transportation access routes around the site. The map also should include elevation contours, waterbodies and an indication of drainage patterns for the area.**

Figure 1 illustrates the study area covered under this mine application. Figure 2 shows drainage patterns for the study area, and Figure 3 shows the natural topography of the proposed mine area and the location of previous exploration activities. Figure 4a indicates the location of the proposed open pit mine, plant facilities, tailings and mine rock management areas, and related infrastructure. The camp is a graphical representation, and will be modified.

Figure 4b presents possible granular borrow quarry sites and possible rock fill borrow quarry sites. Should suitable construction material be available from stripping and early mining of the open pit and underground workings, these quarry sites may not be required.

Site infrastructure will include:

- mine (open pit and underground access portal);
- processing facilities;
- tailings management area (TMA);
- mine rock management area (MRMA);
- effluent treatment facility;
- transportation routes around site;
- collection (interception) trenches and sediment collection ponds;
- pump house and water intake; and
- fuel and chemical storage facilities.

Some features, not shown on the plans that have not been subject to detailed engineering and layout, will be provided on detailed plans later. These include:

- proposed drainage controls;
- utility routes; and
- piping.

The storage of concentrate will be required in order to provide for surge capacity prior to the hydrometallurgical processes to maintain steady-state milling and concentrating operations. The high pressure acid leach oxidation (cobalt) circuit and chloride leach electro-recovery (bismuth) circuit will occasionally require maintenance. Concentrates produced will be diverted to storage areas within the mill to allow for continued production. These storage areas will consist of tanks and a sump for cobalt concentrate. Should Fortune produce cobalt carbonate, bismuth cement, or bismuth concentrate instead of cathode metal, storage bunkers will be located at the processing facilities for packaging.

**1.7 If applicable, provide a brief history of property development which took place before the present company gained control of the site. Include shafts, adits, mills (give rated capacity, etc.), waste piles, chemical storage areas, tailings disposal areas and effluent discharge locations. Make references to the detailed map.**

From 1968 to 1970, New Athona Mines Ltd. explored a group of claims staked to cover two cobalt-bismuth-copper-arsenide showings which are part of the NICO leases. Exploration by New Athona included geological mapping, electromagnetic and magnetic surveys, trenching and approximately 4,600 feet (1,400 metres [m]) of diamond drilling in 21 holes. This work led to an inferred resource totalling approximately 215,000 tons, averaging 3.24 pounds of bismuth per ton with lesser values in cobalt,

copper and gold. Chemical analysis of a bulk sample of massive arsenopyrite yielded 2.36% cobalt, 0.63% bismuth, 22.5% iron, 16.0% sulphur, 40.8% arsenic, 0.18 ounces per ton (oz/t) silver, and 0.14 oz/t gold. In 1977 and 1978, Eldorado Nuclear Limited conducted exploration for uranium in the area, and Noranda Exploration followed up the New Athona work from 1978 to 1989.

Rock and drill cores excavated by New Athona Mines are stored on the rocky outcrop between the Lou Lake camp and the Grid Pond watershed. Drill cores from Fortune work are stored near the Lou Lake camp at the location shown on Figure 3.

### **1.8 Give a short description of the proposed or current freshwater intake facility, the type and operating capacity of the pumps used and the intake screen size.**

Although the proposed water intake facility has not been designed in detail, it is anticipated to require a combined working/nominal capacity of approximately 7,200 m<sup>3</sup>/d (cubic metres per day) or 83 L/s (litres per second) to accommodate both ore processing and camp operations.

The maximum water requirement for the process plant operation at full capacity will be 13,900 m<sup>3</sup>/d (161 L/s), of which 7,800 m<sup>3</sup>/d (90 L/s) will be recycled to the process from the tailings thickener overflow and from the hydrometallurgical circuit. This represents an internal recycle rate of 56%. As a result, the net fresh water draw from combined sources for process requirements is 6,100 m<sup>3</sup>/d (71 L/s). Because of the internal plant recycle, the net fresh water draw will quickly be achieved during start up operations.

As part of the front end engineering and in conjunction with the pilot plant, Fortune has identified additional technologies that could further reduce fresh water requirements. These locations include:

- Recycle of bismuth and cobalt concentrate thickener overflow solution within the process water tank (contain low concentrations of cyanide that may depress bulk concentrate flotation);
- Filtration of the carbon-in-pulp (CIP) circuit tailings with recycle of the cyanide solution within the CIP circuit, minimising cyanide destruction circuit requirements and optimising reagent use;
- Filtration and treatment of recycled water as cooling water in the autoclave, subject to meeting certain specifications and targets with respect to chloride and other scaling ions; and
- Reuse of decant water from the tailings pond supplied year-round by a barge.

Of the technologies listed above, the use of decanted water from the tailings pond would have the greatest impact in reducing the net fresh water draw.

At steady state condition, the overall tailings are anticipated to be discharged to the TMA at 60% solids, representing 3,096 m<sup>3</sup>/d (36 L/s) of water. Assuming that the tailings consolidate at an estimated 80% solids, approximately 1,161 m<sup>3</sup>/d (14 L/s) of water will be lost as pore water in the settled tailings, representing the absolute minimum fresh water draw required to sustain NICO process requirements, discounting other TMA inputs. The difference, 1,935 m<sup>3</sup>/d (22 L/s) would be available for reclaim water use as autoclave cooling water, or as process water for the concentrator. Use of decant water would be dependent on meeting certain specifications with respect to cyanide (a flotation depressant) and chloride (autoclave corrosion) as well successfully managing reclamation of the water during the winter season. Criteria for use of reclaim water will more than likely be met as the process water is diluted by other inputs to the TMA by a factor of approximately 3.4:1.

Potable and general water requirements for the camp, truck shop, haul road watering, and all other non-ore processing requirements during operations will be 1,100 m<sup>3</sup>/d (13 L/s). Using a ratio of 0.38 m<sup>3</sup> per person per day, which is based on water usage at other mining operations in the NWT, and the estimated permanent NICO camp population of 150 people, the daily volume of potable water required would be 57 m<sup>3</sup>/d (<1 L/s), a small proportion of the total general water requirement. During construction, potable water use could be as high as 228 m<sup>3</sup>/day (2.6 L/s) if the construction workforce reaches 600.

General water requirements, including potable and process water will be sourced at Lou Lake or Chalco Lake. Burke Lake has been designated to provide additional process water for the processing plant on an as-needed basis. The intake structure(s) will be designed to meet Department of Fisheries and Oceans Canada (DFO) guidelines for water intakes. The use of diesel-fired pumps to supply water from these sites has been proposed as an option, particularly for the case of Burke Lake, which is expected to be an intermittent operation. Further details are provided below in Section 1.9.

**1.9 At the rate of intended water usage for operations, explain water balance inputs and outputs in terms of estimated maximum draw down and recharge capability of the river or lake from which fresh water will be drawn.**

Three potential water withdrawal locations were considered. These are Lou Lake, Burke Lake and Chalco Lake. Of the three proposed sources, Lou Lake has the largest volume (12,767,000 m<sup>3</sup>) followed by Burke Lake (4,640,800 m<sup>3</sup>), and Chalco Lake (706,138 m<sup>3</sup>). Lou Lake was initially the sole preferred source of water for the mining operations; however, upon further consideration of regulatory criteria, it was decided that Lou Lake is only marginally suitable in terms of available supply, since water withdrawals from that source alone would be very close to the regulatory limit for winter withdrawal. Consequently, it is proposed that Lou Lake withdrawals would be reduced and some water for processing would come from Burke Lake. Another potential source of potable water could come from Chalco Lake. An overview of the decision process and the hydrological information follows.

Lou Lake would be adequate as a sole source of water during the construction period or as a source for the operational camp water; however, it may only marginally meet the mill processing requirements given restriction guidelines during the winter months. DFO (*Protocol for Winter Water Withdrawal in the Northwest Territories*, 2005) recommends that total water withdrawal from a single water body during one ice covered season should be limited to 5% of the under ice volume. This criteria has now been increased to 10% of the under ice volume (pers. comm. DFO). Both DFO documents also indicate that the assumed ice thickness for locations south of the tree-line to north of Fort Simpson is 1.5 m.

In winter, the water body volume is the liquid volume below the ice surface, calculated using the appropriate maximum expected ice thickness (1.5 m). For Lou Lake, the total volume (ice free) was calculated to be 12.77 Mm<sup>3</sup> (million cubic metres) as shown in Table 1.9-1. The volume of water below 1.5 m of ice was calculated to be 10.19 Mm<sup>3</sup>. Thus, the allowable volume that could be pumped from Lou Lake in winter is approximately 1.02 Mm<sup>3</sup> (10% of the under ice volume). Similarly, the under ice volume for Burke and Chalco lakes are 2.11 Mm<sup>3</sup> and 0.35 Mm<sup>3</sup>, respectively. Thus, the available volume over the winter for Burke Lake is approximately 211,000 m<sup>3</sup>, while Chalco Lake can supply 34,727 m<sup>3</sup>. This assessment of available supply assumes negligible inflow and outflow in the winter.

**Table 1.9-1 Water Volumes at Depth for the Top 2 m of Lou, Burke, and Chalco Lakes.**

Depth (m)	Lou Lake Volume (m <sup>3</sup> )	Burke Lake Volume (m <sup>3</sup> )	Chalco Lake (m <sup>3</sup> )
Surface	12,767,000	4,640,800	706,138
1.0	11,015,000	2,835,760	443,894
1.5	10,198,000	2,114,439	347,266
2.0	9,417,000	1,477,417	265,511

During construction, the annual potable water demand for the project is expected to be approximately 83,220 m<sup>3</sup>. Thus, 41,610 m<sup>3</sup> would be required during the six month winter period. This amounts to about 4% of the available supply from Lou Lake.

During operations, the maximum required process volume was calculated to be 2.226 Mm<sup>3</sup> annually. Over the six month ice cover period, the required volume for operations would be 1.113 Mm<sup>3</sup>, which is greater than the allowable volume from Lou Lake of 1.02 Mm<sup>3</sup>. With the additional available water from

Burke Lake (211,000 m<sup>3</sup>), the combined available water from both sources would be 1.231 Mm<sup>3</sup>, which exceeds the demand required for processing. Potable water for a 150 man camp during operations is approximately 20,805 m<sup>3</sup> per year or 10,402 m<sup>3</sup> over the winter months. This is approximately 30% of the available volume from Chalco Lake. Chalco Lake could serve as the source of potable water from the camp if needed. Lou Lake and Burke Lake could provide 1.231 Mm<sup>3</sup> of process water. When not utilizing a substantial component of the TMA decant water as reclaimed process water, a component of the general water consumption (1,100 m<sup>3</sup>/d) would need to be rationed in this situation by, for example, eliminating the washing of haul trucks. Fortune is examining all options for water use to meet regulatory requirements, while assuring an adequate water supply for the mine.

**1.10 Will any work be done that penetrates regions of permafrost?**

Yes

**1.11 If “Yes” above, is the permafrost continuous or discontinuous?**

Discontinuous.

Eight boreholes were drilled vertically to 15 m below ground surface in the probable TMA location (Figure 4b). Each was completed with a string of 10 thermistors to measure temperatures below ground; then the borehole was backfilled. Two of the three thermistors south of the Mine Rock Management Area (MRMA); namely EBA-04-05 and GA-06-07), and two of the five thermistors installed in the Tailings Management Area (TMA; namely, GA-06-10 and GA-06-11) indicated permafrost. The four thermistors showing permafrost are all located centrally in low-lying areas of the valleys, as shown on Figure 4b. The other four thermistors (namely, EBA-04-03, GA-06-09, GA-06-13, and MC-06-23), which did not show signs of permafrost, were located in high ground and hillsides. A 50 m long string of thermistors was installed in a ninth inclined exploration borehole located in the “bowl zone”, upstream of the ground water seep that flows into the stream draining into Grid Pond. This thermistor did not show evidence of permafrost.

The tailings dams will be founded in excavations dug down to bedrock, where practical, removing frozen soils with potential to thaw from the under the dam foundations. Where not practical to dig down to bedrock, excavation in permafrost will be taken down below the active zone with an anchor trench constructed in the permafrost.

**1.12 Were (or will) any old workings or waterbodies (be) dewatered in order to bring the present property into production?**

Yes. The exploratory bulk sampling ramp will be dewatered in order to provide ramp access to the underground mining operations.

The grid ponds and runoff collected during development will be dammed and utilized as start-up water supply.

**1.13 If “Yes”, above, indicate the name of the waterbody, the total volume of water to be discharged and the chemical characteristics of that water.**

Existing Workings:	Bulk sample ramp into deposit
Total Volume:	15,000 m <sup>3</sup>
Receiving Watercourse:	Grid Ponds

The following table illustrates the measured mine water quality recorded during dewatering in 2007 for the bulk sample program. The water chemistry of the receiving environment, which comprised the Grid Ponds, Nico Lake, Peanut Lake and Burke Lake, is also presented. Results shown are for the beginning of dewatering program (February 08, 2007) and after the dewatering was completed, when initial test

mining had commenced (March 26, 2007). These concentration ranges represent the anticipated water quality for dewatering of the mine workings when mine development is initiated. The exception to the indicated concentrations ranges is the ammonia concentration, which on March 26, 2007, reflected renewed use of explosives and is considered higher than would be expected during dewatering.

**Table 1.13-1 Mine Water Chemical Characteristics during Dewatering, 2007:**

Parameter	Units	MMER Criteria	Mine Water		Grid Ponds	Nico Lake	Peanut Lake	Burke Lake
			8Feb07	26Mar07				
<b>General Parameters</b>								
pH		6.0-9.5	8.1	8.1	7.7-8.1	7.3-7.8	7.2-7.8	7.4-7.8
Specific Conductivity	µS/cm	-	420	905	177-217	65.7-70.2	73.9-75	70.6-76.3
TSS	mg/L	15	4	9	<3-14	<3-4	<3	<3-6
<b>Nutrients</b>								
Total NH <sub>4</sub> - N	mg/L	-	2.80	22.3	0.030-0.083	<0.005	0.02	<0.005
Un-ionized NH <sub>4</sub> <sup>a</sup>	mg/L	-	0.04	0.4	0.002	<0.00007	0.0003	<0.00007
<b>Total Metals</b>								
Aluminum	mg/L	-	0.07	0.83	0.02-0.06	0.02	0.02	0.03
Arsenic	mg/L	0.5	0.0649	0.0456	0.135-0.218	0.0092	0.0054	0.0021
Cadmium	mg/L	-	<0.0002	<0.0002	<0.0001	<0.0001	<0.000	<0.0001
Copper	mg/L	0.3	0.006	0.008	0.0032-0.0098	0.0013	0.0007	0.0011
Lead	mg/L	0.2	0.0005	0.0044	<0.0001	<0.0001	<0.000	0.0001
Manganese	mg/L	-	n/a	n/a	0.003-0.027	0.007	0.036	0.003
Mercury	mg/L	-	n/a	n/a	<0.0001	<0.0001	<0.000	<0.0001
Nickel	mg/L	0.5	0.0026	0.0045	0.0004-0.0008	0.0008	0.0006	0.0007
Zinc	mg/L	0.5	0.038	0.009	0.010-0.013	0.005	0.008	0.008
<b>Other</b>								
Total Cyanide	mg/L	1	n/a	n/a	n/a	n/a	n/a	n/a
Oil & Grease	mg/L	-	<1	2	n/a	n/a	n/a	n/a

Notes: MMER = Metal Mining Effluent Regulations. Canada Gazette Part II, Vol. 136. No.13. SOR/DORS/2002-222;  
 Feb = February; Mar = March; N = nitrogen; NH<sub>4</sub> = ammonia; mg/L = milligrams per litre; µS/cm = microSiemens per centimetre; TSS = total suspended solids  
 a = Un-ionized ammonia is dependent on the total ammonia concentration, the temperature and pH condition (i.e. value increases with both). The calculated un-ionized ammonia concentration shown is based on the pH measured and assumed reasonably conservative temperatures.

Relative to the existing water quality in the Grid Pond system, aluminum, lead, nickel, zinc and conductivity levels in the dewatering discharge were elevated. Notable is arsenic that was much lower in the mine water than in the Grid Ponds. All of the results were well below the Metal Mining Effluent Regulations (MMER) criteria.

At the end of the exploratory bulk sampling program, the underground pumps were removed and the majority of the mine water was stored underground at the 115 m level. The majority of water stored on the surface in lined ponds was either discharged in 2008 after meeting the discharge criteria of the exploration water license, or used by fire fighters to battle the forest fire that threatened the NICO site in 2008. When access to the decline ramp is required for mine development, stored water will be pumped to surface. This water will have been stagnant for a number of years and additional testing of the mine water will be undertaken to further assess the concentrations. It is anticipated that the starter dam at the TMA will be complete during dewatering, and the water will be stored for use during start-up of the mill.

**1.14 Was (or will) the above discharge (be) treated chemically?**

The water chemistry from underground dewatering will be tested prior to discharge. As indicated above, it is anticipated that this water will be stored for use in process. Excess water in the TMA is diverted to the ETF for treatment prior to discharge, if necessary.

**If “Yes” above, describe the applied treatment.**

The mine water is not anticipated to require treatment for mine dewatering purposes, based on the chemistry presented above and due to the planned use in process as make-up water. Excess water in the TMA will be diverted into the ETF. A portion of this water would then be recycled into the mine process or treated and released as discussed in later sections of this questionnaire.

## SECTION 2 -- GEOLOGY AND MINERALOGY

**2.1 Physiography: Provide an analysis and interpretation of the geologic and hydrologic environment in the immediate vicinity of the mine or plant. The investigation should extend from ground surface downward to the base of the glacial drift. Include large scale topographic map(s) covering the area where the mine, mill and waste disposal basin are (or are to be) located. The map(s) should provide information on ground water patterns and permafrost variations in the area.**

Figure 2 illustrates the watersheds for the regional study area. Figure 3 shows the existing exploration facilities and topography. The location of permafrost and groundwater monitors are shown in Figure 4b.

**Table 2.1-1 Geological and Hydrological Environment in Mine and Plant Vicinity**

Feature	Mine	Mill and Underground Portal	Mine Rock/Tailings Area
Geology	The proposed mine is situated mainly on a hill of exposed bedrock: metasedimentary rocks and felsic volcanic rock cut by porphyritic dykes.  The north end of the open pit will be in the Grid Stream depression, known as the Bowl Zone, which has several metres of till underlying organics.	The mill and mine portal will be mainly on a flat-lying area of exposed bedrock (altered felsic volcanic and greywacke cut by porphyritic dykes) or shallow organic topsoil cover.	The tailings areas are characterized as low lying land of shallow glacial till overburden or bog, several metres thick, bound by hillsides of exposed bedrock. Bedrock in the TMA consists of altered felsic volcanic and greywacke cut by porphyritic dykes. In the MRMA bedrock consists of altered siltstones with local porphyritic dykes.
Hydrology	The open pit will be at a height of land. Ground water is within 10 metres of ground surface and the water table mimics the topography.  The north end of the pit will encroach into the Bowl Zone, uphill of the stream and ponds.	Only surface runoff is expected. The site is a raised area east of the open pit and southeast of TMA	Both the MRMA and the TMA are in runoff catchments. The MRMA is at the top of its catchment. The TMA has a small catchment (Grid Stream) upstream.

Notes: TMA = tailings management area; MRMA =mine rock management area

**2.2 Briefly describe the physical nature of the orebody, including known dimensions and approximate shape.**

The mineralization at NICO is generally in three sub-parallel zones of ironstone, composed of iron rich biotite and amphibole, magnetite, hematite and feldspar with, or without, chlorite and carbonate. These are separated by weakly altered host rock. The three zones strike northwesterly and dip to the northeast at moderate angles. They range in thickness from 20 m to 60 m and occur below an area 1.4 km long by 300 m wide.

**2.3 Briefly describe the country rock in the general vicinity of the orebody (from the surface to the orebody).**

The deposit is hosted within a 200 m thick package of northwest-striking and northeast-dipping metasedimentary rocks (schist and greywacke). The greywacke is overlain by a rhyolite volcanic rock. Two or more generations of coarse-grained porphyritic dykes, exposed on surface, crosscut the ore zone and surrounding rocks.

**2.4 Provide a geological description of the ore minerals of the deposit. (If possible include the percentage of metals.)**

The sulphides are typically disseminated (as opposed to concentrated and massive) and represent 3% to 10% (averaging 5%) of the mineralized zones. The arsenopyrite-cobaltite grains are commonly aligned along the foliation planes. Bismuth typically occurs as bismuthinite and native bismuth. Gold occurs as grains of native gold ranging from <1 to >100 microns ( $\mu\text{m}$ ) in size attached to various sulphide and gold-bismuth telluride grain boundaries, as inclusions with sulphide minerals and associated with silicate gangue minerals. Chalcopyrite, pyrrhotite and pyrite are present as local fracture fillings or localized disseminations.

**2.5 Describe the geochemical tests which have been (or will be) performed on tailings solids and different geological units of ore, country rock and waste rock to determine their relative acid generation and contaminant leaching potential. Outline methods used (or to be used) and provide test results in an attached report (i.e., static, kinetic tests).**

**Geochemical Test Methods**

Geochemical testing was performed in two stages: static and kinetic testing. The components of each stage are outlined below.

Static testing methods used to describe rocks at the NICO deposit during previous and ongoing test programs include the following:

- Acid-Base Accounting (ABA) – used to develop estimates of the potential for acid generation based on the balance between acid producing and acid buffering minerals;
- Trace Metals Analysis on Solids – used to determine the total amount of metals in the solid phase of the rock samples;
- Short-Term Leach Testing – used to develop initial estimates of metal leaching from weathered materials;
- Net Acid Generation (NAG) test – used for estimates of acid generation potential and end-point chemistry evaluation on acid generating and non-acid generating materials; and,
- Mineralogy – used to identify mineral assemblages as they have a large influence on metal leaching, acid generating and buffering reactions.

Data obtained from the laboratory program is compiled and summarized to determine the potential for acid generation and metal leaching.

The purpose of kinetic testing is to confirm the results of static testing. Humidity cell tests are laboratory scale, repetitive leach tests that are performed to evaluate the rate of reaction of rocks, both acid generating and non-acid generating, over a period of weeks to months. The testing is designed to enhance sulphide oxidation and/or weathering reactions, so that meaningful information with respect to leachate water quality can be developed in a relatively short period of time, as compared to actual field conditions (which may take years to develop). In addition to lab scale testing, field scale leach testing was initiated in August 2008, where leachate is collected at regular intervals from waste rock samples that are exposed to atmospheric conditions.

**Ore, Host Rock and Country Rock**

Samples of NICO ore, host rock and associated country rocks representing 6 different rock types were collected from approximately 37 holes drilled across the deposit (Figure 4b): breccia; black rock schist; feldspar porphyry; rhyolite; siltstone; and, sub-arkosic wacke.

Geochemical characterization was performed in two stages. Static testing, including acid base accounting, whole rock analysis and short-term leach testing, was performed to identify rock types with acid rock drainage and metal leaching potential. Acid base accounting was performed on 181 samples, and short-term leach testing on a sub-set of samples. Kinetic testing (i.e., humidity cell testing) was initiated to confirm the results of static testing. Based on the results of the static tests, 12 country rock samples (representing 8 rock types) were selected for humidity cell testing. Supplemental geochemical characterization, including acid base accounting, short-term leach tests, net acid generation tests, and field scale leachate tests is underway as of the time of submission of this application.

### ***Acid Generation Potential***

The acid generation potential of rock types at the NICO deposit was determined using the results of acid base accounting and kinetic testing. An initial assessment of acid generation potential was defined using the results of acid base accounting (i.e., neutralization potential and sulphide concentration). Acidity is generated by the oxidation of sulphide minerals in a sample. Alkalinity, released by the weathering of the rocks, neutralizes acidity generated by sulphide oxidation. The amount of alkalinity available to leach from a rock is represented by the measure of neutralization potential. Thus, samples with an excess of neutralization potential relative to sulphide concentration are considered non-acid generating. The rate of sulphide oxidation and rock weathering was quantified by repetitive leach tests (i.e., humidity cell tests) under laboratory conditions.

The results of static testing indicated that a small proportion of each rock type is capable of generating acidity, based on the relative concentration of sulphide minerals and neutralization potential contained in the rock. Kinetic testing confirmed that rocks with elevated sulphide concentrations (i.e., greater than approximately 0.27% sulphide) are capable of generating acidity in the long-term. One rhyolite sample, containing 0.49% sulphide, produced acidity through the duration of the humidity cell test. One sample of black rock schist, containing 0.27% sulphide, had a limited potential for long-term acid generation potential.

Humidity cell testing confirmed that sulphide-poor rhyolite, black-rock schist, feldspar porphyry, breccia, siltstone and sub-arkosic wacke rock types will not generate acidic leachate in the long-term. Rocks with low sulphide concentrations have slow rates of sulphide oxidation, and sufficient neutralization potential exists to buffer the acidity generated by sulphide oxidation. Field testing of representative waste rock and ore types is underway in order to evaluate the rate of weathering under site conditions.

Furthermore, as sulphide-rich rocks outside of the reserve model typically contain concentrations of cobalt, bismuth and gold, Fortune may segregate sulphide-rich rocks for the purpose of processing during periods of high metal prices. Should segregation of rock with elevated sulphide concentrations not occur throughout the operating period of the mine, mitigation strategies will be implemented if necessary.

### ***Metal Leaching***

Short-term and long-term leach testing was performed in order to identify the metal leaching potential of ore, host rock and country rock. Metals that leached from rocks under acidic conditions during humidity cell testing included iron, copper and aluminum. Aluminum and arsenic exhibited a significant potential to leach in neutral conditions from most rock types that underwent humidity cell testing.

Elevated arsenic concentrations were identified in short-term and long-term leachates from most rock types, regardless of the solid phase sulphide or arsenic concentration in the rock. Short-term leach tests reported arsenic concentrations in excess of the MMER discharge limits. Arsenic concentrations were elevated in humidity cell test leachates, but rarely exceeded the MMER criterion. The highest arsenic concentrations measured during humidity cell testing were noted in samples of black rock schist and rhyolite (and to a lesser degree, breccia and wacke).

Elevated of arsenic concentrations in the humidity cell leachate do not necessarily correlate with the solid

phase sulphide or arsenic concentrations. Rocks with relatively low solid-phase arsenic concentrations are capable of contributing arsenic to leachate. The rate of arsenic release could be controlled by the mineralogical form(s) of arsenic in the rock. Given the fact that much of the waste rock is considered unlikely to generate acidity in the long-term (based on the results of geochemical testing), arsenic solubility under site conditions is a consideration in long-term mine planning and engineering designs.

Additional testing and evaluation is underway to evaluate the mineralogical source and spatial distribution of arsenic in the country rock / host rock for the purpose of rock control and site designs. Rock with a potential for metal leaching may require special handling and storage, or other mitigation over time. Metal leachate is a current consideration in mine design, and will continue to require consideration. All material selected for use as aggregate to build roads, site infrastructure fill and tailings dam structures will be tested to limit the use of potentially acid generating / metal leaching material in site infrastructure.

### **Tailings**

Static and kinetic testing of tailings is ongoing. To date, three tailings products sourced from the mini-pilot plant stage of flow sheet development have undergone geochemical testing: a composite tailings sample (representative of the average tailings composition at the NICO mine), cyanide destruction tailings and iron-arsenic precipitate. Humidity cell testing of flotation tailings is underway. Field testing of tailings, generated from the pilot plant, under site conditions will be initiated in the Spring of 2008.

Static testing of cyanide destruction (CD) tailings and the iron-arsenic precipitate (FAP) identified a potential for acid generation: these streams only constitute minor components of the overall tailings, 10 % and 2% respectively. CD tailings contained low concentrations of most metals during humidity cell testing; arsenic concentrations exceeded the MMER criterion during the initial stages of testing, and decreased to concentrations below the criterion. Arsenic concentrations in FAP tailings humidity cell test leachate generally exceeded the MMER discharge limits. In addition to arsenic, metal concentrations that exceeded the MMER discharge limits in FAP humidity cell test leachate included aluminum, copper, nickel and zinc.

A composite tailings sample comprised a mixture of aged flotation tailings, CD tailings and iron-arsenic precipitate (FAP) in the approximate proportions indicated by the feasibility processing plan. This sample did not generate acidity during humidity cell testing over 172 weeks; however, original calculations indicated that, in the long term, the acid generating capacity of this sample may exceed the neutralization potential in the long-term. This difference demonstrates the importance of laboratory and field testing to correlate different testing conditions or assumptions to actual conditions. Concentrations of most metals were generally low in the leachate from the composite tailings: arsenic exceeded the MMER criterion during the initial weeks of the test, and decreased below the criterion through the remainder of the 40-week test.

Tailings decant water will be directed to the ETF, described in more detail in subsequent sections, or collected for recycling in the process.

Fortune will continue to refine its approach to the final disposal of tailings and waste rock as mine development moves forward.

**2.6 Estimate the percentage of sulphides in the orebody:**

**Table 2.6-1 Percentage of Sulphides in the NICO orebody**

<b>Sulphide</b>	<b>Maximum (Percent)</b>	<b>Average (Percent)</b>
Pyrite	5	<1
Pyrrhotite	3	<1
Pyrite/pyrrhotite mixture	2	0.25
Arsenopyrite	10	4

## SECTION 3 -- THE MINE

### 3.1 Indicate the type of mining method to be used on the property.

Open Pit X  
 Underground X  
 Strip Mining

#### Other mining activity? Explain.

N/A

### 3.2 Outline any possible operational changes and when they might occur (i.e., open pit to underground).

In year 3 of mine operations, the mine will switch from a concurrent underground and open pit operation to only an open pit operation.

### 3.3 Describe the type(s) of explosives to be used in mining operations.

Explosives will be used in underground and open pit mining operations, as well as during construction of surface facilities. ANFO (ammonium nitrate & fuel oil), high explosive detonators (sometimes referred to as 'stick powder'), and blasting caps will be stored in separate magazines at the Explosives Storage area. Emulsion products will be used sparingly, but may be utilised during freshet, or during periods of high ground water infiltration or precipitation. The manufacture of ANFO will be the responsibility of a dedicated blasting contractor, who will furnish the mixing plant, transport equipment and personnel.

### 3.4 Indicate the number of shafts or other openings that are presently on the property. Signify whether or not the openings are presently in use: (submit measurements in metres)

**Table 3.4-1 Shafts and Adits Present at NICO and Present State of Development**

Shaft (name or number)	Present Depth (mbgs)	Present Elevation (m)	Proposed Depth (mbgs)	Proposed Elevation (m)
Ventilation Shaft	166	154	166	154
<b>Adit (name)</b>				
Bulk Sample Ramp	143 <sup>a</sup>	115	168 <sup>a</sup>	90

Notes: m = metres; mbgs = metres below ground surface.

a = Relative to portal elevation of 258 metres above sea level.

Open Pit (name)	NICO Open Pit
Present surface length	Not open yet
Maximum future surface length	1340 m
Present surface width	Not open yet
Maximum future surface width	460 m
Present depth	Not open yet
Maximum future Depth	230 m

**Table 3.4-2 Proposed Area and Height for the Rock Storage Facilities**

Rock Pile	Area Occupied (hectares)	Height (metres)
Mine Rock Management Area	78.14	92

**3.5 Are any entrances to shafts, adits, etc. below ground water level?**

No

**3.6 Are permafrost conditions expected?**

No permafrost conditions have been identified or are expected in the mine areas. Permafrost has been indicated in the tailings and dam areas. Permafrost has also been indicated in the MRMA, but downstream of the structure (Figure 4b).

**3.7 Indicate the expected life of the mine.**

15 years based on current reserve estimate

**3.8 Indicate the present average rate of production from all ore sources on the property.**

No current production

**3.9 Indicate the expected maximum rate of production from all ore sources on the property.**

The average rate of production will be 4,650 t/d (tonnes per day) of ore (1.695 million tonnes per year), with a maximum production of 5,160 t/day.

**3.10 Outline all water usage in the mine, indicating the source and volume of water for each use.**

**Table 3.10-1 Expected Water Usage for the NICO Mine**

Source	Use	Volume (m <sup>3</sup> /d)
Burke and Lou Lake	Ore Processing	6,100
Lou Lake and/or Chalco Lake	Camp Requirements, Haul Road Watering and all other, non-ore processing requirements	1,100
<b>TOTAL</b>		7,200

Notes: m<sup>3</sup>/d = cubic metres per day

The sum of all water usages, for planning purposes, and conservatively assuming no recycling of water collected in the tailings pond (the maximum), is estimated at approximately 6,300 m<sup>3</sup>/day, or 83 L/s.

**3.11 Indicate the volume of natural ground water presently gaining access to the mine workings.**

Based on the dewatering rate from the underground bulk sample ramp, the volume of water seeping into the mine workings in summer, fall and winter will be approximately 5 m<sup>3</sup>/d.. During the spring melt, the flow rate will be approximately 40 m<sup>3</sup>/d..

**3.12 Outline methods used (planned) underground to decrease mine water flow. (i.e., recycling)**

For the purpose of this section of the questionnaire, and based on the clarifications provided by the questions themselves, mine water is considered to represent water pumped from the mine, exclusive of any additional water that may be required for processing from sources external to the mine itself.

No measures are anticipated to decrease ground water flow into the underground mine. The exploration boreholes will be grouted prior to mining to minimize flow from surface. Mine water will be directed through the processing plant for use in processing. In other words, mine water will be fully recycled to the processing plant through the tailings thickener.

**3.13 Indicate the average daily volume of water to be discharged from the mine during normal operations.**

Mine water will not be discharged from the mine directly to the environment. This water will be recycled through the TMA or the processing plant. Ultimately, all mine and process water subject to discharge to the environment will be treated in an ETF, which is described in a subsequent section of this

questionnaire.

On average, approximately 40 m<sup>3</sup> of water per day will be pumped from the mine during operations. This quantity will vary substantially at different stages of mining since the underground operations will cease after 2 years and the open pit will gradually progress deeper.

**3.14 If a mill will be operating on the property in conjunction with mining, will all mine water (underground, open pit, etc.) be directed to the mill for reuse?**

Yes, the mine water will be pumped to the tailings thickener or the TMA. Tailings thickener overflow will be recycled as make-up process water.

**3.15 If not, indicate the proposed point and volume of discharge for the mine water.**

N/A

**3.16 What are the chemical and physical characteristics of the preceding mine water?**

Three exploration drill holes that penetrated the country rock and ore body were completed with the installation of standpipe piezometers to serve as monitoring wells, to allow sampling of water at depth that would be representative of future mine water (Table 3.16-1).

**Table 3.16-1 Physical Characteristics of the Three Exploration Drill Holes**

<b>Drill Hole</b>	<b>Inclination (Degrees)</b>	<b>Sample interval in vertical metres below ground surface</b>	<b>Geology of the sample interval</b>
03-283	-65	60 to 85	Upper Ore Zone
03-282	-45	105 to 129	Country Rock or Mine Rock, including a dyke
03-281	-45	69 to 83	Lower Ore Zone and underlying Country Rock

Because the underground workings will be excavated to 160 metres below ground surface (mbgs), measured from the portal opening (which will be at about elevation 240 metres above sea level (masl)), the 2004 water quality analyses from the monitoring wells provides a reasonable indication of the expected mine water quality (Table 3.16-2). Although this water is not proposed to be discharged directly to the environment, the MMER Criteria have been referenced below for context.

These three standpipe monitoring wells listed above are completed with 2" ID PVC tubing, and are sealed above the screened intervals. Each well has been sampled twice for groundwater quality, once in 2004 and once in 2005. These monitoring wells will be sampled again in 2009.

**Table 3.16-2 Chemical Characteristics from the Monitoring Wells:**

Parameters	Units	MMER Criteria	03-281	03-282	03-283
<b>Field Parameters</b>					
Dissolved Oxygen	mg/L	-	2.2	1.65	1.4
pH		6.0 to 9.5	9.26	11.7	9.8
Redox	mV	-	152	129.5	141
Temperature	°C	-	2.7	7.5	6.6
<b>General Parameters</b>					
Alkalinity as CaCO <sub>3</sub>	mg/L	-	88	93	51
Conductivity	µS/cm	-	258	298.5	296
Hardness asCaCO <sub>3</sub>	mg/L	-	104	88.5	54
pH		6.0 to 9.5	8.3	11.1	9.4
Total Dissolved Solids	mg/L	-	148	148	177
<b>Nutrients</b>					
Ammonia as N	mg/L	-	0.011	0.034	0.112
Un-ionized Ammonia <sup>a</sup>	mg/L		0.002	0.042	0.066
Phosphorus	mg/L	-	0.089	0.011	0.153
Nitrate as N	mg/L	-	0.291	0.0215	0.064
Nitrite as N	mg/L	-	0.006	0.002	0.005
Total Kjeldahl Nitrogen	mg/L	-	0.44	0.185	0.22
Total Organic Carbon	mg/L	-	7	3.5	3
<b>Major Ions</b>					
Bicarbonate (HCO <sub>3</sub> )	mg/L	-	107	<5	36
Calcium (Ca)	mg/L	-	19.1	33.95	12.6
Carbonate (CO <sub>3</sub> )	mg/L	-	<5	24	13
Chloride (Cl)	mg/L	-	2	1.5	5
Hydroxide (OH)	mg/L	-	<5	18	<5
Magnesium (Mg)	mg/L	-	13.6	0.9	5.4
Potassium (K)	mg/L	-	4.9	2.7	12
Sodium (Na)	mg/L	-	12	18	34
Sulphate (SO <sub>4</sub> )	mg/L	-	41.9	35.5	77.2
<b>Total Metals</b>					
Aluminum (Al)	mg/L	-	0.03	0.01	0.04
Antimony (Sb)	mg/L	-	0.006	0.00265	0.0127
Arsenic (As)	mg/L	0.5	0.404	0.0785	0.272
Barium (Ba)	mg/L	-	0.0334	0.04445	0.0186
Beryllium (Be)	mg/L	-	<0.0005	<0.0005	<0.0005
Bismuth (Bi)	mg/L	-	0.00045	<0.00005	0.00009
Boron (B)	mg/L	-	0.037	0.0365	0.097
Cadmium (Cd)	mg/L	-	<0.0001	<0.0001	0.0002
Cesium (Cs)	mg/L	-	<0.0001	0.0001	0.0001
Chromium (Cr)	mg/L	-	0.0027	0.00885	0.0008
Cobalt (Co)	mg/L	-	0.0077	0.00105	0.0079
Copper (Cu)	mg/L	0.3	0.0074	0.0033	0.0174
Iron (Fe)	mg/L	-	0.852	0.027	0.073

Parameters	Units	MMER Criteria	03-281	03-282	03-283
Total Metals (Continued)					
Lead (Pb)	mg/L	0.2	0.0002	0.0001	0.0002
Lithium (Li)	mg/L	-	0.0014	0.00695	0.0069
Manganese (Mn)	mg/L	-	0.065	0.003	0.012
Mercury (Hg)	mg/L	-	<0.0001	<0.0001	0.0001
Molybdenum (Mo)	mg/L	-	0.0057	0.0243	0.0341
Nickel (Ni)	mg/L	0.5	0.0015	0.0004	0.0018
Rubidium (Rb)	mg/L	-	<0.05	<0.05	<0.05
Silver (Ag)	mg/L	-	<0.0002	<0.0002	<0.0002
Selenium (Se)	mg/L	-	0.0004	<0.0004	<0.0004
Strontium (Sr)	mg/L	-	0.0894	0.19	0.144
Thallium (Tl)	mg/L	-	<0.00005	<0.00005	<0.00005
Tin (Sn)	mg/L	-	0.0002	<0.0002	0.0002
Titanium (Ti)	mg/L	-	0.0006	0.00035	0.0018
Uranium (U)	mg/L	-	0.0018	0.0004	0.0941
Vanadium (V)	mg/L	-	0.0002	0.00205	0.0061
Zinc (Zn)	mg/L	0.5	0.166	0.007	0.017

Notes: MMER = Mining Metal Effluent Regulations. Canada Gazette Part II, Vol. 136. No. 13. SOR/DORS/2002-222;

mg/L = milligrams per litre; mV = millivolts; °C = degrees Celsius; CaCO<sub>3</sub> = calcium carbonate;

µS/cm = microSiemens per centimetre; N = nitrogen; < = less than; - = no value available

a = Un-ionized ammonia is dependent on the total ammonia concentration, the temperature and pH condition (*ie.* Value increases with both). The calculated un-ionized ammonia concentration shown is based on the pH measured and assumed reasonably conservative temperatures.

The ground water is characterized by alkaline pH values (8.3 to 11.1) with concentrations of total dissolved solids ranging from 147 mg/L to 177 mg/L. The pH values measured in monitoring wells 03-282 (11.1) and 03-283 (9.4) are very alkaline and exceed the MMER criteria. None of the total metal concentrations exceed the MMER criteria, although the arsenic concentration in one sample approaches it.

Water accumulation in the pit and underground workings, as well as water used for drilling and excavating, will be pumped or gravity fed to sumps where fines will settle out. Flocculant may be used in the underground sumps to minimize suspended solids being pumped to surface. From the sumps, the water will be pumped to the surface for use in the processing plant.

**3.17 Are there any treatment plans for mine water and will any chemicals be used in such treatment? Explain.**

The mine water will not be treated separately, with the possible exception of flocculants to lower suspended solids levels. It will be pumped to the tailings thickener and recycled to the process in the thickener overflow.

## SECTION 4 -- THE MILL

### 4.1 Attach a copy of the (proposed) mill flow sheet. Indicate the points of addition of all the various reagents (chemicals) that are (or will be) used.

The following paragraphs provide a general description of the proposed process for recovery of cobalt, gold and bismuth. An overview flow sheet of the basic plant process is shown on Figure 5a. Block diagrams showing the processes related to the recovery of cobalt and bismuth cathodes are provided in Figures 5b and 5c, respectively. Block diagrams comparing the recovery of copper by precipitation as outlined by the basic plant flow sheet in Figure 5a, versus a proposed alternative recovery by copper solvent extraction and electro-winning (SX/EW) is shown in Figure 5d.

#### Comminution

Run-of-mine ore from underground and open pit operations at 5% moisture will be stockpiled on the surface adjacent the crusher, if necessary. Ore from the open pit and underground will be direct dumped to a primary gyratory crusher at a top size of 1 metre. Blended ore from run-of-mine stockpiles can also be fed to the primary crusher by front-end loader. During transfer of the product to secondary crushing, an in-line belt magnet will remove tramp steel from the ore.

Secondary crushing will be completed by a standard head cone crusher and the product will be screened at 19 mm, producing a crushed ore product that will be conveyed to a fine ore bin. Oversize from the screen will be further reduced by a tertiary short head cone crusher operating in a closed circuit with the same screen. The fine ore bin will have a live capacity of 3500 tonnes per day, allowing for crushing to be completed over 12 hours. Feed to the milling circuit will be directed from the fine ore bin at 80% passing 12 mm.

While not shown on the flow sheet in Figure 5a, allowance has been made in the crushing plant design for the production of gravel, by use of a diversion gate after the secondary crusher. The resulting product at minus 75 mm will be stockpiled by a mobile stacker. Similarly, a surge stockpile of ore will be managed outside of the fine ore bin by use of a reversible conveyor located on the top of the fine ore bin. The conveyor will divert ore to be stockpiled by the same mobile stacker used for production of crushed waste rock. The surface fine ore stockpile will be returned to the process by a dedicated reclaim hopper and conveyor. Design for the deck for these two stockpiles will incorporate collection and return of water to the process plant or diversion to the TMA.

The fine ore will be fed to a primary grinding circuit consisting of a 12' diameter x 17'-8" rod mill (820 kW), producing a product of 80% passing 1232  $\mu\text{m}$ . NICO ore hardness increases with depth; and as a result, the milling rate achieved will be variable over the life of the mine. The average operating feed rate will be 215 dry tonnes per hour at 90% availability, resulting in 1,695,000 tonnes per year processed.

The rod mill discharge will be combined with the underflow (oversize) material from the secondary grinding circuit hydrocyclone pack, which operate in closed circuit with three 12' diameter x 14'-8" ball mills (970 kW). The cyclones will maintain the product from the secondary milling circuit at a size of 80% less than 72  $\mu\text{m}$ .

Water for the grinding operation will be obtained primarily from recycled water recovered from the bulk tailings thickener, concentrate thickeners, the cobalt hydrometallurgical operation, and reclaimed water from the TMA. The use of reclaimed water from the TMA will be determined by the meeting of cyanide and chloride specifications, which are detrimental to the bulk flotation circuit, and to autoclave corrosion, respectively. As the process water discharged to the TMA is diluted by other inputs such as runoff and potable water at a ratio of 3.4:1, the meeting of the criteria is more than likely. This is important as the recycle rate from the TMA largely controls the fresh water requirements at NICO.

#### Flotation

Bulk flotation of the ground slurry using potassium amyl xanthate (PAX) and methylisobutyl carbinol (MIBC) will produce a combined cobalt/bismuth bulk concentrate in a circuit called the *rougher* stage. PAX is commonly called the *collector*, while MIBC is referred to as the *frother*. The combination causes the target minerals to be adsorbed onto the surface of bubbles introduced into the agitated tanks through aeration. The froth overflows the vessels and is collected separately as a concentrate, while the rejects making up the majority of the feed are discharged as tailings. The flotation circuit can be built in a series

of sequential stages called *cells*, resulting in the desired mass pull. For example, for the bulk rougher circuit, there will be six stages with a total retention time of 45 minutes, and tailings from the last of the bulk rougher cells make up the majority of the plant tailings. Because the flotation is targeting sulphide minerals, the resulting tailings are extremely stable and contain a substantial amount of neutralizing potential. This tailing stream represents approximately 85.7% of the feed to the TMA.

The resulting bulk rougher concentrate will be further upgraded using PAX in a second flotation circuit called the *cleaner* stage. In the bulk cleaner stage, concentrate from the rougher circuit is refloated in a series of cells, which flow in a manner so that the product concentrate is successively refloated in a counter current fashion by the flotation cell that precedes it. The bulk cleaner concentrate is produced at the head of the train, while at the final stage; cleaner tailings are produced, resulting in the second and final effluent stream from the flotation circuit. The bulk cleaner tailings represent approximately 8.2% of the eventual feed to the TMA.

The bulk cleaner concentrate is fed to a cyclone that operates in closed circuit with a regrind mill. The product of the hydrocyclone is selected for size at 80% percent passing 14  $\mu\text{m}$ . Larger material is rejected by the cyclone to the regrind mill for size reduction. Additional PAX is added during grinding to ensure intimate contact with the target minerals. The regrind circuit cyclone product is subjected to separation flotation using dilute cyanide addition to depress the target sulphide minerals and prevent them from floating, specifically arsenopyrite, which contains the cobalt. In this capacity, cyanide is referred to as the *depressant*. On the other hand, PAX collector will selectively float the bismuth in this condition, producing fine bismuth concentrate representing a very small fraction of the overall plant feed, approximately 0.4%. In the bismuth rougher flotation circuit, much smaller flotation cells are utilized with six stages with a total retention time of 14 minutes. The resulting bismuth rougher concentrate, is subsequently cleaned in a manner similar to the bulk cleaner circuit. Unlike the bulk flotation circuit, there are no tailings – material rejected by the circuit represents the cobalt concentrate.

Cobalt concentrate storage is planned in order to provide surge capacity prior to the autoclave. Surge capacity will be provided by a large tank representing 48 hours of concentrator operation, surrounded by a storage pit excavated and lined within the bedrock, representing approximately 7 days of concentrator operation, and enclosed within the heated building containing the autoclave. Storage capacity representing 1 week of production will be constructed in order to allow for maintenance of the single-train autoclave equipment, with a decanted capacity of 1,150 dry tonnes. A monitor (high pressure water) capable of supporting pumping operations in order to return the concentrate to the cobalt thickener for dewatering and preparation as autoclave feed will be installed at the storage pit.

### **High Pressure Acid Leach Oxidation**

The cobalt concentrate will be dewatered and fed in a pressurized vessel called an autoclave at an elevated temperature of 180 °C and an operating pressure of 2,200 kPa (kiloPascals). Prior to the autoclave, lignin sulphonate (lignisol), dilution water, and dilute sulphuric acid can be added in order to optimize the feed conditions. The use of lignin sulphonate was required to prevent the formation of elemental sulphur in the autoclave during pilot plant testing. It is a common observation that the use of lignin sulphonate decreases during scale up, and it may not be required for actual operation.

The vessel will have five compartments, divided by baffles, with two agitators in the larger first compartment, and an overall retention time of approximately 60 minutes. Oxygen, supplied by an on-site oxygen plant, will be continually injected into the slurry. The autoclave will oxidize 85% - 90% of the sulphide, producing sulphuric acid, which, in turn, leaches over 95% of the feed cobalt. Because the generation of acid produces heat (exothermic reaction), high pressure water will be injected into the autoclave as cooling water to maintain operation at a constant temperature and under controlled conditions. This condition is called an autothermic operation, where the amount of cooling water and degree of oxidation is specific for the feed rate and constant vessel temperature. At start up, a steam boiler is required to bring the autoclave up to temperature and initiate the oxidation reactions via the injection of steam. At operating conditions, the introduction of water to maintain the temperature

represents a large water demand. Autoclave cooling water must be low in suspended solids, as well as chloride, which can cause significant corrosion to stainless steels at these temperatures.

The generation of sulphuric acid by oxidation leaches cobalt metal from the concentrate, as well as copper and a number of other metals, which report to the solution. Importantly, in the absence of low cobalt levels (<90 ppm), gold remains in the solid state, allowing for solid/liquid separation of aqueous soluble cobalt and copper, and the gold, respectively. In addition, under these conditions, soluble arsenic reacts with iron and oxygen at stoichiometric ratios to form scorodite,  $\text{FeAsO}_4$ , an extremely stable form of arsenate mineral that reports to the solid phase.

After 1 hour of retention time, the leached slurry will be reduced to atmospheric pressure through a letdown choke controlled by a nuclear level gauge into a flash tank at 100 °C. Off-gases resulting from the maintenance of the vessel pressure will be vented from the autoclave and directed to a venturi-type scrubber. Condensate water discharged from the scrubber will be returned to process with the autoclave slurry product. These products will be sent to the cobalt residue thickener, where the solution discharges to a clarifier for fines recovery.

The washed solids discharged from the underflow of the autoclave thickener at 60% solids are subsequently filtered with process washing water to remove all the cobalt in solution to achieve target solid to liquid separation efficiency. Flocculant and the filtrate solution are directed to the feed of the solution clarifier described above.

The filter cake is re-pulped and blended with the flotation cleaner tailings to recover gold by cyanidation as further described in the *Cyanidation* section. The solid residue from the pressure acid leach oxidation circuit represents approximately 3.3% of the feed to the TMA.

In summary, for the two autoclave products following thickening and filtration: the solids will be forwarded to the cyanidation circuit for gold recovery; the purified aqueous solution containing cobalt and copper are distributed to the precipitation circuit for removal of copper.

### **Copper Precipitation**

The cobalt rich autoclave solution will be pumped to the iron-arsenic precipitation stage following clarification. Lime is added to the precipitation tanks to slowly neutralize and gradually bring the solution pH up to 4.6. At that pH, the majority of the residual iron and arsenic remaining in the solution produce a stable ferric arsenate precipitate, along with gypsum, copper and other metals. Slurry from the precipitation tanks overflows to a thickener where the solids are collected. The overflow from the iron/arsenic precipitation thickener is directed downstream for further copper removal.

Sulphuric acid and process water are added to the thickener slurry in two agitated copper re-leach tanks installed in series. The re-leach step is carried out at a pH of 2. At this pH, the acid causes copper to re-leach back into solution, while iron, arsenic, and gypsum remains in the solids as stable precipitates. The re-leach discharge slurry will be filtered on a continuous pressure filter and washed thoroughly. The cake from the filter is sent directly to the tailings thickener. This stream represents approximately 2.5% of the overall tailings.

The filtrate containing the aqueous copper solution is sent to the copper cementation circuit for the recovery of the copper. In cementation, fine iron powder is added into the solution, which will react with the soluble copper and precipitate elemental copper. The discharged slurry is then pumped to a filter press to recover the copper solids. After washing, the filtrate is returned to the iron arsenic precipitation tanks. The copper cement product is collected and packaged for shipment for further refining.

The overflow from the iron/arsenic precipitation thickener will be treated in a copper precipitation circuit to further remove any residual copper. Copper precipitation is performed at 50°C in two agitated tanks where sodium carbonate is continuously added to precipitate the copper as copper carbonate and to

increase the pH to 6.5. At these pH conditions, zinc and nickel precipitate, as well as a small amount of cobalt. The dilute slurry is fed to a pressure filter where the solids are removed, while the filtrate from precipitation, now barren of copper is pumped into the cobalt precipitation area.

No wash is required on the resulting copper carbonate filter cake as it will either be 1) treated for zinc removal and/or (2) re-pulped and recycled back to the iron-arsenic precipitation circuit feed tank. In the case of zinc removal, the cake will be re-pulped with dilute sulphuric acid in order to re-leach the precipitates including zinc, nickel, copper and cobalt. Subsequently, the solution is treated by sodium hydrosulphide (NaHS), which will precipitate the zinc as a metal sulphide, ZnS. Some nickel would be removed at this stage, although the residual soluble copper and cobalt would be re-circulated to the solution collection tank that precedes iron-arsenic precipitation. The filtered mixed metal sulphide precipitate would be filtered and bagged for sale to a refinery as a minor by-product. The unit operation is included in the base case option, and inclusion of the circuit is required for the case of cobalt carbonate sales. If not required, the copper carbonate filter cake will be returned to the solution collection tank preceding iron-arsenic precipitation where the metals will be dissolved in the low pH solution produced by the autoclave. Thus this stream represents a re-circulating load of copper and cobalt.

### **Cobalt Precipitation**

Cobalt in the pregnant (high grade) solution produced by copper precipitation is recovered in a two-stage cobalt precipitation circuit for recovery.

Stage 1 cobalt precipitation includes four agitated precipitation tanks in series discharging to a cobalt precipitation thickener. Sodium carbonate is used to precipitate cobalt carbonate in Stage 1 at an operating temperature of 50 °C and a pH of 7.4. The underflow from the cobalt precipitation thickener, at 31% solids, is partially recycled as precipitation seed and the remaining underflow is centrifuged to remove as much of the liquor as possible. Centrate (centrifuge overflow) is returned to the thickener feed well.

The dewatered centrifuge product can be sold as cobalt carbonate,  $\text{CoCO}_3$ . At this stage, the major impurity is nickel as nickel carbonate, which isn't necessarily a concern for certain potential purchasers. If sold, the cobalt carbonate would be bagged as a preference. Depending on the distance of transport and the preference of the refiner, Figure 5A shows the inclusion of a dryer prior to packaging. The installation of the dryer is not Fortune's preference due to the related energy costs, emissions, and industrial hygiene issues. Instead, Fortune's preferred plan is to upgrade the cobalt carbonate onsite and produce high purity cobalt cathodes. If utilized, cobalt carbonate would be fed into a paddle dryer via a screw conveyor. The paddle dryer would use pressurized steam to indirectly dry the cake to approximately 5% residual moisture, discharging into a storage bin, before being packaged into 1 tonne bulk bags.

Overflow solution from the stage 1 cobalt carbonate precipitation thickener will be pumped to the Stage 2 precipitation circuit. Any residual soluble cobalt will be precipitated with sodium hydroxide (NaOH) at a pH of 9.3. The resulting precipitate that also contains nickel hydroxide is filtered in a pressure filter. No washing would be required. The filtered liquor is depleted of any valuable metals and is pumped to the concentrator process water tank as makeup water. The cobalt hydroxide cake is re-pulped and recycled back to the iron-arsenic precipitation stage for re-dissolution.

The re-dissolution of cobalt utilizing the acidity from the autoclave solution filtrate has been demonstrated to be highly efficient. Due to the presence of nickel impurities in the cobalt hydroxide product of stage 2, nickel concentrations will be artificially elevated in the precipitation trains; however, this is complementary to nickel removal through the zinc sulphide circuit. If the zinc sulphide circuit is not installed, all nickel will need to be removed from the cobalt carbonate product, downstream after re-dissolution by use of ionic exchange resins as discussed below.

### **Cobalt Electro-winning**

The cobalt electro-winning circuit is shown in Figure 5c. The cobalt carbonate precipitate is re-pulped and re-dissolved in a strong aqueous sulphuric acid solution recycled within the electro-winning circuit. Removal of nickel and other trace impurities, such as zinc, will occur by use of an ion exchange (IX) system. Because of this combination, this operation is commonly referred to as *IX/EW*.

The cobalt carbonate will be slowly dissolved, passing through a series of leach tanks. To maintain the valence of the dissolved cobalt cation, dilute hydrogen peroxide can be added. Stripping of the solution with nitrogen to prevent cobalt oxidation can also be undertaken to polish the solution. The solution is fed to the first stage of ionic exchange at ambient temperature. The first stage of ionic exchange will be utilized to remove dissolved nickel using an ionic exchange resin specific for nickel over a series of packed bed reactors or columns. Ionic exchange resins immobilize target metals through surface reaction exchange of hydrogen ion, concentrating those metals, until the resin can be stripped and recycled.

After passing through stage 1, the resulting solution is heated to 60°C and passed through the second stage of ionic exchange. The second stage of ionic exchange will be utilized to remove dissolved zinc from the solution using a series of columns containing ionic exchange resin specific for zinc.

Intermittently, the nickel and trace impurities are stripped from the ionic exchange resin using a strong sulphuric acid solution, and precipitated as a mixed hydroxide using sodium hydroxide, or precipitated as mixed metal sulphides utilizing sodium hydrosulphide. The sodium hydrosulphide process is described in detail in the *Copper Precipitation* section. Because of impurities in either of these two products, the most prevalent impurity being cobalt, the resulting precipitates will be filtered, bagged, and shipped offsite for sale as a concentrate for refining.

The resultant high purity cobalt solution will be processed in a cobalt electro-winning plant to produce cobalt metal as cathodes. Electro-winning involves passing an electrical current through a liquid containing dissolved metals causing the metal to collect on the positive electrode (the cathode), much like electroplating. Acid and spent electrolyte, which contains cobalt, generated during the electro-winning process will be recycled for dissolution of the cobalt carbonate. A small quantity of solution from the electro-winning circuit must be bled off at each pass to maintain the water balance, and for impurity control. The bleed is equivalent to the moisture contained within the cobalt carbonate precipitate entering the circuit and make up concentrated sulphuric acid which is utilized to regenerate the electro-winning solution. The bleed from the circuit is recycled to the iron-arsenic removal stage, where any cobalt can be re-recovered through cobalt precipitation.

Cobalt metal will be stripped from the cathodes on a 4 to 5 day cycle, cleaned and degassed in a furnace prior to shipment.

For stage 1, the proposed resin is XUS-43578, a product of Dow(R) Chemical Company. XUS-43578 is described as a developmental chelating resin containing Bis-picolylamine functionalized, chloromethylated polymer of styrene and divinylbenzene. It can be thought of as a plastic bead. A more commonly known form of this resin is known as Dowex(R) M4195, although there are slight differences between the two products with respect to size distribution and shape.

For stage 2, the proposed resin is Lewatit®<sup>3</sup> VP OC 1026. This resin is described<sup>4</sup> as a macroporous resin which contains Di-2-ethylhexyl-phosphat (D2EHPA), and is directly incorporated into the resin matrix during formation of the copolymer.

Other selections of resins may be undertaken due to the results of pilot plant optimization, or due to future innovation in resin technology.

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<sup>3</sup> Lewatit is a registered trademark of Sybron Chemicals Inc., a subsidiary of Lanxess.

<sup>4</sup> From product Information specification sheet.

Manganese sulphate will be utilized as a reagent for the maintenance of the anodes within the circuit, preventing corrosion. While NICO ores contain manganese, and manganese will generally follow the path of cobalt through the circuit; because of start up, intermittent bleeding, loss of manganese to the ionic exchange resins, and variability of manganese concentration in the ore, manganese sulphate will be stored on site to maintain the concentration within the electro-winning circuit at target levels.

### **Gold Recovery**

Gold in the bismuth concentrate, which will contain over 50% of the gold in the plant feed, will be recovered in one of four 72-hour batch cyanidation tanks, where each tank contains 24 hours of capacity. The tanks will be operated so that while one tank is being filled or emptied, the other tanks will be at various stages of recovery at 24-hour increments. Spent electrolyte from the downstream high grade gold electro-winning circuit is used to re-pulp the slurry to 33% solids. Sodium cyanide and lime are added to the tank to maintain 2-5 g/L of cyanide in the liquor and a pH of approximately 10.5. A fan is provided on each cyanidation tank to introduce a mixture of air and oxygen gas continuously for the cyanidation reactions. At the completion of cyanidation, the slurry will be pumped from the tank to a pressure filter for dewatering. After a three displacement wash on the filter, the cake will be discharged into the re-pulp tank. Because the liquor from bismuth concentrate cyanidation is very rich in gold, the re-pulp tank is included to provide a means of achieving very high wash efficiency on the cake. The cake will be re-pulped with cyanide-rich water to 33% and will be re-filtered and re-washed. This repulp-refilter-rewash cycle will continue until the filtrates contain minimal amounts of gold. The well-washed, gold-barren bismuth concentrate cake is transferred to a storage bin containing 48 hours of surge capacity. Figure 5a shows that the resulting concentrate will be packaged for shipment to a refinery in 1 tonne bulk bags, or transferred to the chloride leach circuit. The pregnant (high grade) leach solution and the wash liquors will be directed to the high-grade gold electro-winning circuit, or to the CIP circuit, depending on the grade in solution.

The bulk cleaner flotation tailings and solid residue from the pressure acid leach oxidation circuits will be recovered in a 6 stage carbon-in-pulp (CIP) circuit for the recovery of gold. Washed and filtered autoclave residue will be re-pulped and combined with the thickened bulk cleaner tailings before pH adjustment using lime to a pH of 10.5. Sodium cyanide, oxygen gas and/or air will be added to the leach tanks continuously to transfer the gold into the slurry solution for recovery onto carbon.

Cyanidation discharge slurry will flow into a 6-stage CIP circuit providing a total of 24 hours of residence time. Fresh activated carbon is added to the last tank of the CIP circuit, and the carbon is periodically advanced forward toward the first tank. The gold in the cyanide solution is loaded onto the active carbon in these tanks, decreasing in concentration through the six stages. Thus, the slurry containing gold in solution is moved counter current to the activated carbon, which is transferred intermittently.

Carbon loaded with gold will be removed from the first tank and passed over a carbon screen and washed, where the undersize is recycled back to the first CIP tank. Carbon will be removed at a rate of about 0.5 tonnes per day and advanced to a carbon strip, or elution, circuit. After washing with nitric acid ( $\text{HNO}_3$ ), the carbon will be stripped of gold with a weak sodium hydroxide-sodium cyanide solution by a batch process in a pressure vessel at 480 kPa and 130°C. The stripping of the gold works in a closed circuit with gold electro-winning where the gold is deposited. The stripping solution contains 2% sodium hydroxide, and approximately 0.2% sodium cyanide. The solution is heated by heated exchange with the pregnant eluate, and then further heated to reach the operating temperature via another heat exchanger using steam. The eluate solution is passed through the carbon in the stripping vessels, and the gold concentrated on the carbon is returned to the strip solution at significantly higher concentrations than present in the CIP circuit. This loaded solution is directed to gold electro-winning. At the termination of the process, the vessel is cooled and stripped carbon is sent to the carbon reactivation process.

In order to prevent fouling of the carbon (impurities which slow down the rate of gold adsorption), the carbon is reactivated in a kiln operating at a temperature of 760°C. Kiln off-gasses will be sent to a scrubber to be cleaned with water before being emitted to the atmosphere. After reactivation, water is

added back to the carbon and pumped to a sizing screen. Carbon fines will be collected and packaged in bags while the rest of the reactivated carbon is pumped back to the CIP tanks for reuse. After every few batches, the loaded carbon needs to be washed prior to stripping. Loaded carbon from the CIP circuit is pumped to an acid wash vessel where either diluted hydrochloric or nitric acid is used to wash the loaded carbon before it is sent to the stripping vessel to undergo stripping. Acid wash solution is recycled back to the acid wash tank to be re-used for the next wash.

Both gold solutions from the bismuth cyanidation circuit (high grade) and the pregnant solution from carbon stripping (lower grade) will be fed to electro-winning cells. Spent electrolyte from the electro-winning is recycled back to the barren solution tank for carbon stripping, or used as re-pulp water in the bismuth cyanidation circuit. Gold removed from the stainless steel cathode material as a sludge that is filtered in a pressure filter, dried, mixed with a weighed amount of flux, and melted in an induction crucible furnace, before being poured into gold doré. Furnace off-gasses will be sent to a scrubber to be cleaned with water before being emitted to the atmosphere.

### **Cyanide Destruction**

The CIP circuit discharge slurry, or tailings, will be passed over a safety screen to prevent carbon losses. One option under consideration is to utilize a vacuum drum filter on the CIP tailings to recycle cyanide solutions within the CIP circuit to minimize reagent consumption and fresh water use. In this case, the filter cake would be re-pulped with reclaimed water from the tailings pond. The unfiltered CIP tailings, or the re-pulped filter cake from filtration of the CIP tailings, will be treated in a cyanide destruction circuit. The decomposition reaction is referred to as the sulphur dioxide-air process, often referred to as the *Inco SO<sub>2</sub>-air process* (see Chapter 6, *Applicable Technologies for the Management of Mining Effluents in the Northwest Territories, Department of Indian and Northern Affairs Canada* by Lakefield Research in association with Senes Consultants Limited, April 2002).

This process uses sulphur dioxide in the presence of air in a single reaction tank providing 60 minutes of retention time. The sulphur dioxide required is produced using the reagents sodium metabisulphite and sulphuric acid with copper sulphate as a catalyst. Copper sulphate requirements will be met by using internally produced copper, when available. While the use of the Inco process has been demonstrated in the NICO process, Fortune is also evaluating the CombinOx® process, where the use of Caro's acid in combination with SO<sub>2</sub>-Air may provide faster destruction kinetics and a shorter retention time, resulting in reduced capital and operating costs. An advantage of this process is that copper sulphate required to catalyze the process can be greatly reduced, as well as the "fuel" for the process, metabisulphite. Caro's Acid, H<sub>2</sub>SO<sub>5</sub>, is produced by the combination of hydrogen peroxide and sulphuric acid at the location of addition.

### **Bismuth Recovery by Chloride Leach Electro-recovery**

Surge capacity representing 4 days of production of bismuth concentrate cyanide leach residue will be constructed in order to allow for intermittent maintenance on the chloride leach electro-recovery facilities. Because of the use of hypersaline chloride concentrations and sulphuric acid, corrosion of the equipment and instrumentation will be a constant concern. This capacity represents about 80 tonnes of contained concentrate.

The bismuth recovery flow sheet is shown in Figure 5c. The recovery of bismuth to a high purity metal cathode is an innovative process. The process was developed in conjunction with SGS Lakefield and other contributors. It has been run at a pilot scale. Fortune is presently evaluating patent protection on certain components of the recovery process. Fortune refers to the process as *chloride leach electro-recovery* (CLER).

Bismuth concentrate resulting from cyanidation and filtration will be leached counter-currently in two stages using a concentrated chloride solution. In the second stage of leaching, fresh make-up chloride solution, produced through the mixing of sulphuric acid and salt, sodium chloride (NaCl) is contacted with

partially leached residue from the first stage. The counter current leaching optimizes reagent use and maximizes bismuth recovery. The leached/washed slurry from the second stage is filtered and washed with the filtrate transferred onto the first stage of leaching. The washed residue in the second stage is discharged as a waste stream.

The resulting medium grade bismuth solution is utilized in the first stage leach to maximize the concentration of bismuth in solution by leaching bismuth in the feed concentrate. After liquid/solid separation by filtration, the bismuth rich solution from the first stage leach circuit filtrate is fed to a modified electro-winning circuit for the production of high purity bismuth cathodes. Purity of these cathodes has been demonstrated to exceed 99% on a consistent basis. The modified electro-winning cells incorporate specific technology to prevent corrosion of the anodes in the sulphuric acid/chloride solution, as well as other design modifications that differ from typical electro-winning cells that are critical to the success. For these reasons, Fortune refers to this unit operation as *electro-recovery*. This unit operation also regenerates the lixiviant<sup>5</sup> hence avoiding the use/generation of chlorine gas, and without relying on expensive scrubbers. This is one of the most attractive features of the Fortune CLER process for the recovery of bismuth

The filter cake residue from the filtration of the first stage leach is recycled onto the second leach reactor where the remaining bismuth is leached as described earlier.

Spent electro-winning solution is recycled to the second stage of leaching where reagent make up occurs by adding sulphuric acid and sodium chloride. Chloride levels are maintained above 200 grams per litre chloride. To maintain the water balance within the counter current, closed circuit system, bleed from the primary electro-recovery stage is removed from the circuit to compensate for water entering the circuit through reagent mixing and contained within the bismuth concentrate. This bleed stream is directed to a secondary stage electro-recovery circuit based on the same design principles as the first stage circuit. Soluble bismuth levels in the bleed liquor are reduced through the electro-winning of the lower grade solution and producing a bismuth cathode with a purity exceeding 90%. Substantially fewer cathodes are produced through the secondary bleed electro-recovery circuit than from the primary circuit that is treating high grade liquor.

An advantage of this process, which is similar to the ferric chloride leaching process for the recovery of bismuth, is that iron does not have to be regenerated as ferric iron through the use of either chlorine or oxygen gas. In addition, due to the high iron levels in the leach solution, chlorine gas is not produced at the anode of the electro-recovery cells.

Following secondary electro-recovery, the spent solution containing very low levels of bismuth are subjected to scavenger recovery by cementation of the residual bismuth. This process involves contacting the spent liquor by a two stage process (to conserve reagents) with an iron powder that precipitates, or cements, the remaining bismuth as a crude metal product. The bismuth is separated from the solution by filtration and becomes the final by-product, which will be bagged in concentrate bags and sold on for refining by a third party. Most of the barren solution is bled from the circuit for volume control, although a portion can be recycled to leaching stage during reagent make-up to maintain the lixiviant balance.

The chloride leach electro-recovery circuit will produce three waste streams: a solid leach residue, a solution bleed, and an iron-arsenic hydroxide precipitate. Residual concentrations of iron and arsenic in the bleed solution will be removed by the addition of oxygen to precipitate iron arsenate at a pH of 4.6 (buffered by lime). The resulting precipitate will be filtered along with gypsum produced through the neutralization of the sulphuric acid, and blended with the tailings thickener underflow. Filtrate from the iron-arsenic precipitation stage will be returned to process water tank. This solution bleed will contain chloride and a very small portion of the soluble arsenic. The leach residue stream will be well washed and contain minor concentrations of residual soluble chloride. The solid residue from the recovery of the

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<sup>5</sup> lixiviant is the leaching agent

bismuth concentrate represents 0.24% of the feed to the TMA.

Fortune is investigating the use of Electrodialysis Reversal™ (“EDR”) treatment technology developed by General Electric on the chloride-containing bleed stream as part of the ongoing pilot testing of the CLER circuit. EDR is a process developed for the desalination of water that removes ionic species using a combination of direct current (DC) power and ionic exchange (IX) membrane technology. EDR can be potentially optimized for chloride recovery, rejecting ions that are not useful to the CLER, recycle chloride to the leach circuit, reduce sodium chloride consumption, and discharge chloride at controllable concentration levels. The potential impact of integrating this circuit into the flow sheet is discussed further in section 4.14.

The hydrometallurgical production of bismuth on-site is viewed as strategically, environmentally, and economically advantageous to roasting, or the wholesale shipment of the concentrate offsite for smelting. In the event that chloride leach circuit cannot be built at NICO, a filtration and packaging facility will be constructed for the transport of bismuth concentrate off site for further upgrading. This option is demonstrated by Figure 5a.

### **Reprocessing of Chloride Leach Electro-Recovery Residue**

Fortune is presently evaluating the treatment of the residue from chloride leaching in the autoclave of the pressure acid leach oxidation circuit as previously described.

This residue makes up a very small proportion of the overall tailings and a small proportion of the overall feed to the autoclave; however, the benefits of recycle could be improved stability of the residue, with concurrent recovery of bismuth concentrate cobalt (2% of the total feed) and copper values. Additionally, further oxidation may allow for recovery of the balance of the gold from the bismuth concentrate cyanidation stage, which for the first two years is approximately 1 oz per tonne.

Alternatively, for the first two years, the chloride leach residue will be packaged and shipped to a smelter for return of the gold values.

To be successfully implemented, the wash efficiency of filtration of the residue will need to maintain chloride levels less than around 100 ppm. Above this level, gold values report to solution, detrimental to recovery of gold from the autoclave residue. In addition, the chloride residue cannot be detrimental to the overall recovery of cobalt from the cobalt concentrate or the cyanide destruction circuit treating the CIP tailings.

### **Alternative - Copper Solvent Extraction and Electro-winning**

Fortune is evaluating the use of copper solvent extraction and electro-winning as an alternative to the base case copper precipitation presented. Copper solvent extraction and electro-winning is commonly referred to as *SX/EW*. The two flow sheets are presented on Figure 5d.

Advantages of copper *SX/EW* include reduced lime, sodium carbonate, and sulphuric acid consumption, as well as the production of an LME (London Metals Exchange) grade copper cathode that can be sold on the spot market, instead of dependence on an offsite refinery to realize a revenue stream from copper.

Disadvantages of copper *SX/EW* include increased capital cost, increased power utilization, and introduction of kerosene and proprietary organic reagents.

On an overall basis, copper *SX/EW* appears to provide for improved operating cost, reducing the unit cost per tonne of copper removed, which is necessary to produce saleable cobalt carbonate and cobalt cathodes. The operating cost advantage applies for operating environments where the price of power is less than \$0.85 per kWh, regardless of sales revenue. LME grades of copper cathode would sell at a significant premium to the copper cement produced in the base case.

The proposed process is shown in Figure 5d. The cobalt rich autoclave solution will be pumped to the pre-neutralization stage following clarification. Lime is added to the precipitation tanks to slowly neutralize and gradually bring the solution pH up to a pH of 2.5. At that pH, calcium arsenate is precipitated. The resulting precipitates are filtered to produce a clean filtrate for feed to the copper extraction circuit. Because calcium arsenate is not particularly stable, the filter cake will be stabilized upstream of the copper extraction circuit, or alternatively returned to the autoclave feed.

Clean filtrate containing copper as well as cobalt, sometimes referred to as the pregnant *liquor* is contacted with an organic solvent referred to as the *extractant* in the copper solvent extraction stage. In copper extraction, copper is removed from the aqueous phase selectively over cobalt or other metals, leaving behind most of the impurities in the liquor. The resulting aqueous solution is sometimes called the *raffinate*. During the extraction, copper cations in the liquor are exchanged for hydrogen ions in the extractant, resulting in an increased acidity, or lower pH, in the raffinate. The extractant, which is miscible in organic liquids, is transported to and from the extraction phase by use of a reagent referred to as the *diluent*. The loaded solvent consisting of the extractant and the diluent, which is immiscible in the aqueous liquor, is collected and transferred onto the scrubbing stage.

The loaded copper solvent is stripped of its copper by contacting it with a strongly acidified aqueous sulphuric acid solution. In the scrubbing operation, copper is moved back to the aqueous phase, while the organic phase is returned to the extractant form having the hydrogen ion that is exchanged. Thus, following scrubbing, the stripped solvent can be returned to the extraction stage to recover more copper. Make-up hydrogen is supplied by the sulphuric acid required to regenerate the strongly acidified scrubbing solution.

The concentrated copper-bearing loaded aqueous solution is advanced to the electro-winning (EW) stage. In the electro-winning stage of the process, the soluble copper cation is reduced to a metallic copper cathode. The electrochemical production of copper cathode requires access to stable continuous power. The resulting cathodes produced are extremely pure, and easily sold on under different grade categories determined by the LME.

Losses of organic are extremely minimal, and well controlled by the appropriate design of stages with sufficient retention time to adequately separate the solvent (top phase), which is immiscible in the aqueous (bottom phase). It is proposed that kerosene be utilized as the diluent. While there are a number of extractants of proprietary composition distributed by various manufacturers to the copper SX/EW industry, Fortune proposes to utilize ACORGA®<sup>6</sup> M5774 solvent extraction reagent manufactured by CYTEC Industries. M5774 is composed of 5-Nonyl-2-hydroxy-benzaldoxime at 30-60% concentration as well light hydrotreated petroleum distillate at 7-13%.

In addition, the use of GUARTEC®<sup>7</sup> EW is proposed as an agent in the strongly acidified aqueous phase of the electro-winning circuit. GUARTEC EW is a derivative of Galactomannon polymers derived from guar beans and improves the copper tensile strength and elongation at very low levels<sup>8</sup>. This use is sometimes referred to as *smoothing*. The use of cobalt sulphate monohydrate protects the anodes from corrosion in the highly acidified environment.

Raffinate from the extraction circuit is subsequently mixed with the calcium arsenate precipitate from pre-neutralization. The resulting solution is then fed to an iron-arsenic precipitation stage, where the solution pH is slowly increased to pH 4.6 using lime. This pH was chosen as under equilibrium conditions, cobalt will not precipitate. Stable iron arsenic hydroxide precipitates are subsequently formed preferentially to calcium arsenate precipitates, and the resulting product is thickened, filtered, washed and discharged to tailings.

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<sup>6</sup> ACORGA® is a registered trademark of CYTEC Industries.

<sup>7</sup> GUARTEC is a registered trademark of Cognis Corporation.

<sup>8</sup> From the specification sheet.

Over flow from the iron-arsenic precipitation thickener is directed to an aluminum-silica precipitation stage where aluminum and silica are removed. The pH chosen will be determined based on a balance of minimizing cobalt loss, while maximizing aluminum and silica removal. Ideally pilot work will demonstrate that this stage is not required, and can be eliminated. In the base case, aluminum and silica is removed through copper precipitation, and eventually reports to the ferric arsenate tailings.

The balance of the circuit outlines the identical cobalt precipitation stages described as the base case. In the absence of a zinc metal sulphide precipitation stage, zinc removal by ionic exchange will be incorporated prior to, or following cobalt carbonate production. Zinc removal by ionic exchange is discussed in the *Cobalt Electro-winning* section.

## Reagents

Oxygen is supplied to the autoclave and other circuits by way of a vacuum pressure swing adsorption plant (VPSA) at a purity of at least 93%. The location of the oxygen plant has been selected to reduce noise on the camp facility. The operation and maintenance of an oxygen plant requires certain controls to prevent exposure to nitrogen environments that displace oxygen and are a risk to human health. The oxygen plant will have a design capacity of 100 tonnes per day oxygen.

Lime will be delivered to the process plant as pebbled hot lime, CaO, delivered in bulk and transferred by pneumatic conveying into the lime silo. The lime silo will be constructed with a slaking plant, which consists of a small ball mill to grind the lime, and dissolve the hydroxide. Lime will be distributed throughout the plant by a ring main to the cyanidation, precipitation, and bismuth chloride leach electro-recovery circuits.

Sodium carbonate will be delivered to the plant in bulk as a powder and transferred by pneumatic conveying into the soda ash silo.

Sulphuric acid will be delivered to the plant in bulk as a concentrated 98% solution. Because of the difficulty in pumping sulphuric acid at lower temperatures, the sulphuric acid storage tanks will be constructed within the mill building itself.

Bulk reagents will be delivered in a combination of bulk bags for solids, and totes and barrels for liquids. Reagents will be stored on a reagent storage area lined with a geomembrane to contain any spills. The reagent storage area will be fenced. Certain reagents that cannot be stored outside will be stored in the cold storage warehouse. All products produced by the NICO process will be stored in the cold storage warehouse prior to shipping. These locations are shown in Figure 4a.

## Reclaim Water

To maximize the use of reclaim water from the TMA, a reclaim barge will be constructed to return the decanted water to the plant for use in process. Operation during the winter months will be achieved through use of air bubble generation or water jets to maintain the water turbulence around the barge and to prevent freezing.

### **4.2 If milling is in progress on the property at the present time, indicate the rate of milling.**

Not applicable. No milling is currently being carried out on site.

### **4.3 What is the present (or proposed) maximum capacity of the mill?**

The average capacity of the mill will be 4,650 dry tonnes/day, with a maximum capacity of 5,160 dry tonnes/day. Annually, this equates to an average of 1,695,060 dry tonnes/year.

#### 4.4 List the types and quantities of all reagents used in the mill process (in kg/tonne ore milled).

**Table 4.4-1 Reagents and Quantities expected to be Utilized by NICO Mine**

Reagent	Consumption Rate	
	(kg/t)	(t/yr)
Steel grinding rods	0.537	910
Steel grinding balls	1.327	2,250
Flocculant	0.027	45
Lime (CaO)	9.8	16,625
Iron Powder	0.354	600
Sodium Hydrosulphide (NaHS)	0.083	140
Sodium Carbonate (Na <sub>2</sub> CO <sub>3</sub> )	2.8	4,775
Sulphuric Acid (H <sub>2</sub> SO <sub>4</sub> )	2.8	4,775
Nitric Acid (HNO <sub>3</sub> )	0.007	12
Hydrochloric Acid (HCl)	0.007	12
Sodium Cyanide (NaCN)	0.236	400
Sodium Hydroxide (NaOH)	0.236	400
Sodium Chloride (NaCl)	0.088	150
Potassium Amyl Xanthate (PAX)	0.260	440
Methylisobutyl Carbinol (MIBC)	0.053	90
Lignin Sulphonate	0.198	336
Diatomaceous Earth	0.027	45
Refinery Flux	0.002	2.9
Sodium Metabisulphite (Na <sub>2</sub> S <sub>2</sub> O <sub>5</sub> )	0.944	1,600
Copper Sulphate (CuSO <sub>4</sub> )	0.027	45
Manganese Sulphate (MnSO <sub>4</sub> )	<0.059	<100
Activated Carbon	0.007	12
Oxygen gas	21.5	36.500
		<b>(m<sup>3</sup>/yr)</b>
Lewatit® VP OC 1026		3.5
Dow XUS-43578		3.5
<b>Seasonal Storage</b>		<b>(t/yr)</b>
Ferric Sulphate (Fe <sub>2</sub> (SO <sub>4</sub> ) <sub>3</sub> )	0.236	400
Lime, Hydrated (Ca(OH) <sub>2</sub> )	0.472	800
<b>Copper SX/EW Option</b>		
Flocculant	-0.001	-2
Lime (CaO)	-0.767	-1,300
Sulphuric Acid (H <sub>2</sub> SO <sub>4</sub> )	-1.357	-2,300
Iron Powder	-0.251	-425
Sodium Carbonate (Na <sub>2</sub> CO <sub>3</sub> )	-0.265	-450
Diatomaceous Earth/Clay	-0.014	-24
Guartec® EW, or similar	0.001	+1
Cobalt Sulphate Monohydrate (CoSO <sub>4</sub> ·H <sub>2</sub> O)	<0.024	+ <40
ACORGA® M5774	0.0001	+0.2
Kerosene	0.004	+7

Notes: kg/t ore = kilograms per dry tonne of ore; t/yr = dry tonnes per year; m<sup>3</sup>/yr = cubic metres per year

Refinery flux, is a mixture of specific dry compounds that act on processing products such as gold concentrates and electro-winning sludge in order to fuse (or smelt) those products at lower temperatures than normally achievable without them. Flux mixtures are generally customized for a specific mine's requirements and will consist of borax (Na<sub>2</sub>B<sub>4</sub>O<sub>7</sub>), soda ash (Na<sub>2</sub>CO<sub>3</sub>), silica (SiO<sub>2</sub>) and sodium or potassium nitrate (NaNO<sub>3</sub> or KNO<sub>3</sub>).

Similarly, the mine will operate an on-site assay laboratory to generate assays of pit and underground samples, as well as processing products. In addition to the fluxing reagents listed above, the following will be utilized, ideally as a pre-mixed mixture: litharge (PbO), fluorospar (CaF<sub>2</sub>) and domestic flour.

#### **4.5 Is the (proposed) milling circuit based on autogenous grinding?**

No. Autogenous grinding was determined unsuitable for the NICO plant due to the hardness of the typical ore and significant variability within the deposit. Instead, the cone crushers (secondary/tertiary crushing) and a rod mill will be utilized.

#### **4.6 Indicate the amount(s) of concentrate(s) produced in the mill.**

The following products will be produced at the mill<sup>9</sup>:

Cobalt metal	1,500 to 1,975 tonnes/yr
Gold doré	22,000 to 89,500 ounces per year (oz/yr)
Bismuth metal	1,150 to 2,675 tonnes/yr

Copper will represent a significant by-product, averaging 600 tonnes per year. On a daily basis, copper by-product production may vary significantly, from 1.0 to 4.3 tonnes/day produced.

Other minor by-products may include zinc and nickel. Recovery and sale of the by-product would be undertaken to avoid discharge to the TMA and offset the costs of treatment. The most likely form as a saleable concentrate will be as a mixed metal sulphide that will include some of the residual cobalt.

The residue from chloride leach electro-recovery circuit will contain elevated concentrations of gold averaging 1 troy ounce per tonne during the first two years of operation. At these levels, the residue will likely be saleable as a concentrate to a smelter. Production of the residue would be approximately 15 tonnes per day, and could be shipped from site in concentrate bags. After year 2, the grade of this residue would fall to 8.0 grams per tonne (0.25 oz/t), and at the lower grade, would probably preclude further off-site processing. Alternatively, these values may be recoverable by augmenting the autoclave feed and reprocessing the residue.

#### **4.7 Will fresh water undergo treatment prior to use in the mill process? Explain.**

Fresh water will be screened at the intakes on Lou Lake and Burke Lake, but no further chemical treatment is contemplated for use as process water.

Fresh water utilized for steam in the autoclave and site heating boilers will require softening through use of the related water softening reagents, which are not significant, and are often proprietary.

#### **4.8 Indicate all uses of water in the mill. Include the quantity and source of the water for each use.**

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<sup>9</sup> The estimate provided is not based on a N43-101 compliant model. Instead the values provided by the 43-101 compliant study of February 2007 were prorated on the basis of a daily throughput increase of 4,650 dtpd, versus the 4,000 dtpd utilized for that study. To be 43-101 compliant, the values presented above must be evaluated against the increased capital cost and the reduced unit operating costs, a reserve re-calculated, and the production model published against the known process recoveries for these metals.

**Table 4.8-1 Estimated Water Quantity and Source Usages for Mill/Plant**

Source	Use	Volume	
		<sup>3</sup> (m /day)	L/min
Lou and Burke Lakes	Pumped into holding tank during continuous operation (detailed below)	6,120	4,250
	Concentrator make-up water tank	2,583	
	Hydrometallurgical process water tank	1,763	
	Boilers and hot water heating	1,156	
	Wash water	300	
	Reagent preparation	289	
	Gland water	29	
Tailings thickener overflow and minor process flows	Recycled water	5,160	3,583
Hydrometallurgical water recycled		2,640	1,833
	<b>Total</b>	13,920	9,666

Notes: m<sup>3</sup>/day = cubic metres per day; L/min = litres per minute

The total water requirement for the operation is 13,920 m<sup>3</sup>/day of which 7,800 m<sup>3</sup>/day will be recycled back into the process from the tailings thickener overflow and the hydrometallurgical processes. Water recycled in the plant will be augmented by reclaim water from the TMA.

Conservatively assuming no recycling of tailings pond water, the net intake from Lou and Burke Lakes for process demands is 6,120 m<sup>3</sup>/day.

Successful long term use of reclaim water will depend on the response of the flotation circuit where residual cyanide is detrimental to sulphide recovery, and in the hydrometallurgical circuit, where chloride may be detrimental due to corrosion and gold recovery. Use of reclaim water is also determined by maintenance of the pH at neutral conditions. Because of other inputs to the TMA such as runoff, process water will be diluted in the TMA by a ratio of 3.4 to 1 which indicates that successful reclaim of process water will be possible.

**4.9 Indicate the total volume of water discharged from the mill.**

The estimated total discharge from mill/plant is 3,096 m<sup>3</sup>/day (2,150 litres per minute). This includes about 1,935 m<sup>3</sup>/day water decanted and 1,161 m<sup>3</sup>/day entrained in the tailings (assuming the tailings consolidates to 80% solids).

**4.10 Of the preceding volume, what quantity is (will be) recycled to other areas on the property (mine, mill, etc)? Indicate location of use and quantity.**

A tailings thickener was specifically introduced into the plant design to promote water recycling within the plant and minimize solution discharged to the tailings pond.

Eventually, a pond of tailings water will develop as water within the tailings slurry separates from the tailings solids. In order to be conservative, the water balance assumes that none of the tailings pond water will be recycled to the mill/plant. This will be the situation for start up, although after a period of operation, the tailings will consolidate and water will be able to be returned to the process by installation of a reclaim barge and pump within the pond of the TMA.

**4.11 Based on yearly production; indicate the average quantity of tailings (dry weight) discharged from the mill.**

1.7 million t/yr.

The tailings will consist of the following components (reported in percent):

Bulk Rougher Tailings	85.7;
Bulk Cleaner Tailings	8.3;
Cobalt leach residue	3.3;
Iron-Arsenic Precipitate from Copper Recovery	2.5; and,
Bismuth Leach Residue	0.24.

Because the bulk cleaner tailings and the cobalt leach residue are combined for cyanidation, the cyanide destruction circuit residue will represent 11.5% of the tailings.

**4.12 What is the average liquid:solid ratio of tailings leaving the mill?**

By weight 60% solids and 40% solution (ratio = 60:40)

**4.13 If applicable, identify any chemical treatment applied to the liquid phase before being discharged to the tailings area. (Attach flow sheet if available.)**

Treatments directed at certain deleterious substances are incorporated into specific unit operations of the flow sheet upstream of the tailings thickener. These include:

- SO<sub>2</sub>-air cyanide destruction of the carbon-in-pulp CIP tailings;
- Byproduct production of copper, nickel and zinc from the solution phase generated by pressure acid leaching in the autoclave; and,
- iron/arsenic hydroxide precipitation in bleed streams from the cobalt and bismuth recovery circuits.

The tailings thickener and cobalt precipitation thickener will be used to extract water for recycling as process water. The tailings decant will be collected and recycled into the process to the extent possible. Excess tailings water will be treated at the effluent treatment facility (ETF) to meet MMER Criteria, as discussed in a subsequent section.

**4.14 Based on present production or bench test results, describe the chemical and physical characteristics of liquid mill wastes directed to the tailings area.**

The supernatant from three types of tailings were analyzed to assess the potential concentrations of various parameters of liquid fraction of the mill effluent. As described in Section 2.5, the tailings characterised were:

- aged bulk flotation tailings;
- cyanide deconstruction composite; and
- co-mingled composite.

The co-mingled composite tailings are expected to be typical of the tailings to be deposited; therefore, the supernatant from it is considered to be most representative of the likely tailings decant, without consideration of surface runoff, infiltration or evaporation. The approximate proportions of components of the co-mingled composite tailings include 68% aged flotation tailings (at a ratio of 50% bulk thickener underflow solution, and 18% thickener underflow solids), 17% cobalt precipitate thickener overflow solution, 9% CND composite tailings (overall: 5% liquid and 4% solid contribution), and 6% iron-arsenic precipitate rich residue from the cobalt recovery circuit (overall: 5% liquid, <1% solid contribution).

Geochemical test results from 120 day tests suggest that all MMER controlled parameters, with the exception of arsenic and copper, were below the respective MMER discharge limits (Table 4.14-1). The decant water, however, will not be released into the environment, but will be directed to the ETF or will be recycled into the process. The ETF, described later in this application, will reduce concentrations to levels acceptable under MMER.

**Table 4.14-1 Chemical and Physical Characteristics of Mill Effluent based on 120 Day Bench Tests**

Parameter	Strong Acid Digest	Tailings Supernatant	MMER Criteria
<b>General Parameters</b>			
Conductivity		6,030 µS/cm	-
pH		8.27	6.0 to 9.5
Redox Potential, Eh		255 mV	-
	<b>(mg/kg)</b>	<b>(mg/L)</b>	<b>(mg/L)</b>
Acidity as CaCO <sub>3</sub>		< 2	-
Alkalinity as CaCO <sub>3</sub>		166	-
Hardness as CaCO <sub>3</sub>		1,540	-
Total Suspended Solids, TSS		14	15
	<b>(mg/kg)</b>	<b>(mg/L)</b>	<b>(mg/L)</b>
<b>Nutrients</b>			
Total Ammonia as N		11.8	-
Un-ionized Ammonia <sup>a</sup>		0.3-1.0	
Chloride, Cl		41	-
Nitrite as N		9.41	-
Nitrate as N		0.26	-
Sulphate, SO <sub>4</sub>		3,900	-
	<b>(mg/kg)</b>	<b>(mg/L)</b>	<b>(mg/L)</b>
<b>Total Metals</b>			
Aluminum, Al	29,000	0.27	-
Arsenic, As	19,000	3.69	0.5
Cadmium, Cd	0.78	0.0005	-
Chromium, Cr	76	0.001	-
Cobalt	690	1.12	-
Copper, Cu	280	0.0419	0.3
Iron, Fe	230,000	1.54	-
Lead, Pb	37	0.0006	0.2
Mercury, Hg	<0.1	<0.0001	-
Molybdenum, Mo	18	0.402	-
Nickel, Ni	34	0.049	0.5
Selenium, Se	<5	0.2	-
Silver, Ag	0.98	0.0001	-
Thallium, Tl	0.6	<0.0002	-
Zinc, Zn	100	0.012	0.5
		<b>(Bq/L)</b>	<b>(Bq/L)</b>
<b>Radionuclides</b>			
Radium-226, <sup>226</sup> Ra		0.05	0.37
		<b>(mg/L)</b>	<b>(mg/L)</b>
<b>Others</b>			
Total Cyanide, CN		0.06	1

Notes: MMER = Metal Mining Effluent Regulations, Canada Gazette Part II, Vol. 136, No. 13. SOR/DORS/2002-222;

mg/kg = milligrams per kilogram; mg/L = milligrams per litre; mV = millivolts; µS/cm = microSiemens per centimetre; Bq/L = Becquerels per litre.

<sup>a</sup>Un-ionized ammonia is dependent on total ammonia concentration and temperature and pH conditions (higher with both). The calculated un-ionized ammonia concentration shown is based on the pH measured and assumed reasonably conservative temperatures.

Although this water is not proposed to be discharged directly to the environment, these results have been referenced to the MMER Criteria for context. Table 4.14-1 presents the highest result for each parameter over the test period (*i.e.* samples were analyzed from days 1, 15, 30, 45, 59, 90 and 120), as well as the results of a strong acid digest of the tailings solids to evaluate the head grade for comparison.

The supernatant analysis, above, does not include tailings from the bismuth metal process that was not contemplated at the time. This process, which is under development and described in Figure 5c), will be characterized from the pilot plant scale work. The residue from the proposed bismuth leach process would represent less than 0.5% of the overall plant feed, and producing representative material requires a significant effort and a substantial amount of pilot plant feed.

Preliminary calculations suggest that the impact of bismuth metal recovery through use of the chloride leach electro-recovery process will only noticeably affect the chloride concentration of the TMA solution, potentially raising concentrations as much as 10 mg/L through the bleed from the circuit without any treatment. Because chloride concentrations from minor minerals in the ore fed to the plant contribute to the overall solution chloride balance within the process, and determine the recycle versus treatment ratio (“RVT ratio”) of decanted water from the TMA (higher levels represent a constraint for use in the autoclave due to corrosion), Fortune will be investigating the use of Electrodialysis Reversal™ (“EDR”) treatment technology developed by General Electric on the bleed stream. EDR is a process developed for the desalination of water that removes ionic species using a combination of direct current (DC) power and ionic exchange (IX) membrane technology. EDR can be optimized for chloride recovery rejecting cations that are not useful to the CLER, and discharge chloride at controllable concentration levels. The potential of a modified form of EDR (as the Fortune’s goal is not to treat to drinking water standards) is that it may increase the RVT of the TMA, and the resulting brine containing chloride can be recycled to the second stage CLER leach reactor, reducing reagent consumption.

Whether EDR is utilized or not, and if so used, the extent and removal efficiency, will consider the significance and economics of the chloride impact and the use of recycled decant water. The continual recycling of decant water will increase the chloride levels in the TMA over time, which is balanced by dilution from non-process inputs into the TMA, as well as the release rate of chloride from the CLER circuit. For reference, British Columbia has a draft criterion for chloride in surface water of 150 mg/L (Nagpal, N. K., *Ambient Water Quality Guidelines for Chloride*, Ministry of Water, Land and Air Protection) and the US EPA has a criterion of 230 mg/L (*Ambient Water Quality Criteria for Chloride*, USEPA 1988). Because chloride will be bled from the bismuth recovery circuit, recycle of TMA waters will result in the buildup of chloride in the decant water to levels where the TMA effluent must be treated and discharged to remain below these literature values, or another value determined by derivation of a site specific water quality criteria.

#### **4.15 Provide a geochemical description of the solid fraction of the tailings.**

Three tailings streams (Fe-As precipitate rich residue, cyanide deconstruction composite tailings, and co-mingled composite tailings) were analyzed to assess the solid-phase composition of the tailings during the mini-pilot plant phase of metallurgical development. The co-mingled composite tailings are considered to best represent the tailings to be deposited. A summary of the solids content relative to typical crustal abundance is shown in Table 4.15-1.

**Table 4.15-1 Co-mingled Composite Tailings Supernatant Analysis**

Parameter	Tailings (µg/g)	Typical Crustal Abundance <sup>a</sup>
Aluminum	29,000	82,300
Antimony	37	0.2
Arsenic	19,000	1.8
Barium	120	425
Beryllium	3.4	3
Bismuth	1,300	0.0085
Cadmium	0.78	3
Calcium	51,000	41,500
Chromium	76	102
Cobalt	690	25
Copper	280	60
Iron	230,000	56,300
Lead	77	14
Lithium	<0.3	20
Magnesium	23,000	23,300
Manganese	1,300	950
Mercury	<0.1	0.085
Molybdenum	18	1.2
Nickel	34	84
Potassium	26,000	20,850
Selenium	<5	0.05
Silver	0.98	0.075
Sodium	3,900	23,550
Strontium	6.9	370
Thallium	0.6	0.85
Tin	9	2.3
Titanium	1,500	5,650
Vanadium	6.8	120
Yttrium	14	33
Zinc	100	70

Notes: µg/g = micrograms per gram; < = less than

a = Price, W.A., 1997. *DRAFT Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Mine Sites in British Columbia*: British Columbia Ministry of Employment and Investment, Energy and Minerals Division,.

Relative to these typical crustal abundances, the tailings will contain elevated concentrations of arsenic and bismuth, and to lesser degrees: antimony, cobalt, copper, iron, lead, molybdenum, silver, tin and zinc. This is typical for a mineralized area and tailings. As indicated above, the solid residue from the bismuth recovery circuit was not included in the analysis. The bismuth recovery circuit residue will be characterized as part of the ongoing pilot plant. The bismuth residue will represent a very small component of the overall plant tailings stream (<0.5%).

Following completion of the pilot plant to be completed, approximately 0.5 tonnes of composite tailings will be collected and returned to the NICO site over the winter road for ongoing study of the stability when exposed to ambient conditions. The tailings will be placed in large high density polyethylene/steel columns that will simulate an un-flooded tailings deposition, which is a worse case condition from the perspective of acid rock drainage potential. The test work will assess leaching characteristics of the

tailings when exposed to ambient oxygen and rainfall, through analysis of the collected leachate. These results will be correlated to previous work completed in controlled environments and utilized for predictive modeling of tailings pond activity in the field.

#### **4.16 Identify the current source of power production.**

Currently, diesel electric generators produce electrical power on site. During operations, NICO will be designed with its own diesel-fired power plant and heat recovery systems, along with the appropriate switch gear to allow for delivery of grid power at a future date (if, and when, it becomes available).

#### **4.17 Other properties (or will the mill be handling any in the future)?**

Fortune holds a 100% interest in the Sue-Dianne copper-silver deposit, located 25 km north of NICO. In addition, other mineral deposits are known in the vicinity of NICO, including the Damoti Lake gold mine, which could conceivably be processed in the future. Although not currently contemplated, ores from either site could be processed by the same flow sheet with minor modifications.

#### **4.18 If so, specify ore characteristics and describe any mill processes which will change as a result.**

No processing of ores from other projects is currently planned for consideration with respect to these applications.

In the case of the Sue-Dianne project, which Fortune owns, the NICO process described is complementary for treatment of concentrates produced from ores of the Sue-Dianne deposit, especially if the option case of Solvent Extraction and Electro-winning of Copper is utilized for copper recovery from the NICO deposit. In this situation, additional extraction, scrubbing and electro-winning cells would be required, and the process is being designed so that these unit operations can be added on to the NICO circuit in a modular fashion to address recovery requirements.

The recovery of magnetite from the NICO bulk tailings is subject to ongoing market evaluation and batch testing, with gravity and magnetic separation used as the means of recovery. It has been demonstrated that 6-7% of the bulk tailings can be collected into a magnetic concentrate assaying approximately 82% magnetite (72% iron) and 12% silicate. Silicate is a penalty on magnetite sales, although it can be removed to less than 5% of the concentrate through fine grinding. The removal could occur on site or off site.

In a similar fashion, tungsten and certain rare earth elements (REEs) present in the NICO and Sue-Dianne deposits could be recovered, under certain economic circumstances.

An amendment to the water license will be submitted should these modifications be incorporated into the process flow sheet at a later date, which would be subject to the market conditions, geological modeling, and metallurgical test work. The production of a magnetite, tungsten, or REEs concentrate would increase the amount of water used in process; mostly addressed through recycling; but would not be expected to add dissimilar chemicals to the flow sheet already described. Instead, in the case of magnetite production would reduce the TMA volume required per tonne of ore processed.

#### **4.19 If tailings are being recovered in the mill or elsewhere for use as backfill (etc.) in the mine (etc.), indicate the quantity of solid tails (tonnes/day) recovered from the mill process.**

Tailings will not be used as mine backfill. The mine will use unconsolidated rock fill and cemented rock fill for backfilling purposes in the underground mine.

#### **4.20 Will exits be bermed to prevent spills from escaping the mill?**

Yes. Layouts will be developed at the detailed design stage.

**4.21 Will all sumps for process tanks have the required 110% holding capacity of the largest tank?**

Yes.

## SECTION 5 -- THE TAILINGS AREA

### 5.1 Is the tailings containment area (being) designed for total containment?

Yes, all of the tailings will be directed to the TMA which has been designed to contain the total volume generated by the mine.

Figure 4a illustrates the general arrangement plan of the TMA. The facility provides the required storage capacity for the life of the mine, while minimizing any environmental and risk issues. The facility will also provide the possibility for expansion by raising the perimeter dams if further resources are encountered.

The TMA includes four water retaining dam structures, emergency spillways, effluent discharge pump barge and pipeline to the ETF, a potential reclaim water pipeline to the process, and seepage collection systems. The TMA will be constructed in stages.

### 5.2 Attach detailed scale plan drawings of the proposed (or present) tailings area. The drawings must include the following:

- (a) details of pond size and elevation;
- (b) precise details of all retaining structures (length, width, height, materials of construction, etc.);
- (c) details of the drainage basin, and existing and proposed drainage modifications;
- (d) details of all decant, siphon mechanisms etc, including water treatment plant facilities;
- (e) the plan for tailings deposition and final tailings configuration;
- (f) details with regard to the direction and route followed by the flow of wastes and/or waste waters from the area; and,
- (g) indications of the distance to nearby major watercourses.

**NOTE: Individual detailed large scale drawings of any facility (dam, decant system, ditch, dike, water treatment plant, etc.) (to be) constructed must be attached. Specific details with regard to the methods of construction, materials (to be) used, etc., are required.**

The requested items are shown on the figures listed below.

Item	Figure	Description
a)	4a	Proposed Site Development
b)	4a	Proposed Site Development
	6	Tailings Management Area Dams A,B,C and D Typical Section
c)	2	Drainage Areas
	4a	Proposed Site Development
d)	4a	Proposed Site Development
	8	Effluent Treatment System Preliminary Schematic Flow Diagram
e)	4a	Proposed Site Development
	9	Conceptual Closure Plan
f)	4a	Proposed Site Development
	9	Conceptual Closure Plan
g)	1	Project Study Area
	2	Drainage Areas

### 5.3 Explain your choice of location for the tailings pond design by rationalizing rejection of other options. Consider the following criteria in your comparisons: subsurface strata permeability,

**abandonment of tailings, recycling/reclaiming waters, and assessment of runoff into basins. Attach a brief summation.**

Subsurface permeability and abandonment of tailings were not considered as critical criteria for tailings location at the desktop selection phase because these considerations were generally similar at each location. The selected site is the closest to the mine site and subject to the minimum amount of runoff into the basin.

Eleven *possible* sites, defined as capable of holding the tailings volume required regardless of suitability, were identified within a 5 km radius of the proposed mill site. The study area was then divided into two principal watersheds named for the main lake in each: the Burke Lake and Lou Lake watersheds, both of which drain separately into the Marian River. Both Lou Lake and the Marian River were avoided because they are fish bearing waters.

Following completion of basic screening for fish, capacity calculation and assessment of the availability of a suitable area for a polishing pond, the 11 sites were ultimately reduced to 4 that were classified as *potential* sites, subject to further evaluation. The four are located within the Burke Lake watershed. The Burke Lake watershed can be subdivided into sub-watersheds from upper to lower (upper sub-watersheds draining into lower sub-watersheds): Nico Lake, Peanut Lake, and Burke Lake, respectively, as shown on Figure 2. The mine site is in the Nico Lake sub-watershed and the proposed mill site is in the Peanut Lake sub-watershed. The following summarizes characteristics of the 4 potential sites and rationale for selection or rejection.

1. Nico Lake Basin: This basin is approximately 2.2 km from the proposed mill site and receives runoff from the mine area. It has reasonably good topographic containment and offers a high ratio of tailings storage volume to constructed dam fill volume. The potential for expansion to accommodate additional ore beyond the design tonnage is also good. However, a relatively large upstream watershed east of Nico Lake would have to be diverted to reduce the net inflow to the basin. The background arsenic level in Nico Lake is about 0.03 mg/L which is higher than the *Canadian Water Quality Guidelines* ("CWQG," published by the Canadian Council of Ministers of the Environment, 2006) water quality objective of 0.005 mg/L; thus, this water body is naturally degraded. Due to the presence of fish in Nico Lake and its greater distance from the open pit, Site 1 was not further considered.
2. Grid Ponds Basin: This basin, is located immediately north of the ore body and portal and within 1.5 km of the proposed mill site. It is near the top of a watershed in which water diversions would not be required. Water from the spring in the upper basin contains 0.2 mg/L arsenic, well above the CCME criterion for aquatic life and approaching the MMER end-of-pipe discharge limit of 0.5 mg/L. Topographic containment is good, with high ridges forming much of the perimeter, although a long dam would be required along the eastern side of the basin. The Grid Ponds do not support fish. Due to the lack of fisheries, proximity to the project site, and suitable capacity, this site was selected and has been designated as the TMA.
3. Basin South of Ore Body: This basin contains a few unnamed small ponds. The site is immediately south of the ore body, approximately 1.4 km from the proposed mill. The topographic containment is fair, with ridges on the north and south sides of the basin. The small ponds do not support fish. Due to the lack of fisheries, proximity to the open pit and favourable topographic containment, this site was selected for MRMA.
4. Basin South of Mill Site: This basin, located 1.3 km from the proposed mill in a localized watershed of 114 hectares (ha), is effectively an extension of the above MRMA basin. It contains a small pond which does not support fish. The topographic containment is good, but the tailings storage volume to constructed dam fill ratio is relatively low. This site was not considered adequate on its own, but had potential to be used in conjunction with the basin south of the ore body as a polishing pond or for tailings disposal if the ore reserves increase. It was not selected for tailings disposal.

5.4 The total area for the proposed tailings management area is 119.8 ha.

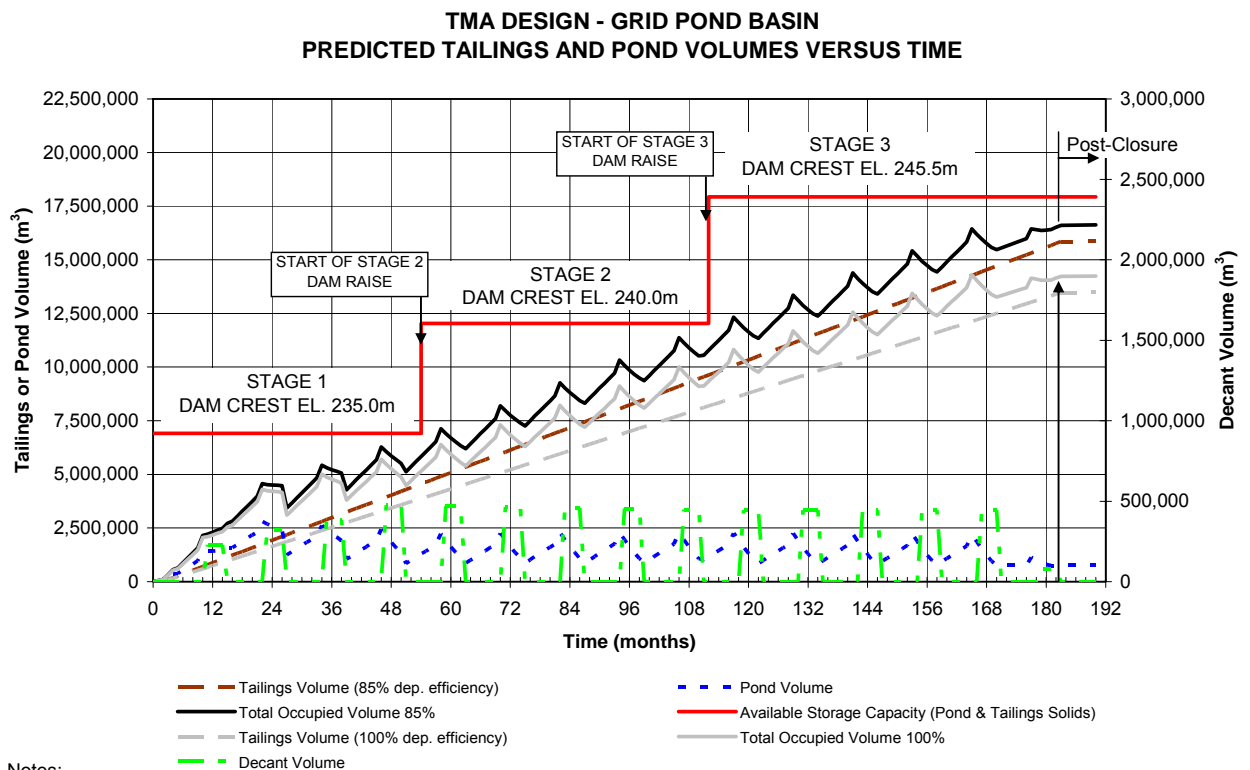
5.5 The average depth of the tailings basin will be: Approximately 20 m.

5.6 Indicate the total capacity for the existing tailings area by using water balance and stage volume calculations and curves. (Attach a description of inputs and outputs along with volume calculations).

Not applicable.

5.7 Indicate the total capacity for any proposed tailings area by using water balance and stage volume calculations and curves. (Attach a description of inputs and outputs along with volume calculations).

The TMA has a capacity that can store the anticipated total tailings production of 13.8 Mm<sup>3</sup> (22 million tonnes). The facility is flexible enough to store an additional 2.8 Mm<sup>3</sup> of tailings should the reason arise (such as less than 100% tailings deposition efficiency due not reaching consolidation targets).



**Notes:**

1. Average annual rainfall conditions.
2. Assumed a tailings deposition efficiency factor of 85%.
3. Month of Mill startup is September 1th (actual timing could vary).

A floating pump barge will pump excess water (snow melt, rainfall, inflows and tailings decant) from the TMA pond through pipelines to the mill for recycling or to the ETF for treatment and discharge. Water not recycled in the plant will be treated on a seasonal basis in the ETF (by the addition of ferric sulphate then lime) to remove arsenic and other metals by co-precipitation. Treated water will be combined with a flocculant solution and discharged to the sedimentation pond, and released to the environment once discharge criteria are met.

Pumping of water from the TMA for recycling in the plant, or treatment for discharge, will start at the beginning of operation and will continue throughout the life of the mine.

Dewatering of underground and open pit workings and runoff resulting from rainfall and snow in the open pit, will be pumped into the tailing thickener or the TMA to recycle the water to the process.

**5.8 Will the present tailings area contain the entire production from the mine mill complex for the life of the project?**

Yes. The tailings dams can be increased in height to accommodate the low grade ore if it is processed. Other options such as co-disposal in the MRMA are being considered.

**5.9 If “No” above, or if production output increases tailings volumes, indicate what plans have been made for future tailings disposal on the property.**

Not applicable

**5.10 Has any land in the immediate area been identified as native or crown land or withdrawn pending Native Claim Settlement?**

With the exception of Fortune’s leases, all of the land surrounding the mine is within the Tłıchǵ settlement lands owned and managed as fee-simple lands by the Tłıchǵ Dèts’ǵ Kâowo as per the Tłıchǵ Agreement. The Tłıchǵ lands are within the Wek’èezhii co-management lands, jointly managed with the Northwest Territory and Federal Government. Fortune’s exploration leases were staked and brought to lease prior to settlement of the Tłıchǵ land claim and as Crown Land are administered by the Federal Government.

**5.11 Do the tailings area and all related treatment facilities lie on company held claims?**

Yes, they occur entirely on the NICO mining leases.

**5.12 If not, indicate mine claim boundaries (and owners) on tailings area plan map (see Q.5.8). Also, attach a copy of all pertinent agreements signed with the owners of the claims not held by the Company.**

Figure 10 shows that all mine facilities are within the NICO claim.

**5.13 Will the proposed tailings area engulf or otherwise disturb any existing watercourse?**

Yes, construction of the TMA will disturb a minor watercourse upstream of the Grid Ponds which has naturally elevated arsenic levels.

**5.14 If “Yes”, attach all pertinent details (name of watercourse, present average flow, direction of flow, proposed diversions, etc.).**

A small unnamed stream discharges from the bowl zone above the Grid Ponds into the ponds and continues downstream discharging into Nico Lake, as shown on Figure 3. The stream discharge rates have not been measured in detail, but based on measurements in the watershed downstream and proportioning on the basis of the catchment area, the following estimates of flow from the tailings basins is presented in Table 5.14-1.

**Table 5.14-1 Estimates of Monthly Flow Rates from Tailings Basins, (cubic metres per second)**

Sub-basin	June	July	August	September	Daily Peak
Tailings Management Area	0.018	0.005	0.002	0.003	0.020

**5.15 If any natural watercourse will gain access to the proposed tailings area, what methods will be used to decrease the amount of runoff water entering the containment area? Indicate the volume of water which will enter the tailings area from the source(s) in question and attach all pertinent details of proposed diversions.**

The TMA receives a small volume of stream flow from the “bowl zone” portion of the mineralized deposit that, in turn, receives base flow recharge from a mineralized spring emanating from the ore bearing bedrock. Since it contains naturally elevated concentrations of arsenic, surface runoff will be incorporated into the TMA water balance and recycled into the process or treated downstream, in conjunction with the mine/tailings effluent. This spring water is slightly alkaline of neutral.

An annual water balance has been calculated by Golder Associates Ltd. (“Golder” in the report titled *Surface Water Monitoring Program 2005*, dated June 2007. An annual average outflow of 0.34 m<sup>3</sup>/s was calculated at the outlet of sub-basin BL8 (see Figure 2). The annual average discharge from the area contained by the TMA, pro-rated on a unit-area basis, would be approximately 0.001 m<sup>3</sup>/s (840 m<sup>3</sup>/d). This would relate to an annual input to the TMA of 308,000 m<sup>3</sup>.

**5.16 Indicate on the tailings area plan drawing (see Q.6.1) all sources of seepage presently encountered in the vicinity of the tailings area, the volume of each seepage flow (m<sup>3</sup>/day), and the direction of each flow.**

None have been identified.

**5.17 Are the seepage flows from the property presently being treated chemically? If so, describe how.**

Not applicable.

**5.18 If not, explain.**

Not applicable.

**5.19 Please attach a conceptual Abandonment and Restoration Plan for all tailings areas being developed. Describe the measures that have been (or will be) taken to contain and stabilize the tailings area(s) against leaching and seepage after operations on the property cease.**

A preliminary conceptual closure plan is presented in Section 7.5 and shown on Figure 9. Attachment 2 provides elaboration on the closure planning. Following restoration, ongoing monitoring, water treatment and site maintenance activities will be required.

**5.20 Describe the proposed or present operation, maintenance and monitoring of the tailings area.**

An operations, maintenance and surveillance (OMS) plan in accordance with the Mining Association of Canada’s latest guideline (*Developing an Operation, Maintenance and Surveillance Manual for Tailings and Water Management Facilities*) will be implemented. The OMS Plan will set out the roles and responsibilities for safely operating the TMA and provide a detailed schedule for monitoring and inspections at different levels of responsibility.

The operation and maintenance manual will include the following components:

- Containment Structures – Dams A, B, C, D, and the Sedimentation Pond Dams;
- Internal Dyke Structure;
- Emergency Spillways – Tailings Basin and Sedimentation Pond Spillways;
- Tailings Delivery and Distribution System;
- Effluent Discharge and Decant Systems: Tailings Basin and Sedimentation Pond;

- ETF;
- Runoff Diversion Ditches;
- Environmental Monitoring System;
- Dam Instrumentation;
- Access Roads; and
- Water Balance.

It will also specify supplies and services that must be in place, including:

- Supply of maintenance tools, chemicals, lubricants, and fuel;
- Designated system for maintaining documentation control for the TMA (e.g. maintenance records); and
- Site security such as proper signage, fencing, etc.

Mine operating personnel will carry out daily drive-around inspections that will be logged. More formal inspections will be undertaken on a monthly basis by experienced Fortune personnel and on an annual basis by an experienced qualified practicing professional engineer. Detailed engineering inspections will also be carried out in the event of certain events such as long return period storms or earthquakes. The OMS Plan will codify the requirements for documenting and following up on maintenance requirements, as well as procedures to be followed in the event of an emergency. Surveillance of geotechnical instruments including piezometers and thermistors, and ground water, surface water and effluent water quality will be described.

## SECTION 6 -- WATER TREATMENT

**6.1 Describe the methods of chemical treatment that are presently being used and/or will be used to control the quality of the tailings effluent. Attach engineering drawings where applicable and a process flow chart. If a pilot test has been conducted please attach description of methodology and results.**

### Design Basis:

The main purpose of the proposed treatment process in the ETF is to remove arsenic from water pumped from the tailings ponds. This arsenic represents residual concentrations associated with the waste rock stockpiles, and that not fixed through the production of scorodite in the autoclave, or precipitated by the production of iron-arsenic hydroxide precipitate in the plant. (While cyanide will be used in the mill process, this will be treated in slurry using a SO<sub>2</sub>-air cyanide destruction system in the mill. In addition, trace residual cyanide will degrade naturally during retention and exposure in the active TMA pond, and be diluted by other inputs such as runoff.)

The basis of sizing for the ETF is a worst case scenario that assumes that fresh water intake from all sources is approximately 7,200 m<sup>3</sup> per day. This would occur when no water is reclaimed from the TMA and when other internal process water recycling strategies outlined in section 1.8 are not implemented:

- Effluent flow rate: 23,940 m<sup>3</sup>/day. (This is the maximum design flow rate based on an average year plus a 1 in 100 year return, 24 hour duration, storm event.); and
- Approximate average flow (for 5 months of treatment in an average year): 20,440 m<sup>3</sup>/day (3,117,200 m<sup>3</sup>/year).

Sizing of the minimum treatment rate for the ETF takes into consideration the case where all daily water usage for process make-up requirements are addressed through use of reclaim water in the TMA, instead of the use of fresh water. Approximately 6,100 m<sup>3</sup> of fresh water per day is avoided with the use of reclaim water. This assumes a minimum average flow (for 5 months of treatment in an average year): of 5,840 m<sup>3</sup>/day (890,700 m<sup>3</sup>/year).

Effectively, the size of the ETF would be controlled by the rate of recycling of reclaim water from the tailings pond. Inputs other than process water such as potable water and run-off result in dilution of the process water by a ratio of 3.4:1. This should allow for reclaim water use on at least a seasonal basis with the reclaim water meeting targets for cyanide and chloride concentrations that are not detrimental to the process. To address the variability between minimum and maximum, the ETF could be constructed in trains or modules that are brought into service when required.

Other design criteria of the ETF for treatment of waters leaving the TMA (inputs to the ETF) are summarized as follows:

- pH of the raw effluent: 8.0 (average),
- Arsenic concentration: 3 mg/L (average),

The assumed dosages of chemicals used in the treatment process based on laboratory testing are:

- Ferric sulphate, Fe<sub>2</sub>(SO<sub>4</sub>)<sub>3</sub> - 100 mg/L of treated effluent;
- Hydrated lime, Ca(OH)<sub>2</sub> - 200 mg/L of treated effluent, or the equivalent dosage of hot lime (CaO); and,
- Flocculant.

The removal is via a precipitation, coagulation, and flocculation process. The addition of ferric sulphate and lime solutions scavenges traces of arsenic (+5), (as the highly charged arsenate, AsO<sub>4</sub><sup>≡-</sup>, the stable

from of arsenic under oxidizing alkaline conditions). Ferric arsenate as  $\text{FeAsO}_4$  is produced primarily by a surface absorption effect, so a significant excess of iron is required to ensure stability.

The resulting amorphous ferric hydroxide that is present in excess will tend to precipitate a number other metal species by adsorption and coagulate with the other fine precipitates formed, resulting in the removal of metals such as copper, nickel and zinc.

At a pH to about 9.0, copper ferricyanide is optimally precipitated resulting in the removal of some residual cyanide, if necessary. The main action is by coagulation (agglomeration) and the weighting down of very fine particles to colloidal precipitates.

The precipitation and coagulation, combined with the subsequent addition of weak anionic flocculant, results in a solid that will settle.

In similar flow sheets, low dosages of hydrogen peroxide are occasionally utilized to convert arsenic (+3) (as arsenite) to the oxidized form of arsenate in order to promote removal. At NICO, the installation of aeration piping at the bottom of the reactor pond serves the same purpose, but for smaller concentrations of arsenite.

Based on analytical testing of representative mine rock samples and MMER Criteria (see section 2.5 above), concentrations of other metals do not appear to require targeted treatment. Further verification is underway through the modeling of pilot plant data.

Acute lethality toxicity testing was undertaken on undiluted and untreated leachate obtained from metal leaching and acid rock drainage (ARD) testing of the composite tailings (described in section 2.5 above) on days 15, 45 and 120 of the program. With the exception of *Daphnia magna* in 15 day leachate, the *Daphnia magna* and rainbow trout tests resulted in no mortality. The 15 day leachate *Daphnia magna* test had 66.7% mortality.

#### Brief Description of the Treatment Process and System

Figure 8 provides a schematic diagram of the process to be used in the ETF. Raw effluent from the TMA will be pumped to the reactor pond (13). Ferric sulphate solution will be added to the raw effluent at the inlet to the reactor pond, and the treated effluent will flow through the first two sections of the pond, where it pass over the aeration/mixing system installed at the bottom of the pond. In addition to the mixing, the blown air (11) will oxidize arsenite ions to arsenate and ferrous iron ( $\text{Fe}+2$ ) to ferric iron ( $\text{Fe}+3$ ), producing a more stable precipitate as discussed above. The addition of air will also polish the effluent of ammonia concentrations. The ferric sulphate solution will be stored at the site in two storage tanks (1 and 2), and it will be dosed to the reactor pond with one of two metering pumps (8).

At the inlet to the third section of the reactor pond, lime slurry will be added to precipitate ferric hydroxide together with ferric arsenate (coagulation). The treated effluent will flow through the third and fourth sections of the reactor pond. Lime will be stored at the site in the lime silo (5) and 20% slurry will be prepared using a typical hydrated lime slurry preparation system. Alternatively, pebbled hot lime can be utilized with a slaking mill. The lime slurry will be pumped from one of two tanks (3 or 4) to the reactor pond using one of two metering pumps (9). Operating experience indicates that the treatment efficiency and precipitate stability is not overly sensitive to pH fluctuations if the Fe/As molar ratio is maintained above 5:1.

Re-circulated sludge can also be added to the treated effluent together with lime. The re-circulated sludge will be pumped from the re-circulated sludge pumping station, which will be located next to the sedimentation pond (shown as an option on the Schematic Flow Diagram, Figure 8). One of two compressed air driven diaphragm pumps will suck the sludge from the bottom of the pond and pump it at the required rate to the reactor pond in order to optimize the use of reagents.

The fifth section of the reactor pond is designed to work as a flash mix tank to provide a vigorous mixing of the treated effluent with the flocculant solution. In the final chamber, the treated effluent will be further mixed, but at a much slower rate, to allow formation of large flocs that will settle in the sedimentation pond to which the reactor pond discharges in the final stage of the treatment process. Polymer stock solution will be prepared at the site using a polymer dosing system consisting of dilution tanks (6 and 7) equipped with mixers and a metering pump (10).

The treated water in the reactor pond effluent will flow by gravity to the sedimentation pond where solids, which have been precipitated and flocculated in the reactor pond, will settle out. The design volume allowance for accumulated sludge in the sedimentation pond is approximately 60,000 m<sup>3</sup>, sufficient to allow the system to operate for seven to ten years before the accumulated sludge must be removed. The sludge will be removed using a cutter suction dredge and pumped to the active TMA basin.

## **6.2 List the names of chemicals to be used in the water treatment process.**

Ferric sulphate, Fe<sub>2</sub>(SO<sub>4</sub>)<sub>3</sub>;  
Hydrated lime/calcium hydroxide, Ca(OH)<sub>2</sub>, or the equivalent as lime and  
Flocculant.

## **6.3 What is the proposed or present average rate of effluent treatment of the plant (if applicable)?**

The effluent required to be treated by Fortune will vary over the years of operation, depending on seasonal precipitation and radiation, size of the pit, size of the decant pond within the TMA, underground operations, and other factors independent of the process flow sheet. The amount of effluent that will require treatment will primarily be related to the success of reclaim water use from the TMA when it is returned to the plant for use as process water.

In addition, the amount of effluent discharged from the ETF is also determined by the ability to return treated water to the process for re-use. If the proposed design is utilized, treatment will be seasonal over a period from mid-May through mid-October. In this case, the treatment capacity of the ETF over 5 months greatly exceeds the average daily fresh water requirement of the process.

Over 5 months, the ETF plant will treat an average of 20,440 m<sup>3</sup>. If treated water is returned to the process to address all fresh water requirements, 14,340 m<sup>3</sup> of water will be discharged to the environment.

Table 6.3-1 summarizes some of the inputs and outputs of the water balance.

**Table 6.3-1 Water Balance Showing the Maximum Annual Discharge from the Tailings Management Area**

<b>Stream</b>	<b>Volumes (m<sup>3</sup>/y)</b>	<b>Total Volumes (m<sup>3</sup>/y)</b>
<b>Inputs</b>		3,967,000
Fresh Water for Potable Service	401,500	
Fresh Water for Process Service	2,226,500	
TMA Runoff	766,00	
Sedimentation Ponds – Runoff less Evaporation	28,000	
Moisture in Ore	90,000	
MRMA Runoff less Absorption	435,000	
Miscellaneous flows Underground, Open Pit	20,000	
<b>Outputs</b>		849,800
Evaporation	397,000	
Retained in Tails	424,000	
Seepage	28,800	
Discharge from TMA	3,117,200	3,117,200

Notes: m<sup>3</sup>/y = cubic metres per year; MRMA = Mine Rock Management Area; TMA= Tailings Management Area

**6.4 What is the proposed or present maximum effluent treatment capacity of the plant (if applicable)?**

23,940 m<sup>3</sup>/day (based on an average year, plus a 1 in 100 year return 24-hour storm event).

**6.5 Will treated effluent be discharged directly to a natural waterbody or will polishing or settling ponds be employed? Describe location, control structures and process of water retention and transfer. Attach any relevant design drawings.**

The proposed effluent treatment facility schematic flow diagram is presented on Figure 8 and its proposed location is shown on Figure 4a.

A sedimentation pond will be employed. The treated water in the reactor pond effluent will flow by gravity to the sedimentation pond where solids, which have been precipitated and flocculated in the reactor pond, will settle out. This settling pond sludge capacity will allow the system to operate for about seven to ten years before it becomes necessary to remove the accumulated sludge. The sludge will be picked up using a cutter suction dredge and pumped to the active TMA basin. These maintenance activities will be undertaken prior to, or following the treatment season, in order to avoid interfering with the treatment of water.

**6.6 Name the first major watercourse the discharge flow enters after it leaves the area of company operations.**

Peanut Lake

**6.7 In terms of rate of effluent release, and volume and flushing rate of the receiving watercourse, estimate the extent of the mixing zone within the receiving waters and where background levels of constituents for that watercourse will be attained.**

The treated effluent release will occur into Peanut Lake and effluent will mix within the lake prior to exiting via the creek water that flows into Burke Lake. The Peanut Lake catchment provides over 80% of the flow in the Burke Lake Watershed. During effluent discharge periods, water from Peanut Lake will immediately mix with creek water, increasing the flows that reach Burke Lake. Eventually, Burke Lake will

be fully mixed with the water representing the combined flow into the Peanut watershed and from the ETF.

**6.8 Describe the present (proposed from pilot tests) chemical and physical characteristics of the tailings effluent (decant).**

The tailings effluent will not be discharged directly to the environment. The tailings decant will be directed to the ETF to treat the water in order to meet MMER criteria. In particular, arsenic will be the main target for treatment. Table 6.8-1 shows the expected concentrations of the tailings decant to be treated to meet MMER criteria. Although the tailings decant water is not proposed to be discharged directly to the environment, the concentrations have been referenced to the MMER criteria for context.

**Table 6.8-1 Expected Concentrations of the Tailing Decant**

Parameter	Units	Untreated Decant <sup>a</sup>
<b>General Parameters</b>		
Alkalinity as CaCO <sub>3</sub>	mg/L	111
Hardness	mg/L	180
pH		7.1-8.2
Sp. Conductivity	µS/cm	615
Suspended Solids	mg/L	n/a
<b>Nutrients</b>		
Total Ammonia N	mg/L	0.3
Un-ionized Ammonia	mg/L	0.022
<b>Total Metals</b>		
Aluminum	mg/L	0.15
Arsenic	mg/L	3.02
Cadmium	mg/L	0.0001
Chromium	mg/L	0.002
Copper	mg/L	0.0028
Iron	mg/L	0.06
Lead	mg/L	0.0013
Manganese	mg/L	0.036
Mercury	mg/L	<0.0001
Nickel	mg/L	0.004
Silver	mg/L	<0.0001
Zinc	mg/L	0.005
<b>Others</b>		
Total Cyanide	mg/L	n/a
Oil and Grease	mg/L	n/a

Notes: mg/L = milligrams per litre; µS/cm = microSiemens per centimeter; < = less than  
a = Generally worst case for supernatant analyses, day 1 to 90 testing (Tables 4 and 5, SGS Lakefield report #10226-004 by Wagner and Elliot dated July 22, 2004).

Relative to the existing water quality at the discharge location, the tailings decant is elevated with respect to copper, lead, manganese, nickel, arsenic, chromium, aluminum, specific conductivity, alkalinity and hardness. Relative to the MMER criteria, arsenic is elevated. The treated water effluent will meet MMER.

## SECTION 7 -- ENVIRONMENTAL MONITORING PROGRAM

### 7.1 Has any baseline data been collected for the main waterbodies in the area prior to development?

Yes

### 7.2 If "Yes", include all data gathered on the physical, biotic and chemical characteristics at each sampling location. Identify sampling locations on a map.

Attachment 3 of the Mining *Industry Questionnaire* provides a screening level summary of predicted impacts and proposed mitigation strategies for development of the mine. Baseline data for the main waterbodies in the study area is summarized in a report prepared by Golder Associates Ltd. entitled *Environmental Surveys at Fortune Minerals Limited NICO Deposit, 1998 – 2004*. Further baseline data was collected in 2007 and 2008 and will be presented in the baseline section of the environmental assessment.

### 7.3 Provide an inventory of hazardous materials on the property and storage locations. (attach separate map)

Table 7.3-1 presents projected annual chemical consumptions and assumes that a one month inventory would be stored on site at any given time. Where applicable, inventory maximums are shown as in storage. Proposed chemical, explosives and fuel storage locations are shown on Figure 4a.

The table represents the projected reagent consumption for the base case of copper precipitation, cobalt ion exchange and electro-winning, and bismuth chloride leaching and electro-recovery. For the case of copper solvent exchange and electro-winning, the table shows the corresponding reduction in reagents related to the base case, in addition to the requirements of solvent extraction.

**Table 7.3-1 Projected Annual Chemical Consumption**

<b>Material</b>	<b>Consumption</b> (tonnes per year)	<b>Storage</b> (tonnes) (supply on hand)	<b>Storage Location</b> (Vessel)
<b>Mining</b>			
Ammonium nitrate (NH <sub>4</sub> NO <sub>3</sub> )	2,080	175	Explosives buildings, magazines
Explosive Emulsion	As needed		Explosives buildings, magazines
Cement <sup>a</sup>	15,000	500 (2 weeks)	Lay-down area, bags
<b>Communion</b>			
Steel grinding rods	910	75	Lay-down
Steel grinding balls	2,250	120	Bins in mill building
<b>Processing</b>			
Flocculant	45	8 (2 months)	Cold storage area
Lime (CaO)	16,625	400 (10 days)	Silo, plant
Iron Powder	600	50	Cold storage area
Sodium Hydrosulphide (NaHS)	140	12	Reagent storage area
Sodium Carbonate (Na <sub>2</sub> CO <sub>3</sub> )	4,775	100 (1 week)	Silo, covered, plant
Sulphuric Acid (H <sub>2</sub> SO <sub>4</sub> )	4,775	90 (1 week)	Polypropylene Tanks (50 m <sup>3</sup> )
Nitric Acid (HNO <sub>3</sub> )	12	1	Tote, Reagent storage area
Hydrochloric Acid (HCl)	12	1	Tote, Reagent storage area
Sodium Cyanide (NaCN)	400	30	Box/Bag, Reagent storage area
Sodium Hydroxide (NaOH)	400	30	Bags, Reagent storage area
Sodium Chloride (NaCl)	150	13	Bags, Reagent storage area
Potassium Amyl Xanthate (PAX)	440	40	Bags, Reagent storage area
Methylisobutyl Carbinol (MIBC)	90	10	Tote, Fuel storage area
Lignin Sulphonate	336	28	Bags, Reagent storage area
Diatomaceous Earth	45	4	Cold storage area
Refinery Flux	2.9	0.25	Warehouse, Laboratory
Sodium Metabisulphite (Na <sub>2</sub> S <sub>2</sub> O <sub>5</sub> )	1,600	130	Reagent storage area, or Silo
Copper Sulphate (CuSO <sub>4</sub> )	45	2	Reagent storage area
Manganese Sulphate (MnSO <sub>4</sub> )	<<100, bleed	make-up	Reagent storage area
Activated Carbon	12	1	Cold storage area
Lewatit® VP OC 1026	Break down	make-up	Warehouse
Dow XUS-43578	Break down	make-up	Warehouse
Litharge (PbO <sub>2</sub> )	2.8	0.25	Warehouse, laboratory
<b>Seasonal Storage</b>			
Ferric Sulphate (Fe <sub>2</sub> (SO <sub>4</sub> ) <sub>3</sub> )	400	80	Tank, SS, ETF
Hydrated Lime (Ca(OH) <sub>2</sub> ), or equivalent Lime (CaO)	800	96	Covered Silo, 80 tonnes, ETF
<b>Copper SX/EW Option<sup>p</sup></b>			
Flocculant	-2	-	Reduction
Lime (CaO)	-1,300	-	Reduction
Sulphuric Acid (H <sub>2</sub> SO <sub>4</sub> )	-2,300	-	Reduction of Tankage (25 m <sup>3</sup> )
Iron Powder	-425	-	Reduction
Sodium Carbonate (Na <sub>2</sub> CO <sub>3</sub> )	-450	-	Reduction
Diatomaceous Earth/Clay	-24	-	Reduction
Guartec® EW, or similar	1	1	Bags, Cold Storage Area
Cobalt Sulphate Monohydrate <sup>c</sup> , (CoSO <sub>4</sub> ·7H <sub>2</sub> O)	<<40	4	Bags, Reagent Storage Area
ACORGA ® M5774, or competitor	0.2		Drum or Tote, Warehouse
Kerosene	7	7	SX/EW Plant (10 m <sup>3</sup> , 1 year)
<b>Fuel</b>			
Diesel (Mining)	6,275,000	712,000	Enviro tanks in fuel storage area, mobile tanks, day tanks for pumps and emergency gensets.
Diesel (Power Generation)	30,000,000	1,205,000 (2 weeks)	Fuel storage area, built to suit
Gasoline	9,000	1,000	Enviro tank in fuel storage area

**Table 7.3-1 Projected Annual Chemical Consumption (Continued)**

<b>Material</b>	<b>Consumption</b> (tonnes per year)	<b>Storage</b> (tonnes) (supply on hand)	<b>Location</b> (Storage form)
Aviation Fuel	Emergency	4,100	Airport, Helicopter Base, Drums inside Insta-berm or similar
Propane	2,500,000	454,000	Supplier-installed tanks at plant and fresh air raise
<b>Maintenance Hydrocarbons</b>			
Motor Oils		2,500	500 L totes/205 L drums in warehouse oil storage and maintenance areas
Hydraulic Oils		3,000	500 L totes/205 L drums in warehouse oil storage and maintenance areas
Insulating Oils	Negligible	51,000	Inside transformers
Lubricants	1,750	150	20 L pails/cartridges in warehouse, plant and maintenance areas
Waste Oils		5,000	5000 L tank in Truck Shop
Varsol		500	205 L drums in warehouse and maintenance areas
Antifreeze/Glycol		1,000	500 L totes/205 L drums in warehouse and maintenance areas

Notes: ETF = effluent treatment facility; m<sup>3</sup> = cubic metres; L = litres;

a = Cement consumption in first two years of operation. Cement consumption will drop to 0 tonnes per year in subsequent years of operation.

b = Alternative circuit to copper precipitate has not been demonstrated through a pilot plant.

c = Reagent may not be required due to on-site production of cobalt.

Lignin sulphonate is identified as a provisional reagent used as a dispersant for the prevention of the formation of elemental sulphur in the autoclave (pressure acid leach oxidation circuit). Ideally, it will not be required, or its use will be minimized, which is a common outcome in the scale-up of autoclaves from pilot plant scale to actual production. Other chemicals identified as provisional reagents include copper sulphate (cyanide destruction by SO<sub>2</sub>-air), hydrogen peroxide (arsenite oxidation, cyanide destruction), and chlorine gas (oxidant make up, bismuth recovery).

Oxygen gas will be generated on site, but will not be stored cryogenically. Oxygen consumption will be approximately 100 tonnes per day.

Nitrogen gas will be stored and utilized to maintain pressure at the mechanical seals during emergency power loss.

For the production of cobalt, the Lewatit® VP OC 1026 and Dow® XUS-43578 IX Resin will be ordered on an as-needed basis, and stored in the heated warehouse before being added to the process.

For the copper SX/EW optional flow sheet, the totals in Table 7.3-1 do not include first fill or the storage-in-process for Cyanex 272 and Kerosene. These reagents are recycled in process. The capacity of the circuit is 3,586 kg of Cyanex 272 (extractant) and 102 tonnes of kerosene (diluent).

Onsite diesel fuel storage requirements are based on the assumption of a two week storage supply.

Many of the reagents will be transported to site in 800 or 1,000 kg lined bags, often referred to as "super sacs" or "bulky-bags." Alternatively, some reagents such as MIBC will be delivered in drums and tote boxes. Sodium cyanide boxes will be delivered in plywood crates with the super sacs contained within those crates. A lined reagent storage area will be constructed to store these types of reagents, with an isolated drainage sump that can be pumped out or diverted into the process. The location for the lined reagent storage area is shown in the location in Figure 4a, although the sizing is for representational purposes.

Due to the rate of consumption, some reagents will be brought to site in bulk carriers and delivered into silos or receiving tanks. Silage will be constructed for hot lime, hydrated lime and sodium carbonate. Receiving tanks will be constructed to distribute sulphuric acid and 50% ferric sulphate solution. The storage inventory of one month reflects the assumption that the bridge over the Mackenzie River at Fort Providence is complete. Inventory levels may vary up to 50% on containerized reagents depending on the delivery schedule.

Storage vessels for the on-site production of oxygen, and delivered propane, nitrogen, and chlorine gas (if chlorine is utilized) will be provided by the supplier. Diesel storage will be provided by double walled Enviro-Tanks, or tanks built to suit the lined fuel storage area as shown in Figure 4a.

The mine will operate an on-site assay laboratory to generate assays of pit and underground samples, as well as processing products. In addition to the fluxing reagents listed above, the following will be utilized, ideally as a pre-mixed mixture with the above, once site requirements are determined: litharge (PbO), fluorospar (CaF<sub>2</sub>) and domestic flour in very small quantities.

**7.4 Attach the present or proposed contingency plan which describes course of action, mitigative measures and equipment available for use in the event of system failures and spills of hazardous materials.**

An Emergency Preparedness and Response Plan (EPRP) will be developed for the Project in the context of the relevant territorial and federal regulations and guidelines. The EPRP will address human-caused emergencies and natural disasters that threaten life, the environment and/or property, and are beyond routine operational control. As a minimum, the EPRP will address the following, some issues of which are described further in Attachment 4.

- on-site and off-site chemical and/or fuel spills;
- tailings pipeline rupture;
- catastrophic failure of the TMA;
- pit wall failure;
- underground head failure;
- extreme drought;
- extreme precipitation, including effects on the TMA and open pit;
- plane crash;
- bus crash;
- pressure vessel failure;
- gaseous release of hydrogen sulphide, chlorine, or arsine;
- facility fires, including:
  - camp,
  - underground,
  - titanium,
  - solvent (for the case of copper SX/EW, if required),
  - fuel storage,
  - magazines, and
  - conveyors;
- earthquakes; and
- on-site forest fires.

**7.5 Provide a brief overview of the conceptual abandonment and restoration plan for the site.**

Table 7.5.1 provides a preliminary outline of conceptual activities that will be undertaken to achieve closure after the NICO Mine ceases operations. Figure 9 shows a conceptual plan of the NICO Mine site after this closure is implemented. Table 7.5.2 provides a list of probable post-closure activities. Attachment 2 provides more detailed discussion.

**Table 7.5.1 Summary of Mine Closure Activities**

Item	Activity	Comment
Tailings Management Area	Place sand and gravel cover (approx. 0.5 m thick) on exposed tailings	Cover will reduce water and wind erosion of tailings
	Flood routing study and enlarge spillway if necessary	To cope with long-term extreme precipitation events
	Water flow from pond above bowl zone may be redirected to flow west	To minimize flows into the TMA
	Evaluate stability of dams with regard to long return seismic events and upgrade if necessary	To cope with long-term seismic events
Effluent Treatment System	Confirmation of modeled long-term effluent parameters and completion of an updated trade-off study evaluating the relationship between TMA capping and flooding strategies, and passive or active long-term effluent treatment options.	The potential effects of post-closure discharges from the property to the receiving environment will be assessed during the permitting process.
	Analyse seismic stability of Settling Pond dam and upgrade with a toe berm if necessary	To cope with long-term seismic events
Mine Rock Management Area	Place boulder barrier around perimeter of top surface	Reduce risk of injury
	Regrade top surface to 1% cross slope minimum	Prevent ponding
	Construct chutes	Drain runoff to toe without erosion
	Regrade side slopes by removing benches	Improve local stability
Plant Site	Salvage process plant components where economic. Recover steel for scrap. Demolish structure and dispose of rubble in MRMA/TMA.	
Potentially Contaminated Soils	Environmental testing to assess extent and nature of impacts	Assumed that hydrocarbon impacted soils can be remediated on site in the land farm. Soils impacted with metals to be placed in the TMA
	Treat hydrocarbon impacted soil in on-site landfarm.	
	Bury metal contaminated soil in TMA	
Landfill	Closure of landfill using waste rock and tailings for cover	
Open Pit	Allow to flood. Potential for eventual spill over into north basin	Spill over will be treated in ETF
	Rock mechanics study to establish safe line. Construct perimeter berm/boulder barrier	Prevent inadvertent access
Mine Openings	Construct concrete caps for vent raises and plug ramp portal by partial backfilling	Prevent access
Haul Road and Redundant Site Roads	Remove culverts, scarify road surface and revegetate	Retain access roads to dams and spillways and to ETF.
Electrical Distribution System	Decommission and remove redundant site power lines	Retain high voltage line to site and transformer, as well as low voltage lines to ETF.
Reclamation	Scarify and grade sites of former plant and laydown areas. Revegetate.	

Notes: MRMA= mine rock management area; TMA = tailing management area; ETF = effluent treatment facility; HV = high voltage; LV = low voltage.

## Post Closure Maintenance and Monitoring

Ongoing monitoring and site maintenance activities will be required post closure. The need for post-closure water treatment will be assessed during the permitting process. Table 7.5.2 summarizes the expected post-closure activities. Additional closure concept details are provided in Attachment 2.

**Table 7.5.2 Summary of Post-Closure Activities**

Item	Activity	Comment
<b>Operations</b>		
Effluent Treatment Facility	The need for post-closure treatment is being considered. Natural levels of arsenic and other metals within the receiving environment will be one of the factors examined.	The potential effects of post-closure discharges from the property to the receiving environment will be assessed during the permitting process.
Pumps and Pipelines	Maintain in operation five months/year	Replace pumps approximately every 10 years.
Camp	Maintain small camp for 5 months per year likely proximal to the ETF	For post-closure operations staff
<b>Monitoring</b>		
Tailings Management Area	Annual geotechnical inspection of dams, covers, etc.	To ensure longer-term dam stability
	Inspection of covers (at same time as geotechnical inspection)	
Site Water Quality	Annual sampling and analysis of surface and ground water according to closure monitoring plan.	

## SECTION 8 -- PRESCREENING

In addition to providing sufficient technical and related information for licensing to proceed, applicants must provide adequate descriptive information to ensure that an initial pre-screening decision can be made prior to a project proceeding for regulatory approvals.

Your application and other project details, such as this questionnaire, will be sent out for review by local aboriginal, as well as, territorial and federal government agencies. Their comments (e.g., regarding the significance of project impacts) are considered before a decision is made to allow the project to proceed.

### 8.1 Has this project ever undergone an initial environmental review, including previous owners?

Yes, McKenzie Valley LWB: Class B Water License MV2004L2-0005 (2005), MV2006L2-0002 (renewed 2007)

### 8.2 Has any baseline data collection and evaluation been undertaken with respect to the various biophysical components of the environment potentially affected by the project (e.g., wildlife, soils, air quality), in addition to water related information requested in this questionnaire?

Yes.

The report entitled *Environmental Surveys at Fortune Minerals Limited NICO Deposit, 1998 – 2004*, provided to the Wek'èezhii Land and Water Board gives a summary of the baseline work carried out between 1998 and 2006. Attachment 3 presents a screening level assessment to identify potential impacts to the environment and people due to the proposed development of the NICO Project.

Potential impacts were screened using the following steps:

- Review of the baseline data collected for each environmental component.
- Identification of potential linkages between project activities and potential impacts to the environmental components.
  - Description of effects in terms of the following criteria:
    - magnitude (degree of change relative to baseline conditions);
    - geographic extent (mine footprint or beyond the mine footprint);
    - duration (temporal extent of the effect); and,
    - reversibility.

### 8.3 Has any meteorological data been collected at or near the site? (e.g., precipitation, evaporation, snow, wind)

Yes

### 8.4 If "Yes", please include data and attach copies of reports or cite titles, authors and dates.

For the period from October 8, 2004 to October 8, 2005, the meteorological program included hourly measurements of rainfall, temperature, wind speed, wind direction, relative humidity and solar radiation. The meteorological station is located at 512,375E/7,045,819N (NAD 27). Fortune is in the process of analyzing the meteorological data from 2005 to present day and will provide the results in the baseline section of the environmental assessment.

Rainfall measurements were compared to results from Meteorological Service of Canada weather station located 160 km southeast of NICO at the Yellowknife Airport. Rainfall measurements were similar to the long term record at Yellowknife (*i.e.*, 1971 to 2000). However, rainfall was consistently lower at NICO

Lake than at Yellowknife over the common period of record. Overall, the 12 month period of monitoring indicates lower rainfall is received at Nico Lake than at Yellowknife.

Monthly mean temperatures ranged from -26.0°C in December to +14.5°C in July. The average temperature over the reporting period was -5.4°C. The extremes recorded over the reporting period were a high of +28.6°C in July and a low of -38.5°C in December. The average temperature at the NICO Project was 0.7°C cooler than at Yellowknife (-4.7°C). For the monitoring period, Yellowknife was 0.4°C cooler than the 30 year average (1971 to 2000).

Wind was predominantly from the south-southeast. Seasonal variability was observed with frequent winds from the west-northwest in the fall and from the north in both the spring and summer. Wind predominance is influenced by large scale weather patterns and local topography. The shift in wind predominance from north in the summer to west-northwest in the winter is associated with the northward (summer) and southward (winter) migration of the mid-latitude jet stream.

Relative humidity is a measure of the amount of water vapour present in the air at a given temperature, relative to the amount of moisture that could be present in vapour form in the atmosphere at the same temperature. Average monthly relative humidity values ranged from 56.4% in June to 90.2% in October 2004. Morning humidity readings are typically higher than afternoon readings due to a lack of available heat energy (*i.e.*, as the temperature rises, the amount of moisture the atmosphere can hold increases). If the amount of water in the atmosphere remains constant and the temperature rises, relative humidity will fall.

Solar radiation ranged from an average of 1.4 watts per square metre (W/m<sup>2</sup>) in December to an average of 247.4 W/m<sup>2</sup> in May. The average hourly radiation was greatest in May as this was the month with the greatest total amount of recorded solar energy.

**8.5 If "No", are such studies being planned? Briefly describe the proposals.**

Not applicable.

**8.6 Has authorization been obtained or sought from the Department of Fisheries and Oceans for dewatering or using any waterbodies for containment of waste?**

No impact to fish is anticipated as the small water bodies in the proposed tailings basin are not fish bearing.

**8.7 Please attach an outline briefly describing any options or alternatives considered or rejected for the various mine components outlined in this questionnaire (e.g., mill site, water supply sources, locations for ore and waste piles).**

Water Source

Three potential water withdrawal locations are being considered: Lou Lake, Burke Lake, and Chalco Lake. Lou Lake alone was initially the preferred source of water for the project; however, upon further consideration of regulatory criteria and available withdrawal volumes, it was concluded that Lou Lake is only marginally suitable and may not meet the total demand for processing and potable water supplies. Consequently, it is proposed that all three sources be considered as potential withdrawal locations with no withdrawal limits exceeded from any individual source. An overview of the decision process and the hydrological information follows.

DFO (*Protocol for Winter Water Withdrawal in the Northwest Territories*, 2005) recommends that total water withdrawal from a single water body during the ice covered period should not to exceed 5% of the available liquid volume. In 2008, this guideline was revised to 10% of the under ice volume.

In winter, the available liquid volume is the liquid volume below the ice surface, calculated using the appropriate maximum expected ice thickness (1.5 m). For Lou Lake, the total volume (ice free) was calculated to be 12.8 Mm<sup>3</sup>. The volume of water below 1.5 m of ice was calculated to be 10.2 Mm<sup>3</sup>. Thus, the allowable volume that could be pumped from Lou Lake in winter is approximately 1.02 Mm<sup>3</sup>.

During construction, the annual water demand for the project is expected to be approximately 83,220 m<sup>3</sup>. Thus, 41,610 m<sup>3</sup> would be required during the six month winter period. This amounts to about 4% of the available supply from Lou Lake. During operations, the required annual volume for process and general-use water is calculated to be 2.226 Mm<sup>3</sup>. Over the six month ice covered period, the required volume for operations would be 1.113 Mm<sup>3</sup>, which is slightly more than the allowable volume, 1.02 Mm<sup>3</sup>, from Lou Lake. If used in combination, the allowable volume for Lou, Burke, and Chalco lakes is 1.266 Mm<sup>3</sup>. Lou Lake and Burke Lake could provide 1.231 Mm<sup>3</sup> and appear to be the best combination to address process water requirements. Of course, this analysis assumes that the availability of decant or reclaim water is negligible, which is a worst-case scenario, likely only during the first and/or second year of operations, or in the situation where the technologies listed in section 1.8 are unsuccessful. The assessment of water supply from these lakes also assumes negligible inflow and outflow in the winter.

#### Mine Material Management:

An initial site selection study for the location of the tailings basin was undertaken. The watersheds other than those selected (TMA Grid Ponds and MRMA) were discounted for one or all of the following reasons: Economics – primarily haul distance; and, Aquatics – baseline investigations indicate that the Grid Ponds and the small unnamed ponds in the selected basins do not support fish life. The other topographic lows containing water bodies had better water quality and supported fish habitat.

Fortune also considered the scenario where two separate tailings ponds were constructed in the areas represented by the MRMA and the TMA. In this scenario, the separate tailings ponds, referred to the South Tailings Management Area (now the MRMA) and the North Tailings Management Area (Grid Ponds) would be constructed in different years. The South TMA would be utilised in years 1-5 of operation, followed by the North TMA, subsequent to construction in year 6. Waste rock would be mined and placed upstream of the two tailings management areas to catch any run-off. While this design had advantages with respect to the ability to initiate reclamation in the South TMA after year 6, as well as the delay of capital cost construction of the North TMA, this scenario was found to be more problematic under other important design considerations. A major disadvantage was the difficult scheduling of the waste rock placement within a very small area to the north of the two operating TMAs while, or before, tailings are placed. Other problems included: extended haul routes for waste rock placement above the North TMA; containment of runoff from the North TMA waste rock dump prior to construction of the dam in year 6; water balance issues in year 6; increased piping costs; increased management and treatment requirements; and the potential identification of the wet land south of the proposed MRMA to be considered fish habitat.

#### Mill Site and Bulk Sample Portal and Decline Location:

Initially, a portal located to the southwest of the deposit was conceived. The original mill site was located adjacent to this portal. The site became undesirable because of increased reserves, and plans for greater open pit mining. Consequently, as the economic studies advanced, the portal was moved to its present location east of the deposit. The mill site was moved in tandem. The main reason for the relocations was to allow for more containment capacity in the MRMA.

Following completion of the Feasibility study in February 2007, the mill site was moved away from the location south of the portal. This move was undertaken as front end engineering commenced in order to ensure that all major infrastructures were located at least 800 meters from the pit (blasting), and that the plant was built on competent bedrock. The movement of the plant also opens up a significant area above the TMA containment where low grade waste rock can be stored until it is economic to process. There is approximately 9.8 Mt of this material, and placement of the material in this location will allow for run-off to

be collected and returned to process.

#### Camp Location

The camp was originally planned to be near Lou Lake due to the available water supply and aesthetics of the area; however, Lou Lake will not meet the minimum requirements as a sole source of fresh water for the project, and is too distant from the core mining operations.

Ultimately, it was decided to locate the camp close to the plant site in order to simplify water and sewage systems through the use of utilidors. The final location selected will also minimize the development footprint of the mine site.

#### Airport

The permanent airport was to be located between Hislop and Rabbit Lakes. The new location near Burke Lake is preferred because it is flatter ground and is closer to the mine site and camp.

#### Power Plant and Heat Recovery

NICO will be designed with its own diesel-fired power plant and heat recovery systems, along with the appropriate switch gear to allow for delivery of grid power at a future date (if, and when, it becomes available), and boiler systems capable of heat generation when excess heat is no longer or only partially available from the installed generators.

#### Waste Disposal

Fortune considered backhauling all landfill wastes from the site to an existing municipal waste disposal site in Yellowknife; however, Fortune believes that Yellowknife does not have capacity for additional waste streams. As a result, Fortune designated a location within the proposed TMA and will use waste rock as an interim cover and final cover. Attachment 5 contains the General Waste Management Plan.

### **8.8 Has a socio-economic impact assessment or evaluation of this project been undertaken? (This would include a review of any public concerns, land, water and cultural uses of the area, implications of land claims, compensation, local employment opportunities, etc.)**

Yes. An initial study was completed. Further work is ongoing.

### **8.9 If "Yes", please describe the proposal briefly.**

A socio-economic impact assessment was undertaken in the vicinity of the proposed NICO project, specifically in the communities of Whatì, Behchokò, Gamètì, Wekweètì, and the City of Yellowknife (Rescan Environmental Services Ltd., February 2006). Fortune is in the process of updating the socio-economic study.

The results indicated that, with the exception of Wekweètì, the communities have been increasing in population. Projections indicate declining populations in Wekweètì and Gamètì over the next 15 years. In recent years, crowding in houses has decreased, home ownership has increased, and household core needs have decreased in response to increasing household incomes. Education levels have also increased steadily since 1989. This is attributed to the Government of the Northwest Territories' (GNWT) policy of grade extensions, access to higher level education, and more culturally related studies and stay-in-school programs. Where education levels appear to have dropped, the cause is attributed to out-migration. The increased education and increased employment opportunities from resource development has resulted in higher employment rates in all communities.

Despite this, unemployment rates remain high due to higher participation rates. Local economies in the

small communities continue to be dominated by traditional activities (such as hunting, trapping, harvesting, fishing); and participation in these activities has been increasing, possibly due to increased incomes providing the ability to purchase equipment and due to rotational work schedules. Yellowknife's economy is more diverse and has grown with the opening of new mines.

Language use has remained stable in Gamètì and Wekweètì, but has appeared to be declining in Whatì, Behchokò and Yellowknife. Single parent families are increasing. However, teen births appear to be declining in response to family planning education. Sexually transmitted diseases have been increasing in Gamètì and Wekweètì, possibly due to increased alcohol and drug use, higher incomes, rotational parenting and unsafe sex practice. Suicides have been on the increase in the Northwest Territories, but due to the overall small numbers, conclusions are difficult to draw. Rates of violent crime have increased. This is attributed to increased alcohol and drug use.

Potential impacts are considered to be similar to those already experienced in conjunction with other similar projects in the area, in particular, the diamond mines. Fortune's north-first hiring and procurement policy will focus potential benefits on the local economy. The potential beneficial impacts could be substantial with respect to job creation, income increases and associated measures (such as quality of life, purchasing power, housing improvements, business development, training opportunities and decreased welfare dependency). These benefits lead to social improvements including increased self-esteem, health, and community capacity. The project will also lead to infrastructure enhancements, including the all-weather road, which will lower the cost of importing goods to the communities that now rely on winter roads, as well as facilitate travel.

Rotational shift work and increased income have had positive impacts on some traditional activities. The proximity of the NICO mine to the Tłìchò communities provides an opportunity for more attractive work schedules than the more distant diamond mines.

Potential negative impacts observed in relation to the previous mine developments include increased prices of some goods due to increased income availability, increased drug and alcohol use leading to increased crime rates, increased sexually transmitted disease, increased lone or rotational parents, income inequalities and pressure on support services. Some of these impacts are not due solely to resource development as they were problematic and increasing.

The impact on land use and harvesting as a result of the NICO development and infrastructure is considered to be low due to the relatively small areas affected and the site's distance from the known harvesting areas. A heritage resource impact assessment by Golder (2003) found that the development would have minimal impact due to limited cultural material present and its recent age.

Whether the net impact is beneficial or negative is best assessed by the communities affected; however, although mixed, a net benefit is anticipated with no *new* negative impacts as a result of the NICO project.

## SECTION 9 -- LIST OF ATTACHMENTS

<b>Attachment</b>	<b>Question #</b>	<b>Title</b>	<b>No. Pages</b>
1	n/a	Glossary of Terms, List of acronyms, List of units	4
2	5.19 and 7.5	Conceptual Closure	6
3	7.2 and 8.2	Screening Level Environment Impact Assessment including the electronic version of "Environmental Surveys at Fortune Minerals Limited NICO Deposit, 1998 – 2004." Report.	22
4	7.4	Emergency Preparedness and Response Plan	5
5	LUP #9	General Waste Management	3
6	LUP #6	Community Consultation Records	29