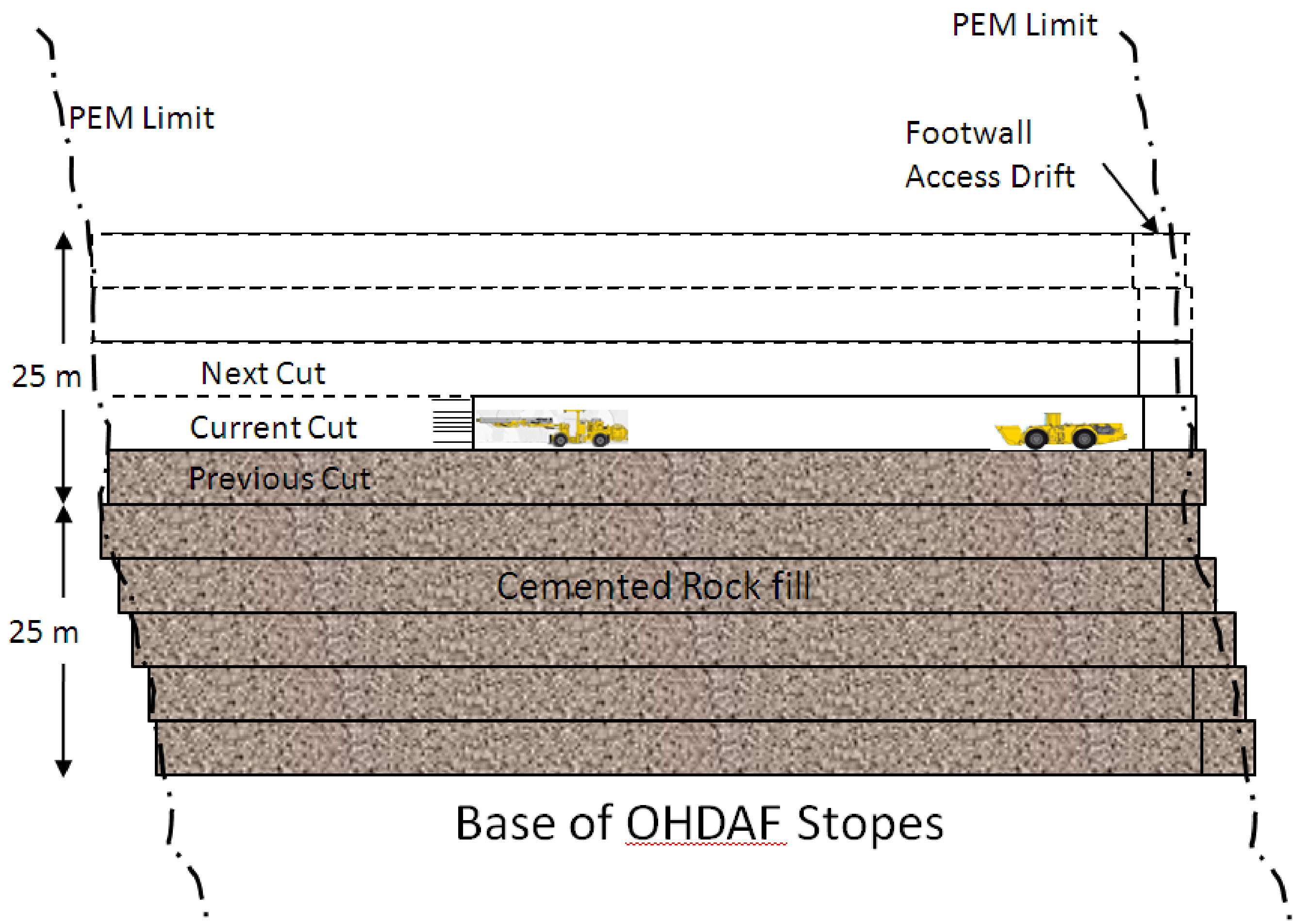

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Data Sources: *Areva Resources Canada Inc.*

FIGURE 5.5-3
TYPICAL CROSS SECTION OF THE OVERHAND DRIFT-AND-FILL MINING METHOD

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5.5.3.2 ***Mine Design***

The complete mine design is shown in Figure 5.5-4 which shows the portal access on surface (165 mASL) and the main decline that extends down 210 m to a main access near the top of the ore zone (-45 mASL). The mine design is shown with the local geology in Figure 5.5-5 (longitudinal section view) and Figure 5.5-6 (NE - SE view).

A ramp is the preferred option for access to the underground workings due to the depth of mineralization, planned production rate and the need for an internal ramp when using the OHDAF method. However, a shaft was also a possible option considered. A shaft would be required from surface to 420 m depth but an internal ramp would still be required from 200 m to 420 m depth to provide equipment access for production mining.

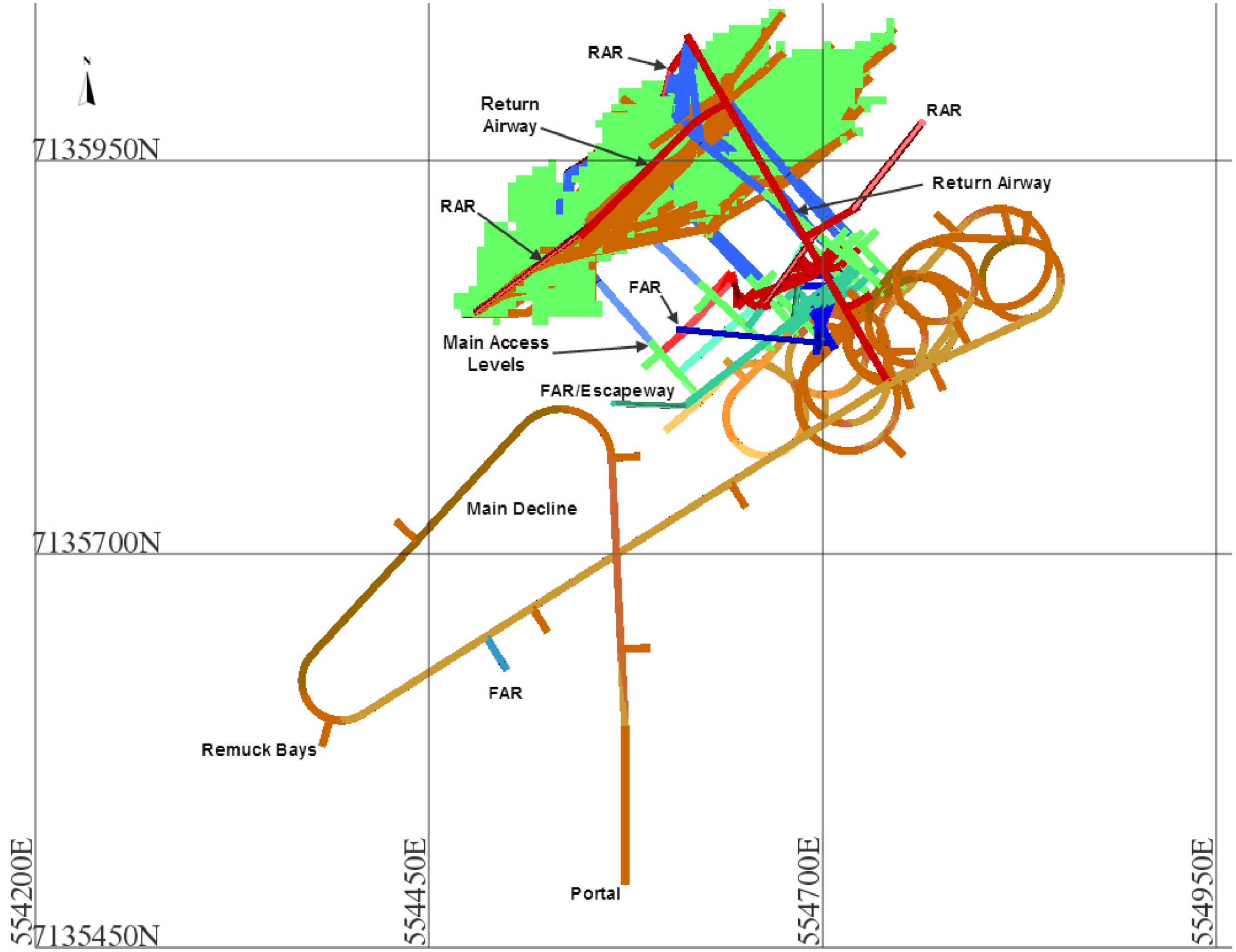
The plan for the End Grid deposit is to provide access by a portal and single ramp driven from surface (165 mASL). This ramp will accommodate all access for personnel, mobile equipment and truck haulage and also acts as one of the two fresh air sources for the mine. The portal will be located to the South of the deposit and, since it will act as a fresh air source, will be situated a sufficient stand-off distance (~200 m) from the main exhaust raise.

The 5 m wide by 5 m high decline will be 3.3 km long with an overall gradient of -13%. Main access levels are planned at every 25 vertical meters with the top level at -45 mASL and the bottom sill at -245 mASL. Each main access will have a re-muck bay, connected to an exhaust raise, a fresh air intake and escapeway connection, a truck loading bay and cut-outs for sumps and mine power.

An average production rate of 840 tonnes per day is planned for the End Grid mine for operating nearly 365 days per year. This productivity assumes two production mining (jumbo) crews, one each on two active production horizons (or main access levels) and a third crew backfilling on a third horizon.

Material from the mining operation will be separated into mine rock, mineralized material below economic cut-off at the time of mining and ore. This will be done using a combination of geologic grade control and load-haul-dump or truck bucket scanning. The level stockpiles will require regular drawdown to ensure adequate space for material segregation. Once hauled to surface each of the three material types will be placed in a corresponding temporary stockpile prior to transfer to the main Sissons stockpiles (Technical Appendix 2H).

At a 2,000 ppm U cut-off the End Grid underground mine is currently estimated to yield approximately 2,710 kt of ore averaging approximately 0.31%U for a total resource of about 8,460 tonnes of contained U metal (approximately 22 million lbs U_3O_8 equivalent); and 1,400 kt of Type 3 mine rock.



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Date: 09/01/2011 Scale:
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Data Sources: *Areva Resources Canada Inc.*

FIGURE 5.5-4
END GRID MINE DESIGN – PLAN VIEW

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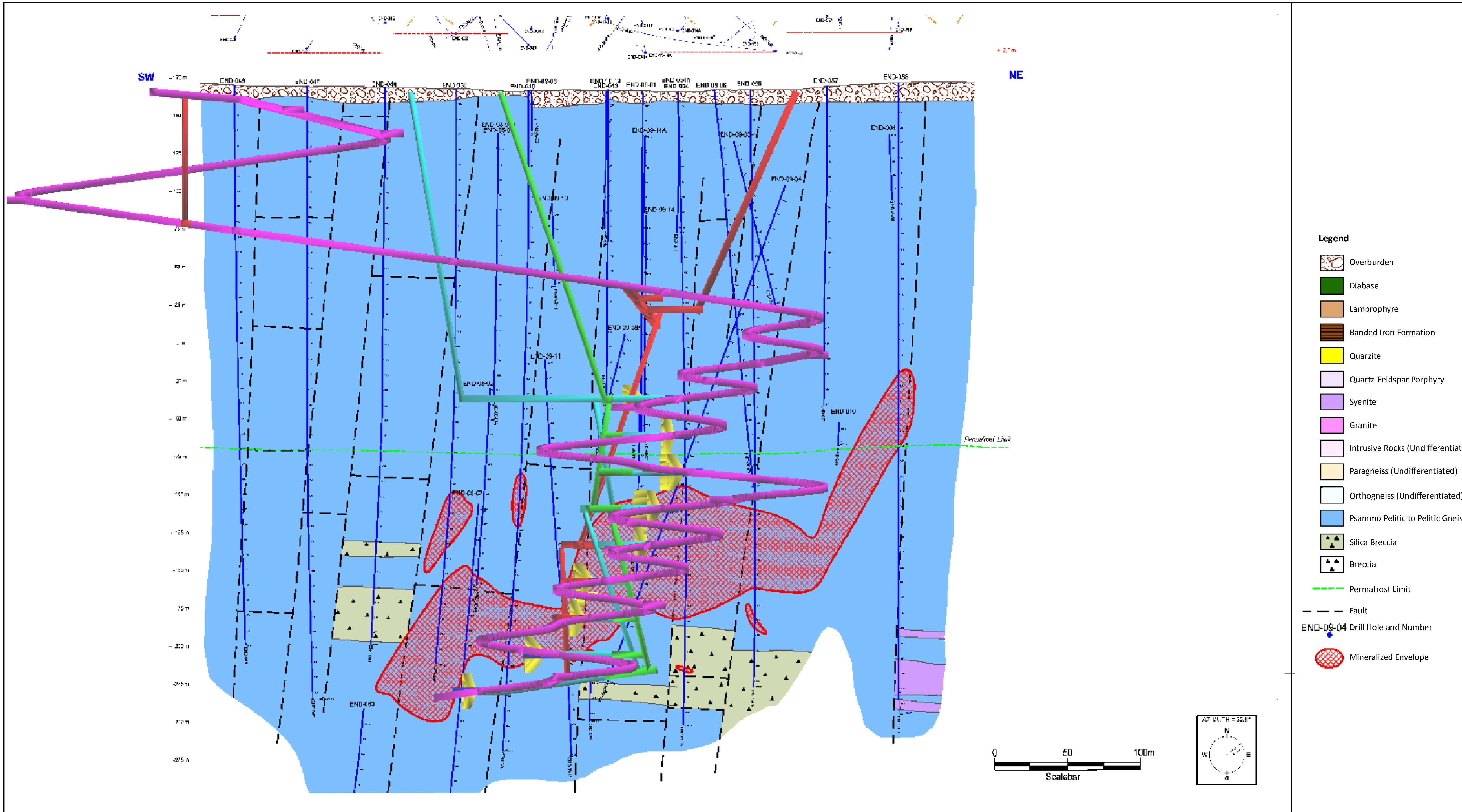

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Date: 05/06/2013 Scale: Not to Scale
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Data Sources: *AREVA Resources Canada Inc.*

FIGURE 5.5-5
END GRID LONGITUDINAL SECTION VIEW

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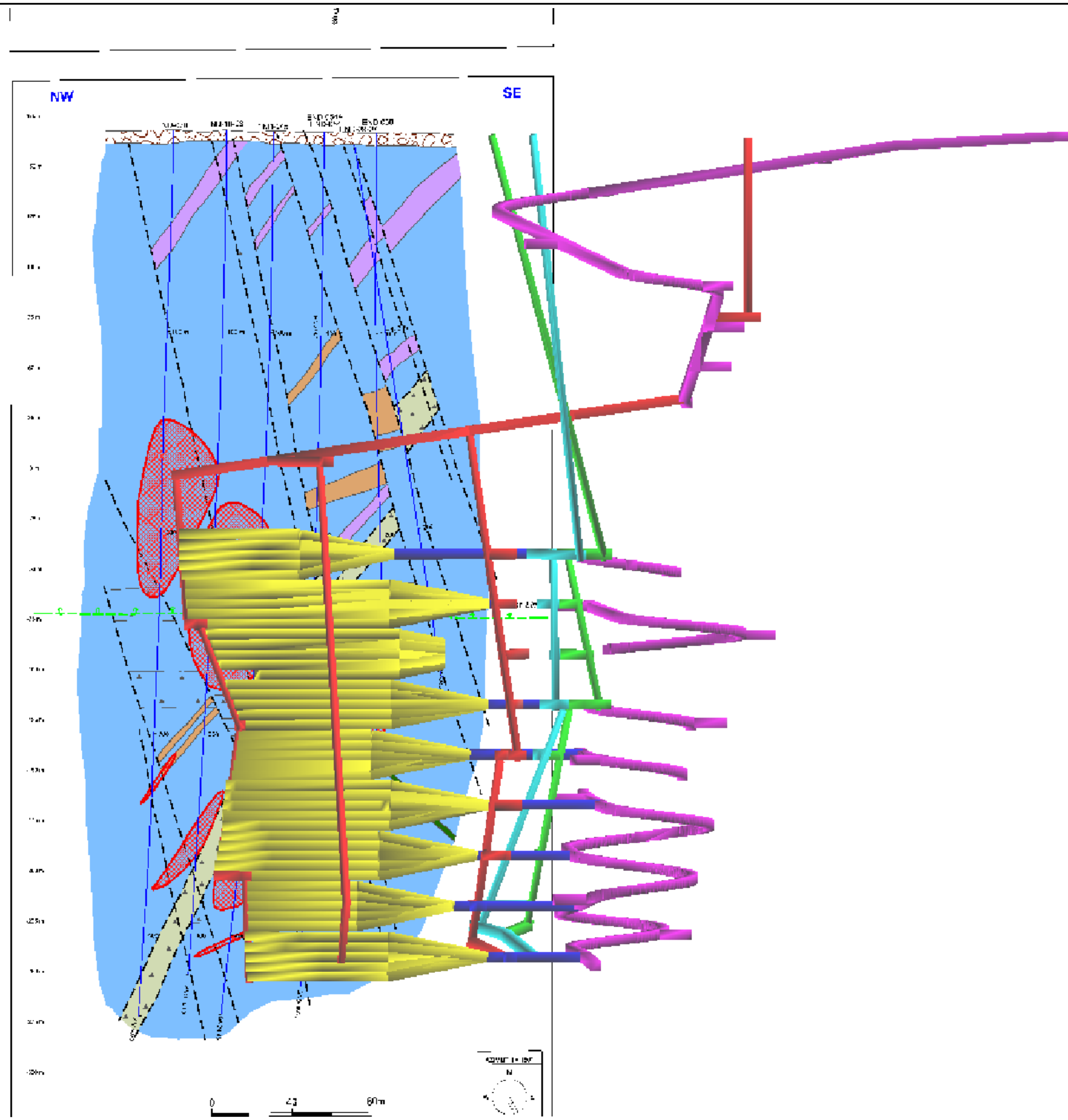

Legend
- Overburden
- Diabase
- Lamprophyre
- Banded Iron Formation
- Quartzite
- Quartz-Feldspar Porphyry
- Syenite
- Granite
- Intrusive Rocks (Undifferentiated)
- Paragneiss (Undifferentiated)
- Orthogneiss (Undifferentiated)
- Psammo Pelitic to Pelitic Gneiss
- Silica Breccia
- Breccia
- Permafrost Limit
- Fault
- Drill Hole and Number
- Mineralized Envelope

Projection: NAD 1983 UTM Zone 14N
Creator: B. Derrow
Date: 05/06/2013 Scale: Not to Scale
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Data Sources: *AREVA Resources Canada Inc.*

FIGURE 5.5-6
END GRID MINE DESIGN NW - SE VIEW

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5.5.3.3 *Drilling and Blasting*

When necessary, development and production drilling will use a recirculating brine solution to prevent drill rods from freezing in the permafrost. The water distribution network may need to be fully insulated and heat traced to ensure a regular feed of water to workplaces.

A conservative powder factor of 2 kg/tonne of ammonium nitrate and fuel oil (ANFO) is estimated for underground blasting. Blasting will be primarily in 5 m wide by 5 m high development rounds with 50 to 60 – 48 mm diameter blastholes that are 3.6 m long (with 3 to 4 empty cut holes reamed up to 89 mm).

Underground explosive management systems and procedures will be implemented to minimize the concentrations of ammonia and nitrate levels in mine effluents. ANFO has no water resistance and will dissolve readily in water so spillage and blowback during loading operations will need to be closely controlled. Wet holes will primarily be loaded with a packaged emulsion product rather than ANFO.

5.5.3.4 *Material Handling*

Mine rock and ore handling at the End Grid underground mine will be by load/haul/dump units (LHD) and 45-tonne haul trucks. Mine rock from development drifting will be loaded into one truck and hauled up-ramp to a temporary storage location near the portal. As the main decline is advanced, temporary rock storage bays will be developed at intervals along the decline to facilitate mucking and truck loading. Similarly, mine rock from main access development will be handled with the same equipment and placed in the corresponding surface stockpiles.

Ore will be scanned and then hauled to surface for placement on the ore transfer stockpile.

5.5.3.5 *Mine Ventilation*

Ventilation is a critical factor in an underground mining operation. The ventilation air flow requirements are based on operating diesel engine requirements and ensuring adequate radon dilution. Single pass ventilation will be practised in all ore headings and air will not be re-used. The mine ventilation requirements are based on a design rate of 4.75 m^3/min per effective kW (125 cfm/hp) of operating equipment. The proposed mining fleet translates to 2,600 kW of installed diesel equipment effectively working at any time (Table 5.5-1).

Table 5.5-1 Underground Equipment Ventilation Requirements

Mobile Equipment	Required	Unit kW	Gross kW	Mechanical Availability	Job Utilization	Effective kW
Production Loaders	2	263	526	0.85	0.85	380
Development Loaders	1	263	263	0.85	0.85	190
Rammer-Jammer Loader	1	263	263	0.85	0.85	190
Production Haulage Trucks	2	437	875	0.85	0.85	632
Development Haulage Trucks	1	437	437	0.85	0.85	316
Backfill Truck	1	437	437	0.85	0.85	316
Production Drill Jumbo	2	110	219	0.85	0.25	47
Development Drill Jumbo	1	110	110	0.85	0.25	23
Explosive Truck	1	110	110	0.85	0.30	28
Production Bolter	2	110	219	0.85	0.25	47
Development Bolter	1	110	110	0.85	0.25	23
Grader	1	123	123	0.85	0.50	52
Shotcrete Machine	2	110	219	0.85	0.25	47
Scissorlifts	2	86	172	0.85	0.30	44
Fuel Truck	1	110	110	0.85	0.50	47
Boom truck	1	37	37	0.85	0.30	9
Jeeps	4	67	268	0.85	0.55	125
Surveyor Jeep	1	67	67	0.85	0.30	17
Mechanics truck	1	110	110	0.85	0.30	28
Electrician Truck	1	110	110	0.85	0.30	28
Personnel Carrier	1	110	110	0.85	0.25	23
Totals	**30**		**4892**	**0.85**	**0.63**	**2612**

Using this conventional approach the total fresh air to be supplied to the underground workings is estimated to be 209 m^3/s; however, in addition to this minimum requirement, air will also be needed to dilute and control radon in the workings. It is planned to separate the decline ventilation from the access levels and production mining areas by installing ventilation doors at each level access from the main decline. Thus fresh air entering through the main decline will not re-used to ventilate levels or stopes. This will also ensure that potential radon contaminated air from the production areas will not enter the main decline system.

Each active level will receive 73 m^3/s to control radon dispersion in the ore stockpile bay (13 m^3/s) and to ventilate up to four active ore headings (one at 21 m^3/s (loader) and three at 13 m^3/s (jumbo, bolter or open heading)) for a total of 219 m^3/s over three active levels. The active headings will initially employ a “pull” ventilation system that uses rigid ventilation ducting that exhausts the face to the central exhaust raise. The negative pressure developed in the rigid ducting by the main surface exhaust raise fans will pull fresh air from the fresh air raise to the headings. A bulkhead positioned at the exhaust raise access will ensure separation of airflows. Once the west footwall drift intersects the westerly located exhaust raise this side can be converted to a flow-through ventilation circuit and ducting will only be required in the active ore headings. The level sump will also require a small fan to exhaust any radon generated from the mine water back to the exhaust raise. In addition to this, 100 m^3/s is needed for up to three trucks in the decline giving a total mine requirement of approximately 320 m^3/s.

Based on the anticipated mine pressure drop it is estimated that 600 kW of surface fan power will be needed. It is proposed to ventilate the mine using one main exhaust fan installation comprising 2-300 kW fans installed in parallel. A primary exhaust raise will consist of a 5 m diameter raise borehole from surface to an exhaust ventilation airway at the -50 m Level. A secondary exhaust raise will also be installed to connect the stockpiles on each main access level to this exhaust airway. The fresh air intake will consist of the main decline portal and a 4 m diameter raisebore hole.

The fresh air raise will also contain an escapeway ladder for secondary egress from the mine in case of emergency. The proposed system consists of a pre-fabricated ladderway that is bolted to the shotcreted wall of the raise.

The distance between the fresh air intakes and the exhaust fans will be approximately 200 m to ensure contaminated air does not re-enter the mine workings. A total of approximately 1,400 m of vertical raise development will be required for the mine.

It should be noted that the total fresh air requirements will vary over the life of the mine with a peak when the mine is fully developed and production rate is at a maximum.

The End Grid mine ventilation network is a functional design that can move adequate volumes of air through the mine workings as modelled. The ventilation system will require ongoing design revisions and optimizations to maximise the efficiency and effectiveness of the network and components such as fans and raise locations. It will also require daily management of air flows in active headings depending on the scheduled activities. This can be done using variable speed fans and/or adjustable dampers in the rigid ductwork.

5.5.3.6 *Mine Heating*

It is proposed to heat the mine air to maintain an air temperature slightly above freezing to 2°C It is estimated that a total of 160 Billion BTU of heat will be required per year (over 8 winter months) to maintain a fresh air temperature slightly above freezing. Heaters will be required at both the portal fresh air fan building and at the fresh air raise building.

5.5.3.7 *Mine Water Management*

Water management is a critical element of the safe and efficient operations of any underground mine. Uranium mines have the additional challenge of preventing water build-up in the mine as a radon control measure, since groundwater hosts radon and can potentially circulate radon through the mine openings. The operating philosophy of this management system is to ensure that water collects only in a well-ventilated main sump located at the bottom of the mine. The mine water collected in this sump is pumped to surface and treated. The main sump is vented directly to the exhaust air system to minimize the accumulation of radon in the sump and the mine.

Mine water at the End Grid mine will include three components:

- groundwater inflows to the mine;
- technical water (i.e., drilling, wall and mine equipment washing, dust suppression, backfill, combustion); and
- surface water entering the mine.

The sum of these three components is not expected to exceed 1,000 m^3/day for the final mine configuration or peak production phases of the operation.

Groundwater Inflows

Although End Grid extends into the groundwater regime beneath the permafrost, groundwater inflows into the mine openings are expected to be small due to the low hydraulic conductivity of the rock mass. Model results suggest that inflow to the End Grid underground mine may vary between approximately 30 m^3/day and 160 m^3/day as development progresses below the permafrost horizon.

Technical Water

During development and production, technical water requirements are estimated to be 205 m^3/day, approximately. This water will come from a number of sources within the mine however the drilling jumbos and bolters are the prime source of the technical water. Of the total technical water generated most is produced by drilling and a small fraction (15 m^3/day, approximately) is generated from the combustion of diesel fuel by the mobile fleet in internal combustion engines.

In mine areas within permafrost, the drill process water will be a brine mixture, which will be used to maintain the frozen ground and to cool the drilling string. This solution will be captured and treated with all of the other sources of water in the End Grid mine.

Surface Water

Surface water run-off into the mine will be abated through surface infrastructure, such as ditching, berming and collection ponds. On surface, all of the fans will be installed on graded pads with curbs to prevent water inflow via the raises. The portal will be a decline, and surface water will be captured at the portal by designing the entrance with a short run at a positive gradient which will drain into a small sump. This water will be pumped into the main water treatment system on surface. Surface water that does reach the mine will be confined to the ramp and managed in the ramp water handling system.

The management and treatment of water collected at the Sissons site is discussed in further detail in Section 9.

Mine De-watering System

The underground dewatering system will be comprised of a series of drainholes, pumps and pipelines to control water in the mine and to convey the mine water to surface.

The individual mine levels are designed to manage and control groundwater and technical water through the development gradients, excavating sumps and a network of drainholes supplemented by piping. Each sump on each level will be connected to the other by means of a drain hole that will be drilled as each of the levels are developed. All of the water will flow to the sump located just before the start of the primary ore access cuts. The initial two declines will require a small wall slash to collect water that will then be pumped to the sump and into the main drainage system for the mine.

The proposed mine water management system will handle all of the technical and groundwater in the mine. Inflows from surface will be managed either by complete abatement, such as curbs around fan installations, or by a ditch system in the main ramp. Water in the ramp is expected during the spring run-off season and heavy rainfall events. All of the water in the ramp will report to the main sump on the bottom level.

The mine extends to approximately 400 m below topography. Sufficient pumping capacity will be installed to move the water from the main sump to the treatment area on surface.

Depressurization

During the development of the decline in permafrost, pressure heads beneath the permafrost will not be reduced; therefore, if prior depressurization is not undertaken, pressure heads will initially be 250 m or more when the mine first penetrates beneath the permafrost. If these pressures are determined to be high, based on stability concerns, or the inflow volumes are considered to be difficult to manage, then a depressurization program will need to be implemented.

An effective depressurization system would likely consist of vertical or sub-vertical boreholes drilled from a development excavation. The boreholes could either be allowed to flow under artesian conditions, or pumps could be installed (if larger diameter holes are drilled) to provide additional lowering of water levels. The water would then be pumped to surface for treatment.

Detailed designs for such depressurization systems, and for handling the inflow water, and the containment of the radon gas likely to be associated with this inflow, are not considered to be necessary at this stage given the predicted low groundwater inflows. Depressurization and radon control procedures, if necessary, will be described at the licensing stage. This will include analyses of the rates of depressurization that are achieved with vertical boreholes, and determinations of the number of boreholes, the lead time required to drill the holes, and the inflows that will result.

5.5.3.8 *Electrical System*

Underground electrical power demand for the End Grid mine is estimated at 1.6 MW.

Electrical power will feed the underground loads using two feeders installed in two boreholes from surface. This will provide redundant power supply to the underground mine in the event that one feeder fails. However, in order to develop the main decline, a cable will be installed as the ramp is developed using messenger wires. This cable will be continually extended as the ramp progresses to the time when a Mine Power Centre (MPC) can be installed. The MPC will be centrally located underground to minimize long cable runs to Mine Load Centres.

The estimated electrical power demand for surface equipment required for the End Grid Mine is 1.2 MW, approximately. The surface loads will be supplied from the surface 4,160 V switchgear. A standalone back-up generator will be available to provide sufficient emergency power to the mine if primary power fails (see Section 11).

6 Mine Rock Management

6.1 Concept

Mine rock is the material that must be excavated to gain access to an ore body during mining operations. The proposed mine rock management plan for the Kiggavik Project has been designed according to the following principles:

- to avoid interaction between mine rock and natural water bodies;
- to maximize the use of suitable mine rock as construction material; and,
- to ensure the long-term protection of the terrestrial, aquatic and human environment surrounding the Kiggavik Project.

6.2 Characterization

An assessment of the mine rock material that will be excavated from the Kiggavik and Andrew Lake open pits and the End Grid underground mine during the mining operation is provided in Technical Appendix 5F Mine Rock Characterization and Management. A summary is provided in this section.

A portion of the mine rock material that will be produced during the mining operation contains small quantities of naturally-occurring sulphide minerals which were conservatively assessed as having potential acid generation and/or metal leaching behaviour. An investigation was undertaken to characterize and evaluate the mine rock material to be generated at both the Kiggavik and Sissons sites. The results of this investigation were intended to characterize the elemental makeup and acid generating potential of the mine rock material, as well as produce estimates of the loadings of constituents of potential concern (COPC) from the mine rock in order to assess the potential to affect water quality both during and after operations.

Static testing was completed on samples of mine rock material that included elemental solids analysis and acid base accounting (ABA), from both the Kiggavik and Sissons sites. Kinetic tests in the form of simplified Special Waste Extraction Procedure (SWEP), laboratory humidity cells, field test cells, and columns with submerged rock were also completed on the various materials. The test results were used to establish segregation criteria for the expected mine rock material in order to determine the appropriate management strategy or potential use (i.e. construction) of the material. Humidity cell results were used to calculate anticipated loading rates in the field, according to the mine rock type, by applying adjustments for grain size and temperature. The field loading rates were used to estimate COPC loads and pore water concentrations expected from the permanently stored

Kiggavik and Andrew Lake rock in the proposed stockpiles. Currently only mine rock excavated during development activities for the End Grid mine will be stored in the permanent stockpiles.

6.3 Segregation Criteria

Evaluation of the mine rock characterization test work suggests that, if properly segregated at the source and managed, the mine rock materials will not pose an environmental risk. Based on the geochemical evaluation results, segregation criteria for mine rock are proposed as follows:

- Type 1: for mine rock that can be used as construction material (i.e., access road, pad, fill, etc.), it is proposed that concentrations of uranium be less than the unconditional clearance level as per the *Nuclear Substances and Radiation Devices Regulations* issued pursuant to the *Nuclear Safety and Control Act* (1 Bq/g which equates to approximately 80 ppm based on the specific activity for U_{nat}) and have total sulphur content less than 0.1%;
- Type 2: for mine rock that can be permanently stockpiled and managed above ground, it is proposed that uranium contents be less than 250 ppm and total sulphur content be less than 0.1% that ensures that the neutralizing to acid generating (NP/AP) ratios for the rock remain greater than 2 and that there is no risk of acid generation; and,
- Type 3: for mine rock that requires specific management (e.g. in-pit disposal), it is proposed that all material not described as Type 1 or 2, nor considered to be ore be included in this category.

These segregation criteria are consistent with mine rock management practices developed and successfully applied in northern Saskatchewan.

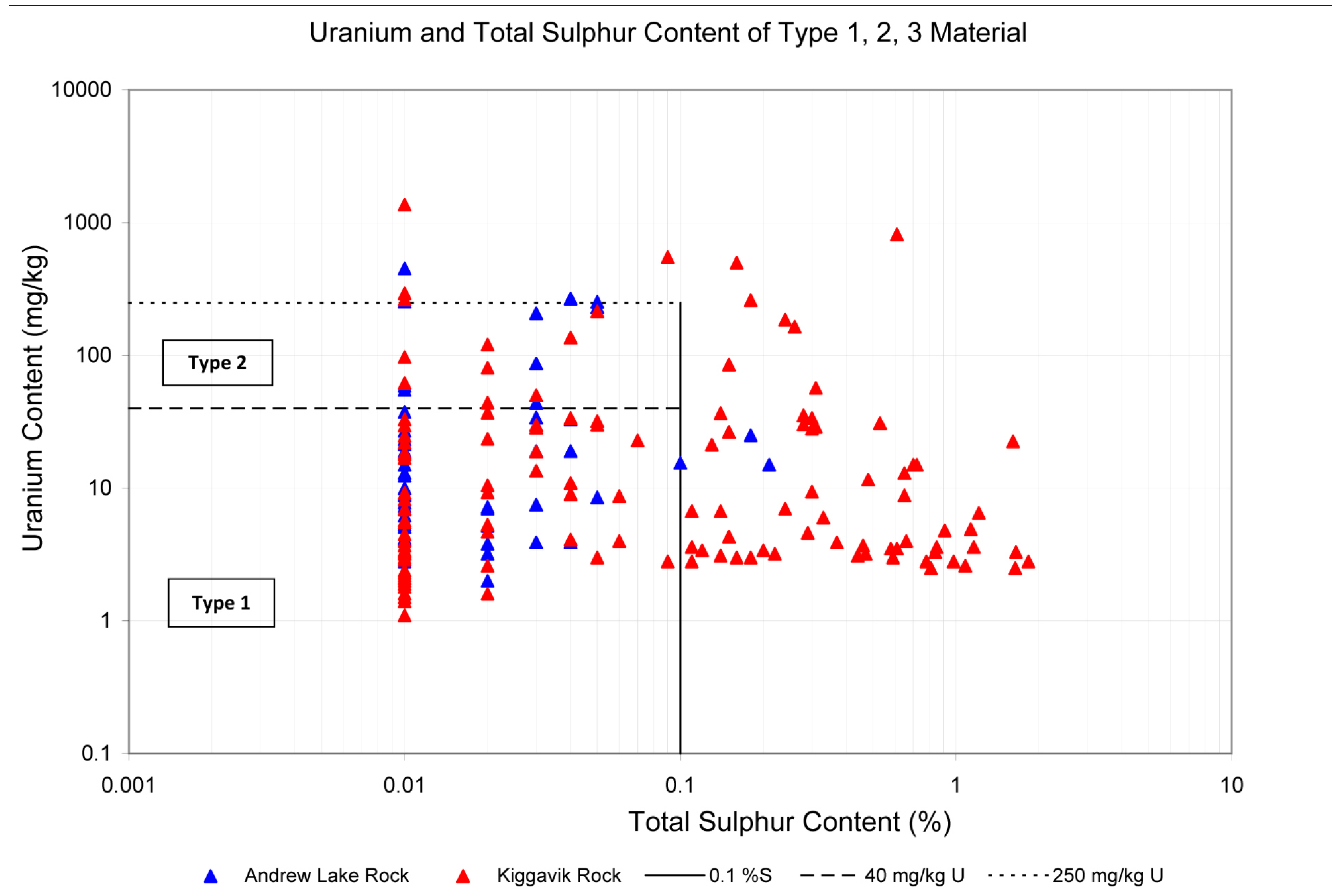
The three types of mine rock material expected at the Kiggavik and Sissons sites were assessed for potential effects on site water quality in accordance with the proposed management strategies. The application of segregation criteria outlined above, result in the following average characteristics for each mine rock type.

- The Type 1 mine rock material from the Kiggavik open pits is predicted to be non-acid generating with a geometric mean (geomean) total sulphur content of 0.019% S. The Type 1 mine rock material from the Andrew Lake pit is also predicted to be non-acid generating with a geomean total sulphur content of 0.015% S. End Grid material is predicted to similar to the Andrew Lake material. The loading rates for the Type 1 and Type 2 materials were derived from the same kinetic tests because the bulk-constituent contents were nearly identical, with the exception of uranium. The use of the Type 1 mine rock material for construction purposes will not present a risk of acid generation or substantial metal leaching.

- The Type 2 mine rock material from the Kiggavik open pits is expected to be non-acid generating with a geomean total sulphur content of 0.02% S. The Type 2 mine rock material from the Andrew Lake pit is expected to be non-acid generating with a geomean total sulphur content of 0.016% S. End Grid material is predicted to be similar to the Andrew Lake material. The loading rates calculated for the Type 2 material were used to calculate the expected pore water concentrations in the permanent stockpiles. Results show that the permanently stored Type 2 mine rock material will not present an acid generating issue, although the initial contact water may require management prior to discharge in order to reduce any COPC concentrations that may exceed water quality guidelines.
- The Type 3 mine rock material is predicted to have the potential to generate acid if stored on land. Acid generation can be mitigated if the rock is properly managed over the long term with in-pit, underwater disposal, for example. The geomean total sulphur content of the Type 3 mine rock samples from the Kiggavik deposits was 0.35% S. The geomean total sulphur content of the Type 3 mine rock samples from Andrew Lake material was very low at 0.05% S and is therefore unlikely to produce acidic drainage. Type 3 mine rock samples from the End Grid deposit were chemically similar to those from the Andrew Lake deposit. During the operational phase the Type 3 mine rock will be placed on lined pads to collect and treat the water as required. After operation, this material will be placed for the long-term in the mined out pits, and thus eventually submerged underwater as the pits fill.

Based on initial test work and assessment (Technical Appendix 5F), the proposed mine rock management strategy will have minimal environmental effects. Additional test work related to the placement of waste rock and tailings together in the same pit is detailed in Technical Appendix 5F.

The classification of the samples that were tested can be seen graphically in Figure 6.3-1.



Projection: N/A
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Data Sources: *Areva Resources Canada Inc.*

FIGURE 6.3-1
MINE ROCK CLASSIFICATION

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6.4 Segregation

During open pit mining, mine rock will be segregated according to uranium grade as determined by radiometric scanning. Operational procedures will be used to ensure that Type 2 and Type 3 mine rock are effectively categorized, separated and transported to the appropriate disposal area. Segregation of mine rock will be based on the following:

- radiometric scanning of blast hole cuttings in clean rock zones to detect anomalous radioactivity levels;
- sampling of blast hole cuttings and, if necessary, analysis by the x-ray fluorescence (XRF) method to detect anomalous metal content (e.g., arsenic, nickel);
- radiometric probing of blast holes in ore zones to define ore/waste boundaries;
- radiometric scanning of working faces during excavation to confirm blast hole scanning/probing results;
- overhead scanning of mine rock in the proximity of Type 3 material or ore once loaded onto trucks;
- daily scanning of the Type 1 / 2 mine rock disposal area to ensure that no potentially problematic mine rock or ore was inadvertently placed;
- sampling to assess acid generation potential of clean waste rock; and,
- if uncertainty as to classification exists, all questionable Type 1 or Type 2 mine rock will be considered as Type 3 for long term disposal in the Main Zone and Andrew Lake open pits as a mitigation method.

The uranium content of mine rock will be estimated in the field by radiometric techniques and subsequently confirmed with drill cutting assay results. In addition, XRF technology has been identified as a reliable field evaluation tool that can be used to directly determine potentially problematic metals in mine rock during mining. The XRF technology has greatly advanced in recent years, and it is generally accepted as a quantitative screening tool for environmental investigations and industrial site clean up activities.

To ensure that the segregation and mine rock management program is effective, random samples will be collected and analyzed during operations to confirm expected sulphur and uranium contents, and that the material is being designated to the appropriate stockpiles. In addition, follow-up programs are proposed, including a Mine Rock Optimization and Validation Program that will focus on Type 3 mine rock and will provide further insight into the geochemical characteristics of the materials to validate the proposed long-term management plan. Monitoring wells may also be installed within the backfilled rock in the pit to monitor and compare predicted water quality to predicted values within the pit and in the active zone downgradient of the permanent clean mine rock stockpiles to verify seepage and groundwater quality. Shallow active layer monitoring wells are also proposed to be installed down gradient of the Type 3 temporary stockpiles and ponds during operations to provide any early warning of water quality effects related to that material.

The above measures have been successfully used for mine rock segregation at the McClean Lake Operation and it is proposed to apply them during mining of the Kiggavik and Andrew Lake pits.

Mine rock from the End Grid mine is expected to be Type 1 and Type 2 during initial development activities and mainly Type 3 during operation. Type 3 mine rock generated from the End Grid mine will be managed with the Type 3 mine rock from Andrew Lake.

6.5 Quantities

A production mining rate for rock material has been estimated based on the proposed mine plan. Table 6.5-1 provides an estimate of material volumes. The “split” between Type 2 and Type 3 is considered a conservative estimate, such that the total volume of Type 3 material is likely over-estimated. The overburden will be comprised mainly of frozen sand and gravel, and granite boulders with some zones of silt and glacial till. The clean overburden will be stockpiled for use in reclamation while the contaminated overburden, if any, will be stored with the Type 3 mine rock and disposed of within Main Zone Tailings Management Facility during decommissioning.

Table 6.5-1 Estimated Quantities of Mine Rock and Overburden Materials

	Situ Overburden (Mbcm)	In Situ Mine Rock (Mbcm)		Mine Rock Broken Estimates (Mm^3)		
		Type 1 + Type 2	Type 3	Type 1	Type 2	Type 3
Kiggavik						
Purpose Built Pit	0.1	0.35	-	0.3	0.1	-
East Zone	0.6	2.0	0.06	0.5	2.1	0.1
Centre Zone	1.8	4.6	0.1	1.2	4.8	0.2
Main Zone	3.0	21.3	0.3	5.5	22.1	0.5
Sissons						
Andrew Lake	3.3	33.1	0.5	8.6	34.4	0.8
End Grid		0.3	0.04	0.1	0.3	0.1

Notes:
Open pits: the volume/tonnages are at a cut off grade of 900ppm and post mining recovery
End Grid: the volume/tonnage are at a cut off grade of 2100 ppm and are post mining
In situ mine rock
Volumes are in million bank cubic meters (Mbcm), in situ volume prior to excavation
Mine rock broken estimates
Volumes are in million loose cubic meters (Mm^3)
A swelling factor of 1.30 is considered
A Type 1/ Type 2 ratio of 20% is considered for all deposits
A Type 1/ Type 2 ratio of 75% is considered for the Purpose Built Pit
Type 3 estimates for broken mine rock conservatively include a 30% contingency

6.6 Mine Rock Disposal

6.6.1 Kiggavik Site

6.6.1.1 *Permanent Stockpiles*

Mine rock not utilized for construction and deemed to have very low potential for acid rock drainage and metal leaching will be permanently stockpiled at the Kiggavik site. It is proposed to construct two stockpiles to accommodate the Type 2 material excavated from the Kiggavik pits.

Approximately 22 million bank cubic metres (BCM) (unbroken) of Type 2 mine rock material will be generated. The cumulative volume of the two stockpiles is estimated to be on the order of 29 million loose cubic metres (LCM), based on a swelling factor of 1.3. Excavated mine rock will be loaded on trucks and hauled to the appropriate stockpile.

Stockpiles will be located to the north and south of the Kiggavik pits (refer to Figures 4.4-1 and 4.4-2). Both stockpiles will be surrounded by perimeter ditches designed to collect runoff water from the stockpiles. Drainage water from the Kiggavik stockpiles will largely consist of direct precipitation (i.e., rainfall and snowfall), with minor amounts of blowing snow and surrounding catchment runoff, impinging the pile perimeter. Berms will be constructed along the outer edge of each ditch. These berms will prevent surface runoff from surrounding undeveloped areas from flowing into the perimeter ditch and mixing with runoff water from the stockpiles.

The stockpiles will be constructed in order to meet appropriate physical stability criteria. A layered approach to stockpile construction is proposed to increase the overall stockpile stability. The layered placement creates a high uniform density while minimizing segregation to create a stockpile with minimal permeability to air and water penetration. The method also reduces settlement and therefore further enhances overall stockpile stability. It is expected that the stockpiles will ultimately be 30 m to 40 m high and constructed in approximately 10 m lifts with catchments remaining at the completion of each lift. An angle of repose of 37° and an overall stockpile slope of 26.5° are considered at this stage. The catchments will also act as a slope break and minimize erosion caused by surface runoff.

More detailed information regarding the hydrology and thermal behaviour of permanent piles is included in Technical Appendix 5G (Thermal and Water Transport Modeling for the Waste Rock Piles and TMF), Technical Appendix 5H (Waste Rock Water Balance) and Technical Appendix 5I (Hydrology of Waste Rock Piles in Cold Climates).

6.6.1.2 *Temporary Stockpiles*

Type 3 mine rock will be segregated and temporarily stored during operation in a stockpile along the north perimeter of the Main Zone pit. Runoff and water percolating through the temporary stockpile will be collected using ditches and a holding pond, such that the water can be recycled for use in the mill and/or treated before release. During decommissioning of the site, all Type 3 mine rock from the Kiggavik pits will be hauled and placed within the Main Zone TMF.

It is estimated that the volume of Type 3 mine rock from the Kiggavik Pits will fall within the range of 700,000 to 1,400,000 LCM.

The proposed design for the Type 3 mine rock pad and sedimentation pond for the Kiggavik site includes a liner system. Design information for the mine rock pad and sedimentation pond are provided in Technical Appendix 2D. Due to the permafrost foundation conditions, the liners are required to be constructed on a rockfill pad. The pad footprint will be stripped of organic materials and ice rich soils (soils containing ice lenses or high ice contents) which may be prone to creep when loaded. Excavations in permafrost ground will be covered immediately using Type 1 mine rock fill to prevent permafrost degradation. Fill material will include non-potentially acid generating free draining rock materials. Pad grading will be designed to drain into the sedimentation ponds.

The rockfill pad will include perimeter berms. The pad liner system will run up the perimeter berms and will be anchored in a trench at the berm crest to provide control for runoff. The drainage from Type 3 waste rock will be recycled as process water or treated prior to discharge to the environment. More detailed information regarding pads is included in Technical Appendix 2D .

The option of using a base drain, constructed of crushed Type 1 mine rock, as an alternative to the liner will be further evaluated during the licensing phase.

6.6.2 Sissons Site

6.6.2.1 *Permanent Stockpiles*

At the Sissons site it is proposed to manage the Type 2 mine rock in one stockpile. This stockpile is designed in a manner similar to that of the Kiggavik stockpiles. The volume of the Sissons mine rock stockpile is estimated to be on the order 35 million loose cubic metres (LCM), based on a swelling factor of 1.3. Excavated mine rock will be loaded on trucks and hauled to the mine rock pile.

It is proposed to stockpile End Grid mine rock from mine development with the Andrew Lake mine rock. Some mine rock from End Grid will be stockpiled separately and crushed to be used in mine backfill.

6.6.2.2 ***Temporary Stockpiles***

It is proposed to manage Type 3 mine rock from the Sissons site during operation in a temporary surface stockpile with drainage collection and treatment of surface runoff. This material will then be placed at the bottom of the Andrew Lake pit during the decommissioning phase. The design concept for the Sissons temporary stockpile will be similar to that of the Kiggavik site temporary stockpiles (Section 6.6.1.2).

6.7 Mine Rock Monitoring

Contingency plans are intended to address unforeseen circumstances which could result in a substantial increase in predicted environmental impacts. Extensive investigations into the chemical properties of mine rock at Kiggavik and Sissons sites will continue to be undertaken during operation as part of a Mine Rock Optimization and Validation Program (MROVP). This program will ensure Type 2 contact water quality expectations are met. This program will also focus on Type 3 mine rock and will provide further insight into the geochemical characteristics of the materials to validate AREVA's proposed long-term management plan.

7 Milling

7.1 Introduction

The Kiggavik Project will include the construction, operation and decommissioning of a mill facility. The purpose of the mill will be to produce uranium concentrate from the ore. Most modern uranium mills are based on hydrometallurgical processing, and are comprised of a series of circuits that separate the uranium from the other materials in the ore and then produce the packaged uranium product commonly referred to as yellowcake. Uranium concentrate is an intermediate product that requires further processing before it is suitable for use as nuclear fuel. Uranium concentrate is the only product that will be produced at the Kiggavik mill.

The key circuits included in the proposed Kiggavik mill are:

- crushing and grinding where the rock is first introduced into the plant and reduced in size and mixed with water to create a slurry
- leaching where the slurry is mixed with reagents to dissolve the uranium, along with other metals and elements, from the ore into solution
- solid-liquid separation where the gangue (waste)material is separated from the metal-bearing solution
- extraction and purification where the uranium dissolved in solution is separated from any other metals and elements that were solubilised
- precipitation, drying and packaging of the uranium as a granular concentrate (yellowcake)

7.2 Mill Design Considerations and Criteria

The milling process is an important factor in the environmental and economic performance of the Kiggavik Project. The milling design approach has been focused upon a number of considerations in order to tailor the process to the site, as both the environment and location of Kiggavik present several unique challenges and opportunities.

The primary mill design considerations are as follows:

- Production of up to 4,000 tonnes U per year with uranium concentrate conforming to international refinery specifications
- Water management considerations, noting that the milling process is the key consumer of freshwater and determines the effluent treatment required.

- Tailings characteristics, noting that the milling and tailings neutralization processes will determine the geochemical and physical properties of the tailings. In turn, the tailings physical properties impact the water balance as the water expulsed from the tailings during consolidation requires treatment.
- Reagents. The type of reagents used will influence the quality of mill effluent and hence the water treatment processes required. In addition, the quantities of reagents required impact significantly upon the transportation logistics.
- Radiation protection. While the current grade estimates of Kiggavik and Sissons ores are considerably less than that of deposits typically mined in northern Saskatchewan, radiation protection measures similar to those applied at the McClean Lake Operation will be implemented at Kiggavik (see Section 15 for proposed measures).
- Dust Control. Consistent with radiation protection principles, dusty areas of the mill (i.e. grinding) are designed in an enclosed location with appropriate ventilation to minimize dust.

7.3 Mill Design Option

The preferred option selected for the Kiggavik mill process is the Resin in Pulp (RIP) process. RIP is the preferred process based on the relative simplicity of the process and lower water consumption, compared to traditional Uranium milling processes. The use of counter current decantation and solvent extraction (CCD/SX) similar to uranium mills currently operating in northern Saskatchewan would require a slightly larger footprint, and require significant additional water use and effluent discharge. The Resin in Pulp process has operated successfully in other jurisdictions and test work has shown that it can be successfully applied to milling Kiggavik project ores.

Solvent extraction may also be implemented in the future if additional deposits are developed that require further purification. The current deposits planned for processing are low in common contaminants and the resin process will be sufficient to produce uranium concentrate meeting purity specifications. However, future deposits could contain higher levels of contaminants and may require further processing by SX. Any such implementation would be the subject of a separate future assessment.

The design of the preferred mill process includes standard acid leaching, resin-in-pulp, elution and resin regeneration, gypsum precipitation, uranium precipitation, drying/calcining, packaging, and tailings neutralization. The capacity of the current mill design is approximately 3,200 tonnes of ore at 0.4% U per day, to produce up to 4,000 tonnes of U per year. It is expected that the mill will operate for an average of 310 days per year (85% availability).

7.4 Mill Facilities and Structures

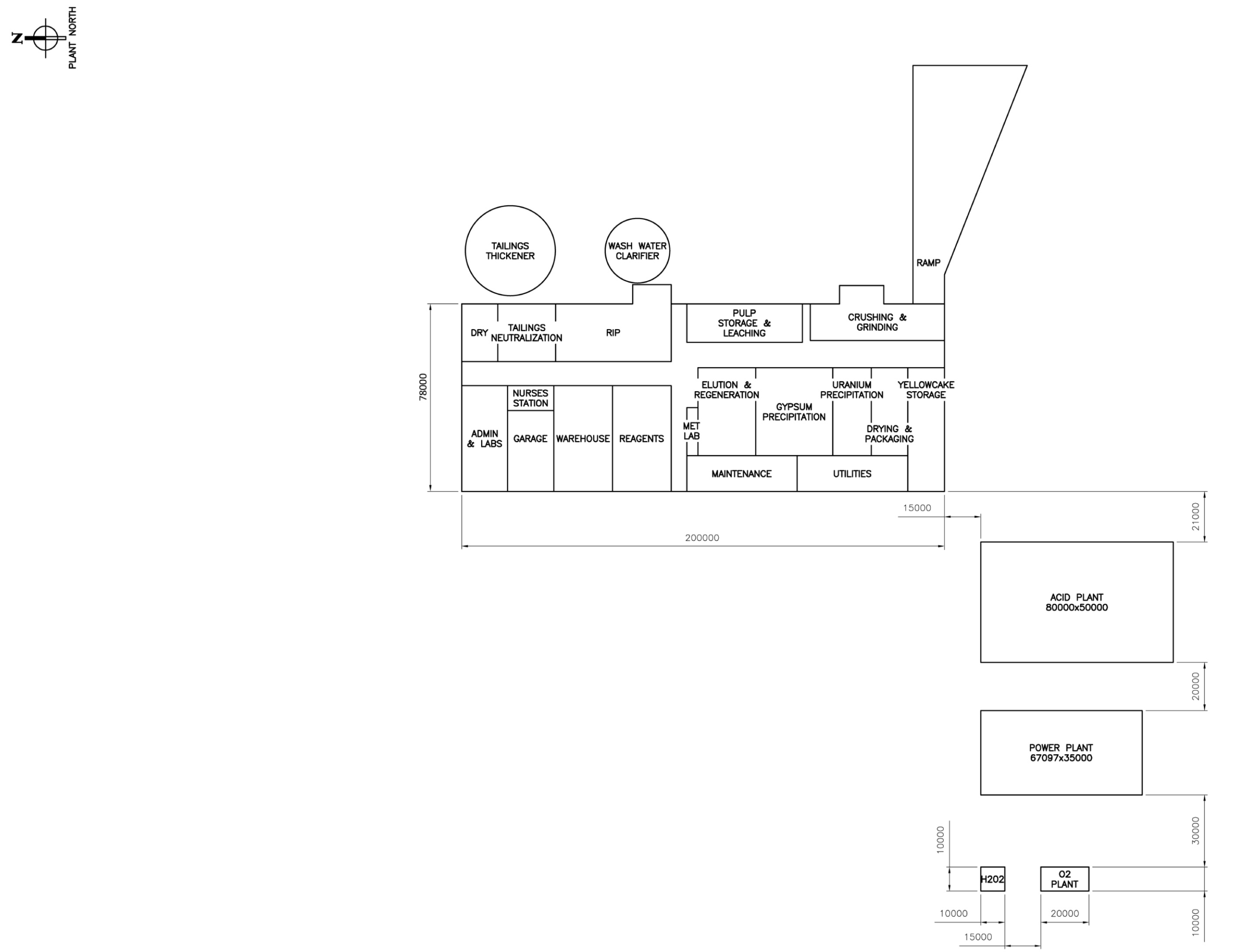
The mill complex will consist of a central mill building, housing the milling process equipment and services, and several ancillary buildings which will support the mill. These buildings will be joined by arctic corridors with cable trays and pipe racks. A plan view of the mill and ancillary facilities is shown in Figure 7.4-1.

7.4.1 Mill Building

The mill and other building foundations were based upon site-specific geotechnical information and best practices for construction in a permafrost environment. The proposed concept, which is described in more detail in Section 12.6, is to remove overburden and fractured rock and replace it with engineered fill (compacted crushed rock).

The mill foundation concept will utilize a structural slab on the engineered fill with piles as needed to support heavy equipment foundations. Insulation around the perimeter of the buildings, the buildings themselves and a sub-surface pumping system will ensure that the permafrost is protected and the building foundations stay on a warm dry bed with little risk of frost heaving.

The mill building will be steel construction with insulated steel clad walls. Roof heights range from 22 m to 40 m. The interior of the building is divided with interior walls and ceilings to create separate areas to control ventilation. The interior walls are generally steel clad. The area containing crushing and grinding has concrete walls for noise and dust control purposes.



NOTE:
LOCATION OF ANCILLIARY BUILDINGS IS APPROXIMATE
ALL DIMENSIONS ARE IN MILLIMETRES UNLESS OTHERWISE NOTED

FOR APPROVAL
09 FEB 2011

FIGURE 7.4-1

Association of Professional Engineers & Geoscientists of Saskatchewan
CERTIFICATE OF AUTHORIZATION
MARCH CONSULTING ASSOCIATES INC.
Number C1014
Permission to Consult held by:
Discipline SK. Reg. No. Signature

March Consulting Associates Inc.
CONSULTANT

March DWG No. KI110C002_A(1) | PROJECT No. M09163 | DIVISION No. 16

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REFERENCE DWG No.	DESCRIPTION

No.	DATE	DESCRIPTION	BY	CHK	DSN	AP1	AP2

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B	02/09/11	FOR APPROVAL	JJ	CF	AC		
A		NOT ISSUED					

		DATE MM/DD/YY
SCALE:		
DESIGNED:	A.CARINO	01/25/11
DRAWN:	J.JANIS	02/08/11
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APPROVED:		
APPROVED:		

AREVA
KIGGAVIK PROJECT
PROJECT No. 255KI

KIGGAVIK INITIAL FEASIBILITY MILL DESIGN

SITE PLAN OVERALL
GENERAL ARRANGMENT

DRAWING No. KI110-C-002 REV B

7.4.1.1 *Mill Layout*

The mill layout has been designed based on the following considerations:

- maintaining segregation between slurry flow and solution flow (i.e. crushing through to tailings neutralization versus elution through to uranium precipitation);
- maintaining service access to circuits as necessary
- minimizing opportunities for short-circuiting between process exhaust and fresh-air intakes;
- maintaining areas with higher potential for dust and radon emissions (ore pad and crushing) down-wind of other activities (predominant wind direction is from the NNW);
- minimizing pipe runs and elevation head within the mill and from the mill to the WTP and TMFs;
- facilitating single pass ventilation design and radiation protection requirements; and,
- maintaining areas with potential for spills up-gradient of containment ponds and TMFs.

The mill building layout has also been designed to facilitate the flow of process materials. The slurry will flow along the east side of the building from south (crushing) to north (tailings neutralization). The resin and solution phases will travel back down the middle of the building so that uranium packaging and shipping will be at the south end of the building and accessible by truck.

General layouts of mill equipment have been prepared with consideration of maintenance requirements and safe access. A mill maintenance facility will be included in the mill building. Whenever possible, equipment repairs and re-builds will be completed on-site to maximize self-sufficiency and minimize transport requirements.

The design of each circuit incorporates primary, secondary, and tertiary containment. All process solutions and slurries are contained within covered vessels, which are further contained within bermed areas graded towards sump and pump systems. Exterior doorways will also be bermed to provide additional containment.

The entire mill building will be crossed by two corridors. The purpose of these corridors is to facilitate safe transportation of materials, personnel and equipment within the mill and to provide for the ventilation system. Services such as utilities and maintenance shops will be located on the outside wall of the west side of the building and the administration offices, nurses' station, garage and warehouse will be in the northwest quadrant.

7.4.1.2 *Fire Suppression*

The Fire Suppression System will be centered in the mill building. The bottom portion of the raw water tank will be dedicated to fire suppression. A heat-traced fire loop will run through the facility from the dedicated fire water tank through a Jockey pump to maintain pressure in the loop. The sprinkler system, hydrants, stand pipes, deluge and pre-action systems will be connected to the fire water loop. Pressure sensors will be connected to the control system to provide notification and control of the main fire pump. A diesel fire pump will be available for backup in case of power failure.

7.4.1.3 *Laboratories and Metallurgy*

There will be 3 laboratories located within the Kiggavik mill: the Radiation Protection Laboratory, the Chemical Laboratory, and the Metallurgical Laboratory.

The Radiation Protection Laboratory will be used to analyze radiation samples collected in the mill, the mines and ancillary facilities.

The Chemical Laboratory will be used to analyze mill samples collected for metallurgical accounting and control. Analysis of environmental samples and urinalysis may also be conducted in the Chemical Laboratory; however sample preparation will be segregated to prevent cross-contamination between mill and environmental samples. The Kiggavik Chemical Laboratory will be designated as a Basic Level Radioisotope Laboratory. It will comply with CNSC Regulatory Document R-52, *Design Guide for Basic and Intermediate Level Radioisotope Laboratories*.

The Metallurgical Laboratory will be used to prepare slurry samples for analysis in the Chemical Laboratory and will also be used for on-going metallurgical test programs and optimization.

Laboratories will be primarily staffed by technicians and technician trainees, and supported by metallurgists, chemists, and radiation protection coordinators.

7.4.2 Ancillary Buildings

There will be 4 key ancillary buildings supporting mill operations; these are:

- acid Plant;
- oxygen Plant;
- hydrogen peroxide storage; and,
- powerhouse.

7.5 Process Description

Extraction of uranium from ore is typically achieved via hydrometallurgical processes whereby the uranium is solubilized and purified in solution form prior to precipitation as solid uranium oxide. The preferred Kiggavik mill process includes acid leaching, resin-in-pulp extraction and acid elution, gypsum precipitation and uranium precipitation. All remaining rock materials and solid byproducts will be treated in a Tailings Neutralization circuit. Solution streams will be recycled where possible, with bleed streams treated in Tailings Neutralization.

The overall Process Flow Diagram for the proposed mill is shown in Figure 7.5-1. The following sections provide summary descriptions of each stage of the process.

7.5.1 Ore Handling, Crushing and Grinding

Ore from the mines will be transported to the ore stockpiles (Section 5). The design average ore feed grade is 0.4% U, however the actual grade of ore contained within a haul truck may vary from 0.2% to 0.8% U. For ore grade blending purposes separate stockpiles will be maintained on the ore pad, with the following approximate grade ranges: < 0.25% U, 0.25 – 0.5% U, >0.5% U. The distribution of stockpile grades will be adjusted over the life of the Kiggavik Project to maintain mill feed consistency.

Concerns have been expressed by community members about the dust generation from the milling process, specifically from the crushing circuit (EN-KIV OH Oct 2009[88], EN-RI OH Nov 2010[89]). The Crushing and Grinding circuits will be located at the southwest corner of the mill facility and will be enclosed to prevent the spread of dust.. The crushing and grinding circuits have been sized appropriately such that are can be fed directly from crushing to grinding and an intermediate stockpile is not required. Front-end loaders will be used to deliver approximately 3,200 tonnes of ore per day to Feed Hoppers. Ore will be drawn from the feed hoppers to the crushers, which will reduce the ore size to approximately 300 mm.

The wet-grinding circuit will reduce the ore from 300 mm to less than 300 µm. The circuit will consist of a semi-autogenous grinding (SAG) mill, ball mill and hydrocyclones for classification. The storage

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[88] EN-KIV OH Oct 2009: *When material goes through the crusher it gives off radiation, how will you deal with this? I worked in a gold mine near the crusher and lots of dust came out.*

[89] EN-RI OH Nov 2010: *Concern with dust from crushing and possibility of dispersion*.

pachucas will provide surge capacity between Crushing and Grinding and Leaching in case of weather delays or maintenance downtime.

A dedicated scrubber will be used to control dust emissions from the Crushing and Grinding circuits.

7.5.2 Leaching

Slurry from the storage pachucas will be pumped to the Leaching circuit. The purpose of the leach circuit will be to dissolve uranium from the ore into the solution phase. Sulphuric acid, oxygen and steam will be added to control pH, oxidation-reduction potential and temperature within the ranges suitable for efficient uranium leaching. If required, hydrogen peroxide and/or ferric sulphate may also be used.

Slurry will flow by gravity through the leach pachucas until it discharges from the final vessel to the leach discharge surge tank. The slurry will then be pumped to the Resin-In-Pulp (RIP) circuit for the recovery of soluble uranium.

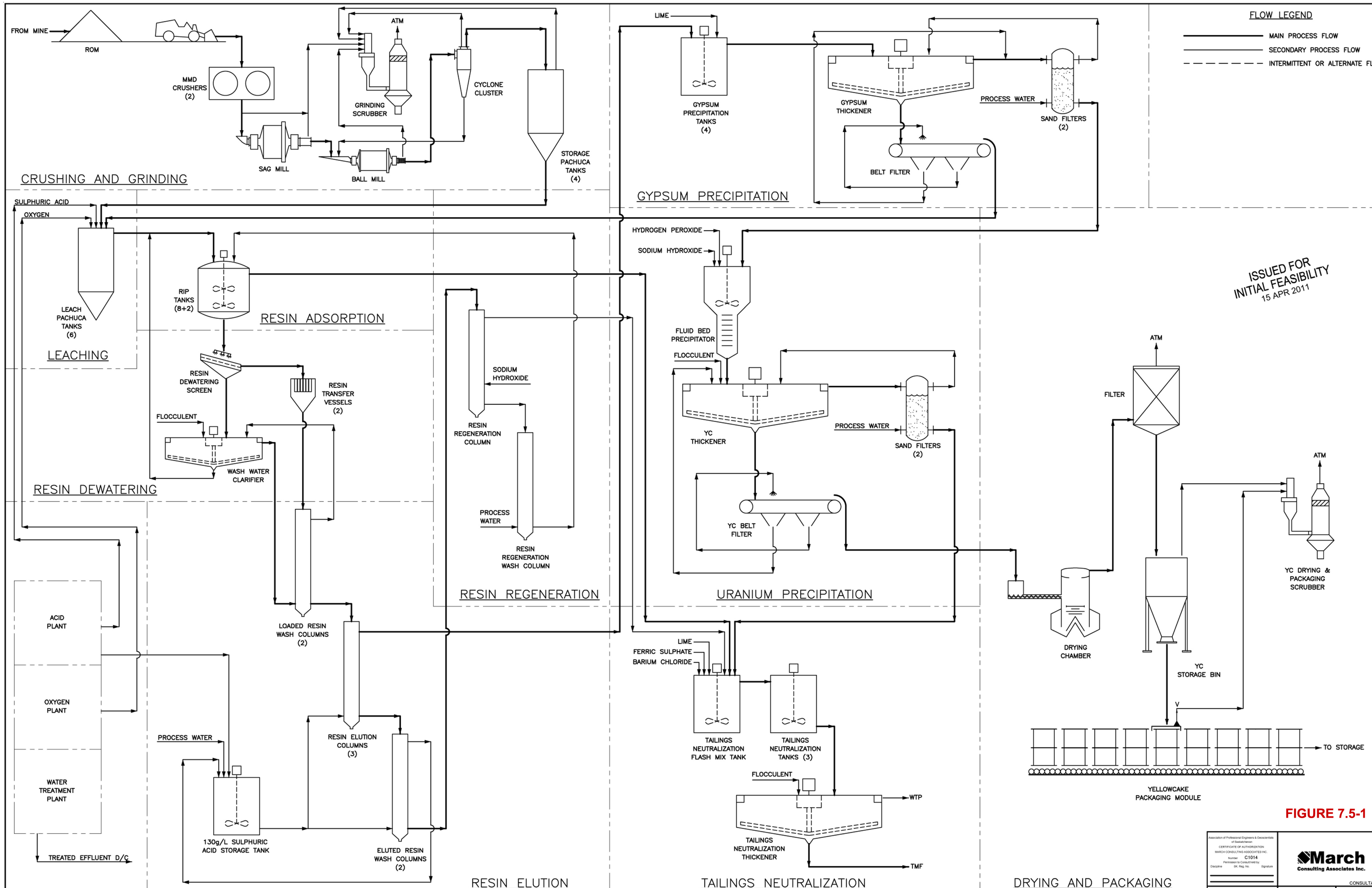

FIGURE 7.5-1

March Consulting Associates Inc.

CONSULTANT

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March DWG No. KI300A001_F(1) | PROJECT No. M09163 | DIVISION No. 16

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F	04/15/11	ISSUED FOR INITIAL FEASIBILITY	CF	AC	AC	TN	

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E	03/15/11	FOR CLIENT APPROVAL	CF	AC	AC		
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C	10/03/10	FOR REVIEW AND COMMENTS	CF	AC	AC		
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A	07/08/10	FOR REVIEW AND COMMENTS	JJ	AC	AC		

		DATE MM/DD/YY
SCALE:	NTS	
DESIGNED:	A.CARINO	06/23/10
DRAWN:	J.JANIS	08/04/10
CHECKED:		
APPROVED:		
APPROVED:		

AREVA

KIGGAVIK PROJECT

PROJECT No. 255KI

KIGGAVIK INITIAL FEASIBILITY MILL DESIGN
MILL FLOWSHEET SUMMARY

PROCESS FLOW DIAGRAM

DRAWING No. KI300–A–001 REV F

7.5.3 Resin-In-Pulp

The resin-in-pulp (RIP) circuit will use small resin beads to recover uranium from the leached slurry. Uranium preferentially adsorbs onto the beads, which can then be easily separated from the barren slurry using industrial screens.

The resin and slurry will be contacted in large vessels that allow slurry movement from vessel to vessel while retaining resin within each vessel. As the slurry moves through the vessels, the uranium in solution progressively adsorbs on the resin, until more than 99% of the uranium in solution has been adsorbed. The barren slurry will then pass through a safety screen to recover any entrained resin before being pumped to the Tailings Neutralization circuit.

Once the resin inventory in a tank is loaded with uranium, it will be taken off-line and the resin/slurry mixture pumped to the dewatering screen. The screen separates the resin from the slurry. Wash water used for resin washing will be directed to a clarifier and recycled within the mill.

The decrease in resin volume over time in a resin in pulp (RIP) operation is the result of two on-going processes: resin breakage and resin attrition. Resin breakage is caused by the fracturing and cleavage of the resin bead into sizable fragments. This process is very undesirable as it can lead to potential uranium loss as the fragments can capture uranium but may pass through the safety screens and report directly to tailings. Accordingly, RIP plant equipment is specifically designed to minimize/eliminate this type of resin bead failure. The process of attrition is the slow gradual reduction in diameter of the resin bead primarily through collision with ore particles. In this case the quaternary ammonium functional group on the attritted debris has limited functionality as its ability to complex uranium has been reduced. The most important operational source for potential uranium loss are therefore the resin beads that gradually decrease in diameter through attrition and pass through the RIP process screens directly to tailings.

Although not well known in the western world, RIP plants for gold and uranium processing were common in the former Soviet Union and are still operating today. Resin manufactures, such as Purolite and Rohm and Haas, have produced resin for uranium RIP processes for decades to supply these operations. Expressed in terms of the total resin inventory, these manufactures typically report resin loss at 20% to 40% per year. The design criteria for resin loss for the proposed Kiggavik mill are 15% to 20% of total resin inventory per year. This range of values was provided by the technical service departments of the resin manufacturers who in turn obtain these values from RIP plant operators who use their resin. Resin loss values in this lower range requires incorporating RIP equipment designs that contain specific slurry handling and resin mixing features to optimize uranium absorption and minimize resin loss. Such designs are commercially available and would be embodied in the Kiggavik mill process.

AREVA's SEPA metallurgical testing facility in Bessines, France has a complete RIP pilot plant and this unit was used to support the design of the Kiggavik mill process. For example, the pilot plant performance indicated that fouling of the resin by gypsum would not be a significant operational problem. However, it is not practical to use the pilot plant unit to estimate resin loss because too many cycles of resin loading and stripping are required making the effort temporally and financially excessive, and if completed there is still uncertainty as to whether the measured resin loss will reflect the performance of the real plant. It is more effective to obtain resin loss values from full size working plants and adapt them to our mill design situation, which is what has been done.

Tests were conducted at the SEPA facility using a simple batch resin breakage and attrition test (not using the pilot plant). The purpose of the test was to create data in order to judge how reasonable the resin manufacturers' estimates of resin loss were. Approximately 2 L of resin (fines removed by a 200 μm screen) were added to 10 L of real leach pulp (L/S ≈1.0) while subjected to mechanical agitation (≈250 rpm). The size distribution of the resin was examined at regular intervals (every 15 days). The results are for Resin Attrition for Kiggavik ore are presented in the following table.

Table 7.5-1 Resin Attrition Test – Size Distribution

Days	Size Distribution, mL						Volume Total (mL)
	200-315 μm	315-400 μm	400-630 μm	630-800 μm	800-1000 μm	>1000 μm	
0	0	0	4	295	1300	346	1945
15	1	1	6.5	474	1320	136	1938.5
30	3	0.5	12	530	1240	92	1877.5
40	2	4	16.5	480	1260	90	1852.5
60	1.3	3	16	460	1230	130	1840.3
75	1	4	14	460	1170	100	1749.0
90	1	7	24	685	1010	31	1758.0

This simple test was conducted for three months with a total resin volume loss of slightly less than 10%. Assuming a linear attrition rate this would amount to about 38% resin volume loss per year consistent with the upper end of the range provided by the resin manufacturers. As the test proceeded, the size distribution gradually became finer as observed in the table. With reference to resin lost to undersize in the RIP tank screens, the screen size is 630 μm. In our test, resin lost due to undersize was 0.2% at time zero and increased to 1.8% after three months of continuous agitation. This simple intuitive test contained no measures to optimize resin attrition and resin loss numbers in the upper range provided by the resin manufacturers were obtained. As such, the resin loss values of 15% to 20% for the proposed Kiggavik mill provided by the resin manufacturers are reasonable when design features to minimize resin attrition are incorporated into the RIP facility design.

In an actual full size RIP plant, the quantity of resin loss is important from a financial perspective because the resin lost has to be replaced. However, it does not necessarily equate to uranium loss. The mean diameter of new resin is typically around 900 µm. Actual RIP plants commonly use a process screen of 630 µm openings followed by a safety screen of 500 µm. As the new resin size distribution progressively becomes finer it eventually will undersize the 630 µm process screen. However it will generally be captured by the safety screen at 500 µm. The resin oversizing the safety screen is periodically collected and processed manually on a batch basis to recover the uranium and to isolate and discard the undersize, barren resin. Resin loss therefore continually occurs without significant loss of uranium. Operation of RIP plants require a strong focus on screen performance to ensure they are separating resin cleanly from the pulp. When operated efficiently, the magnitude of soluble uranium losses from a RIP plant are generally consistent with the combined losses from the counter current decantation (CCD) and solvent extraction (SX) processes from a conventional uranium plant. AREVA will monitor the resin characteristics and optimize the RIP process to ensure that resin attrition does not adversely impact the tailings properties.

7.5.4 Elution

The elution circuit will recover uranium from the loaded resin. This circuit will include two sub-unit processes: Resin elution, which will recover the uranium into solution and resin regeneration, which will replenish the resin for re-use in RIP.

Uranium will be recovered from the resin in elution columns which contact the loaded resin with sulphuric acid or sodium carbonate eluant. The resin will be counter-currently washed with process water as needed before and after elution. The wash solutions will be recycled where possible for further use within the mill.

A portion of the eluted resin from the elution circuit will be regenerated as required to mitigate the effects of impurity build up by regeneration with sodium hydroxide (NaOH) and sodium sulphate (Na_2SO_4) solution. Following regeneration, the waste regenerant solution will be pumped to Tailings Neutralization. The pH and impurity levels of the solution are expected to render it non-recyclable.

Regenerated resin will then be rinsed with process water to remove any entrained regenerant prior to return to the RIP circuit for re-loading. The spent wash solution will be recycled for use in regenerant make-up.

7.5.5 Gypsum Precipitation

Eluate from resin elution will contain sulphate and possibly iron in excess of that suitable for uranium precipitation. The gypsum precipitation circuit will remove these impurities to appropriate levels.

The circuit will consist of a number of mix tanks where lime will be added to achieve pH 3.5 to 3.8, thereby precipitating gypsum and iron oxide. The purified eluate will then be separated from the gypsum cake in a thickening and filtration circuit. The solution will be transferred to uranium precipitation. The filter cake will be washed counter currently with process water and recycled filtrate.

The filter cake from gypsum precipitation will be re-slurried and pumped back to RIP circuit to recover any uranium entrained or co-precipitated with the gypsum.

7.5.6 Uranium Precipitation

Uranium peroxide will be precipitated from purified eluate by contact with hydrogen peroxide (H_2O_2) at pH 3.4. Sodium hydroxide (NaOH) will be used to maintain pH.

Precipitated solids will be dewatered using thickening and filtration prior to entering the drying circuit. Barren eluate will be recycled where possible within the mill with the remainder bleeding to tailings neutralization.

7.5.7 Yellowcake Drying and Calcining

Yellowcake from uranium precipitation will be dried at 120°C to less than 2% moisture, producing $UO_4.2H_2O$. The product may then be calcined at approximately 800°C to produce U_3O_8 and discharged to the yellowcake storage bin.

A dedicated scrubber and HEPA filter will be used to control dust emissions from the drying and calcining area.

7.5.8 Yellowcake Packaging

Product packaging will be accomplished in a semi-automated packaging system that will fill steel drums with up to 434 kg of yellowcake each. The essential components of the system will be:

- sectional roller conveyor, with some selected sections operated automatically;
- a series of air locking sections to control workplace dust;
- a drum filling station;

- a drum lidding station;
- a drum weighing station; and,
- a drum washing and drying station.

Following packaging, the drums will be placed in a secure storage area for loading into shipping containers. Drums are scanned by the Radiation Protection Department (Section 15) for surface contamination prior to loading shipping containers.

7.5.9 Tailings Neutralization

The tailings neutralization process has been designed to produce tailings suitable for management within the in-pit tailings management facilities. The process is described in Section 8.2.

7.5.10 Process Control

The process control philosophy includes advanced process control capabilities to reduce the likelihood of incidents resulting from operator error, but allows rapid intervention from operators if needed. There will be a central mill control room with an operator who will oversee the operation of each unit process. Each unit process will in turn have a process control station where the area operator can independently monitor and operate that particular circuit. The area operators will have the responsibility of ensuring the circuit and associated equipment are functioning safely and efficiently. When required, the central control room operator may directly control the unit process(s) with field support from the area operator.

Automatic samplers will be installed at appropriate locations to monitor the performance of each unit process and overall metallurgical efficiency of the facility. The samplers are intended to supplement routine process composite and grab samples collected by operators.

7.6 Process Reagents

A number of reagents will be required to operate the mill process; these reagents and the proposed methods of preparation are described in the following sections. Handling and storage of hazardous materials are also discussed in Section 14.

The mill will also provide some reagents to the Water Treatment Plants (WTP) at the Kiggavik and Sissons sites (Section 9). Tote-filling or truck/tanker-filling stations will be installed as required to transfer these reagents.

Conservative estimates of the annual consumption of process reagents are shown in Table 7.6-1. It is anticipated that lime and sulphur requirements will be reduced as a result of on-going tests and studies. Annual consumption will be updated at the time of licensing application.

Table 7.6-1 Process Reagents

Reagent	Use	Maximum Annual Consumption (tonnes)
Lime	pH modifier	33,914
Sulphur	Sulphuric acid generation	23,451
Hydrogen peroxide	Uranium precipitation	1,716
Sodium hydroxide	pH modifier	1,442
Ferric sulphate	Transition metal precipitation	239
Barium chloride	Radium precipitation	303
Sodium sulphate	Resin regeneration	632
Resin	Uranium extraction	1,125
Flocculants	Solid-liquid separation	104

Sodium hydroxide, ferric sulphate, barium chloride, sodium sulphate, and flocculants will be received in drums or bags and stored in a covered building or in sea containers. These reagents when required will then be diluted with water and used in the process. Preparations of the remaining reagents, which require more complex storage and/or further processing, are described as follows.

7.6.1 Sulphuric Acid

Sulphuric acid (93% H_2SO_4) will be produced in an on-site acid plant located near the mill. Acid will be supplied via pipelines to Leaching, Elution, and the Kiggavik WTP. A filling station will be required to provide acid to the Sissons WTP.

The production of sulphuric acid involves the process of burning sulphur in the presence of dried ambient air, reaction of the products in a catalyst bed, and recovering the reacted components in an air absorption solution to produce sulphuric acid. Waste heat in the form of superheated steam is also produced as part of the reaction. As much of this excess heat will be recovered and used in the mill process.

Sulphur will be received as solid prills or agglomerated high purity solid sulphur and stored in cold storage.

The plant will be designed to emit less than 75 g SO_2 per tonne of acid produced. Emissions will be controlled by two systems:

- excess SO_2 will be absorbed in the last pass by a cesium-promoted catalyst with a lower working temperature in one or several layers; and
- a scrubber installed on the exhaust stack will remove particulates, acid mist and excess SO_2.

7.6.2 Oxygen

Oxygen (O_2) for leaching will be produced in an on-site Vacuum Pressure Swing Adsorption (VPSA) oxygen plant.

The oxygen plant will consist of two adsorber vessels filled with a zeolite molecular sieve, a valve assembly, air compressor, air filters, main pressure regulator, and air and product receiver tanks. Dry, compressed air is passed through the air filters, which remove particles and oil vapour, and then through the air inlet regulator, which reduces the air to the final operating pressure. Clean dry air will be directed to one of the adsorber beds where nitrogen and water vapour adsorb faster than oxygen in the pore structure of the molecular sieve, thus increasing the oxygen purity of the product gas stream to 93.0% or higher. Oxygen product will exit at the top of the adsorber bed into the product receiver for delivery to the mill as needed.

Nitrogen enriched waste gas will be discharged to the atmosphere through a silencer.

7.6.3 Lime

Milk of lime will be used for pH adjustment in Tailings Neutralization, Gypsum Precipitation, the Kiggavik WTP and the Sissons WTP. The lime preparation circuit will be located in the mill.

Quicklime (CaO) will be received as a solid and thawed indoors if required. The quicklime will be blended with steam-heated water in a lime slaking ball mill in a closed circuit with a set of cyclones.

The slaked lime will then be stored in lime slurry storage tanks for use in the process. A filling station will be installed to provide lime to the Sissons WTP.

7.6.4 Hydrogen Peroxide

Hydrogen peroxide will be used in the yellowcake precipitation circuit to precipitate uranium peroxide. The reagent will be delivered to site in ISO containers at a strength of 50 – 70% hydrogen peroxide.

Hydrogen peroxide is a strong oxidant and therefore is a risk for fire and explosion if concentrated solution is mixed with organics such as oil or grease. To minimize the risk of accidents, peroxide will be stored in a dedicated building and strictly managed.

Within the hydrogen peroxide storage building, the solution will be stored in a covered and vented container constructed from compatible materials. The storage containers will be located within a containment berm that can accommodate 110% of the largest container volume. The containment area will have a deluge system for diluting any spillage and leak detection tied into the distributed control system (DCS) and local building alarms. Safety equipment in the building will include detectors for signs of oxidation. Cleanliness will minimize possible sources of combustion. Entry to the building will be restricted to those trained in the safe handling of concentrated peroxide.

7.7 Process Water Management

The mill process design has been adapted to reduce water consumption and maximize internal recycle. The overall Kiggavik and Sissons site water balances are discussed in detail in Section 9.

The 3 systems providing water to the mill, listed in order of decreasing quality, will be:

- raw water from Siamese Lake;
- permeate from the Kiggavik water treatment plant; and
- process water, which includes recycled site run-off from the outdoor purpose-built pit and water recycled internally in the mill.

Permeate is expected to be of high quality, such that it will be suitable for use in most applications in the mill. Potable and fire water will be sourced solely from raw water. Site-runoff water is expected to be low in sulphates but potentially containing dissolved metals and radium. Process water in general will have moderate levels of sulphates (≤1 g SO_4/L), dissolved metals and radium. Therefore it is assumed suitable for general process use but not suitable for cake washing or reagent preparation. In order to maximize recycling and use of on-site water, all permeate and site run-off/process water will first be used in the mill with any additional mill water make-up required, due to lack of availability of permeate or process water, supplied from raw water.

Maximum expected mill water requirements and the minimum water quality type required for each use are shown in Table 7.7-1. These inputs, along with known and potential recycle streams, are shown in Figure 7.7-1.

Table 7.7-1 Mill Water Requirements – Base Case

Process Area	Max Flowrate m³/d	Minimum Water Quality Requirements
Crushing	10	Process
Grinding		
Grinding Scrubber	70	Permeate
Mill Make-up	2,500	Process
Leaching	10	Process
Resin Adsorption		
Safety Screen Wash	91	Process
Resin Dewatering		
No.1 Screen Wash	48	Process
No.2 Screen Wash	48	Process
Resin Elution		
130 g/L Acid Dilution	625	Permeate
Resin Regeneration		
Regenerant Make-up	194	Permeate
Gypsum Precipitation		
Belt Filter Wash	159	Permeate
Uranium Precipitation		
Recycle Cooler Cooling Water	10	Permeate
Belt Filter Wash	96	Permeate
Drying and Packaging		
YC Drum Washing	20	Permeate
Drying and packaging scrubber	300	Permeate
Tailings Neutralization	10	Permeate
Barium Chloride Make-up	1	Permeate
Lime Preparation		
Lime Mill Make-up Water	480	Permeate
Lime Mill Pump Box Dilution	757	Process
Miscellaneous Reagent Make-up Water	57	Permeate
Acid Plant	740	Raw Water
Steam Boilers	267	Raw Water
Miscellaneous Usage Included as Allowance Only		
Pump Gland Water	672	Permeate
Washdown Hosing	50	Permeate
Sand Filter Backwash Waters	50	Permeate
Ventilation Cooling Water	100	Permeate
Total Requirements	**7,374**	

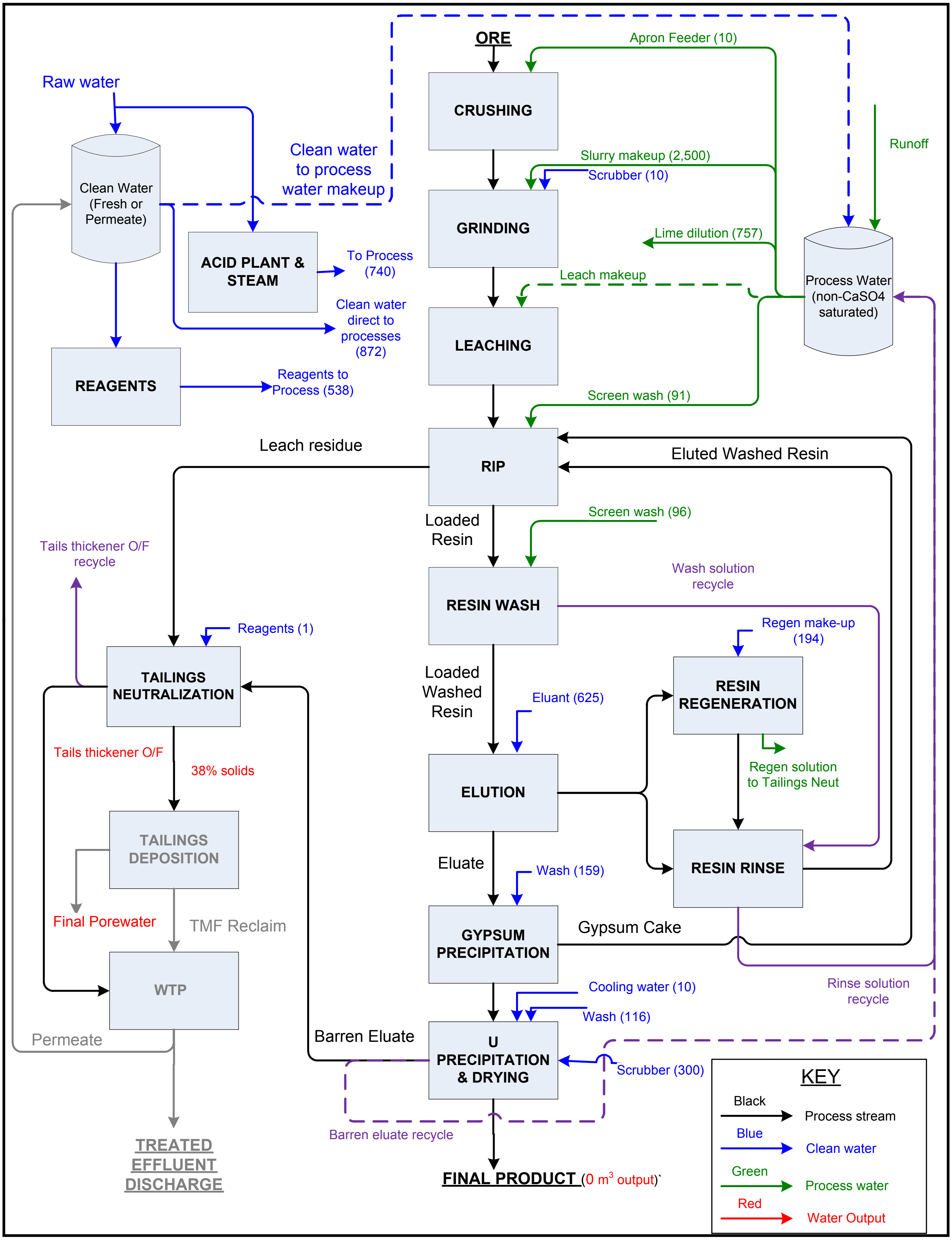

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FIGURE 7.7-1
WATER BALANCE FLOWSHEET

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The majority of the water entering the process, not accounting for miscellaneous allowances, will be used to slurry mill feed for grinding and prepare reagents and steam. This water will be essentially non-recyclable on a unit operation basis as it enters the general process flow.

Screen spray wash and resin column rinse water will be recycled to process water. Key streams that can provide additional substantial sources of recycling include barren eluate and tailings thickener overflow. The proportions of these streams that can be recycled will be dependent on actual process gypsum and impurity concentrations and will vary over the life of the Project. For the purposes of Figure 7.7-1, approximately 30% of barren eluate and tailings thickener overflow are shown as potentially recyclable, however, these recycle streams have not been included in the conservative calculation of water requirements.

Ranges for the expected use of each water type are shown in Table 7.7-2. It is expected that the volumes actually available for each type of water will vary on a seasonal basis as well as over the life of the Project. If insufficient permeate is available, this water will be supplemented by raw water. As it is expected that average permeate production from the WTP will be 1,503 m^3/d, it is likely that average supplementation of permeate will be 1,002 m^3/d during periods of maximum mill production. In turn, if insufficient process water is available due to a lack of stored site drainage, process water flows can be supplemented by either permeate (if available) or raw water. In the extreme case where no permeate or process water is available, a conservative maximum of 7,374 m^3/d of raw water would be required for the mill.

Table 7.7-2 Expected Average and Ranges for Mill Water Use (m^3/d)

Mill Water Type	Average	Range
Raw Water	2,633	1,007 – 7,374
Permeate	1,503	0 – 2,505
Process Water	3,570	0 – 3,862

Water will exit the mill process via the tailings circuit. A thickener has been included in the design to increase the recovery of water prior to tailings discharge to the TMF. Both the tailings thickener overflow and expulsed porewater from the TMF will report to the Kiggavik WTP (Section 9).

7.8 Mill Exhaust Stacks

Mill stacks will have scrubbers to remove particulate and contaminants from the air stream before discharge; regular sampling will ensure that the scrubbers are operating effectively. It is anticipated that stack sampling will be performed on a regular basis to monitor air emissions. The baghouse fines will be filtered through a HEPA filter prior to releasing the exhaust gas to the atmosphere. The

HEPA filter will be cleaned or replaced periodically using safe work procedures. Any fines collected will be either returned to the process or processed through the tailings neutralization circuit for ultimate disposal in the tailings management facility.

8 Tailings Management

8.1 Concept

AREVA's experience in tailings Management in northern Saskatchewan has been used as a benchmark for design of the Kiggavik tailings management facilities.

Kivalliq residents have expressed concerns relating to the safety and management of the tailings, both during operation and upon closure of the Project (EN-CI NIRB May 2010[90]. EN-RI KIA Apr 2007[91]). Residents want assurances that the tailings management selected will be effective under both permafrost and non-permafrost conditions (EN-CI NIRB May 2010[92], EN-KIV OH Oct 2009[93]). People have noted that at other tailings facilities in the north, dykes have failed, and they do not want to see this happen at Kiggavik (EN-KIV MAY Feb 2009[94]). Other concerns related to tailings have focused on waste water(EN-BL OH Nov 2013[95]) and radiation protection(EN-RI RLC Feb 2009[96]).

The proposed tailings management plan for the Kiggavik Project has been designed to build on the experience developed in northern Saskatchewan and adapt that experience to environmental conditions at the Project site. The plan was developed according to the following principles:

- to avoid interaction between tailings and natural water bodies;
- to minimize dust and radon release
- to maximize the use of mine workings for long-term management of tailings; and
- to ensure the long-term protection of Kiggavik's terrestrial, aquatic and human environment.

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[90] EN-CI NIRB May 2010: *What will happen to the tailings during the mining process and after the mining process?*

[91] EN-RI KIA Apr 2007: *So the tailings, are they more of a health hazard? So is it concentrated?*

[92] EN-CI NIRB May 2010: *How do we know the models/technology (southern models) that will be used will work in the arctic and in the permafrost?*

[93] EN-KIV OH Oct 2009: *I see that you will have your tailings underwater but when the ground is frozen, you can control it. But when it starts to warm up, how will you manage it?*

[94] EN-KIV MAY Feb 2009: *Saw tailings dyke fail at Culloten Lake. How will this be prevented?*

[95] EN-BL OH Nov 2013: *Chemicals in tailings can contaminate. Where does the waste water go?*

[96] EN-RI RLC Feb 2009: *Does water protect from radiation?*

Historically, non-uranium northern mines in the continuous permafrost zone have benefited from the presence of permafrost conditions around the area of tailings management. The continuous permafrost in the ground provides a natural hydraulic barrier, which prevents the migration of potential contaminants from the tailings facility into the environment. Above ground tailings impoundments were engineered to allow the permafrost aggrade into the tailings, thereby creating a stabilized mass, encapsulated by frozen conditions. It is now considered possible that climate change may threaten the integrity of such tailings management structures that rely on maintaining present temperatures and permafrost conditions for stability and integrity. Over the long-term, the tailings management have been planned to ensure a robust design for protecting the environment in either the presence or the absence of permafrost. Sucessful long-term stability of the tailings and mine rock facilities do not require permafrost encapsulation.

The proposed tailings management approach for the Kiggavik Project is modelled after AREVA's McClean Lake Operation in northern Saskatchewan and is based on the "in-pit tailings management facility" concept. This approach has been accepted by the regulatory agencies for the currently operating TMFs at three uranium mills in non-permafrost conditions in northern Saskatchewan.

In-pit disposal of tailings in the mined pits does not require the use of dykes and is considered to be the best tailings management option at the Kiggavik Project for minimizing potential operating and post-closure impacts to the receiving environment. In-pit disposal will also reduce the footprint of the tailings management facilities (TMFs), and will provide the most secure, long-term containment for the tailings at the Kiggavik site.

The preferred option for tailings management at the Kiggavik site is summarized in Figure 8.1-1. The preferred option consists of producing thickened tailings that will be neutralized and treated to control uranium, radium-226 and trace metal concentrations. Tailings will be deposited subaqueously. Subaqueous placement of tailings will prevent freezing of the tailings, prevent dust generation and enhance radiation protection by reducing radon during the operational and consolidation period. During the first few months of TMF operation, water from the purpose-built water storage facility will be added to the TMF to keep the tailings covered. After the first few months of operation, water cover of the tailings can be maintained by managing the pumping of TMF reclaim water for treatment in conjunction with water entering the TMF through runoff and precipitation and consolidation of the tailings solids. The maximum amount of water expected to be added to the TMF at any time is 1,000m3/d.

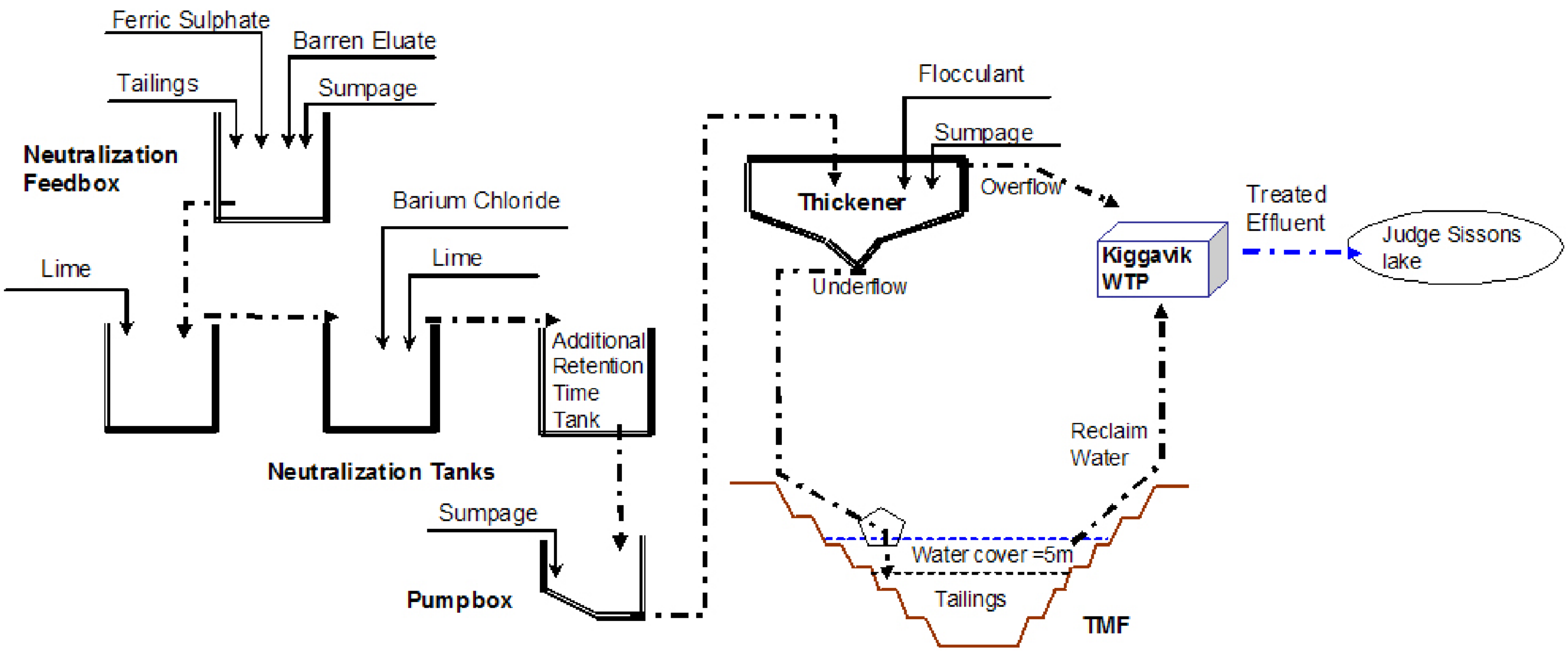

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FIGURE 8.1-1
KIGGAVIK TAILINGS MANAGEMENT SYSTEM SCHEMATIC DIAGRAM

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The natural surround concept will be used. At closure, the tailings will be covered using mine rock to enhance consolidation and prevent wind and water erosion. A final cover of overburden will be added to facilitate revegetation.

Tailings will be deposited sequentially in three open pits at the Kiggavik site. Based on the current mine schedule, tailings will be deposited first into the East Zone Pit, followed by the Centre Zone and Main Zone pits. Mill ore processing will not commence until a tailings facility is available. Any sludge generated from the WTP prior to the tailings facility being constructed will be stored in a temporary holding facility, and re-processed through the tailings neutralization circuit when the TMF is in operation. The East Zone Pit is sized to store tailings from the milling of the East Zone, Centre Zone and initial Main Zone ore, until such time as the Centre Zone Pit is ready to accept tailings. Similarly the Centre Zone Pit is sized to store tailings from the milling of the Main Zone ore. The Main Zone pit is sized more traditionally to optimize resource extraction. The Main Zone pit is the largest of the three open-pits mined at Kiggavik. As such it is also proposed to use the Main Zone pit for long-term management of Type 3 mine rock resulting from open-pit mining of the Kiggavik deposits.

8.2 Tailings Preparation

8.2.1 Waste Streams

In the Kiggavik mill, the following process waste streams will be received in the Tailings Neutralization Feed Tank:

- Pulp residue from the Resin in Pulp (RIP) Circuit;
- Waste regenerant from the Resin Regeneration Circuit
- Sand filter filtrate from the Uranium Precipitation Circuit;
- Water Treatment Plant (WTP) sludge; and,
- Discharge from the area Sump & Pump.

8.2.2 Neutralization and Thickening Processes

In the treatment process ferric sulphate ($Fe_2(SO_4)_3$), lime ($Ca(OH)_2$), and barium chloride ($BaCl_2$) will be used to promote the removal of metals and radium from the discharge water. These reagents will be contained in tanks in the reagent area of the plant and pumped to the tailings preparation circuit.

- Ferric Sulphate ($Fe_2(SO_4)_3$) will be added to the flash mix tank and/or the first Tailings Neutralization Tank. Ferric Sulphate can also be added to the 2nd Tailings Neutralization Tank in the event that the 1st tank is bypassed.

- Lime ($Ca(OH)_2$) will be added to the first Tailings Neutralization Tank to adjust the pH of the slurry to 4.0 and will be also added to the remaining two Tailings Neutralization Tanks to maintain a slurry discharge pH of between 7.0 and 8.0.
- Barium Chloride ($BaCl_2$) will be added to the 2nd Tailings Neutralization tank to reduce radium in solution.

There will be 3 Tailings Neutralization Tanks operating in series. The Feed Tank (flash mix) will discharge by gravity to the 1st agitated Tailings Neutralization Tank. The 1st tank and piping will be arranged to allow bypassing of each tank for maintenance with minimum disruption to the operation of the circuit. Process air or a small amount of hydrogen peroxide may be added to the bottom of the Tailings Neutralization Tanks, if required, to ensure that contaminants are in their oxidized state.

The last Tailings Neutralization Tank will discharge to the high rate Tailings Neutralization Thickener where a flocculant will be added to the feed pipe and/or feed well to promote settling of the solids. Tailings Thickener Underflow Pumps will transfer the thickened underflow at a nominal average of 38% (w/w) solids to the TMF for long term management of the solids. The thickener overflow will discharge to the Tailings Overflow Tank from where it will be pumped by Discharge Pumps to either the Water Treatment Plant (WTP) or the TMF.

A relatively comparable process has been proven to be successful at AREVA's McClean Lake Operation in reducing trace metals concentrations in tailings pore water. Based upon a review of Kiggavik laboratory test data and coupled with a series of geochemical models it is anticipated that the proposed neutralization process for the Kiggavik mill tailings will produce geotechnically and geo-chemically stable tailings with long-term uranium and trace metal concentrations of less than 1 mg/L.

8.3 Characterization

An assessment of the tailings that will be produced from the Kiggavik mill is provided in Technical Appendix 5J (Tailings Characterization and Management). A summary is provided in this section.

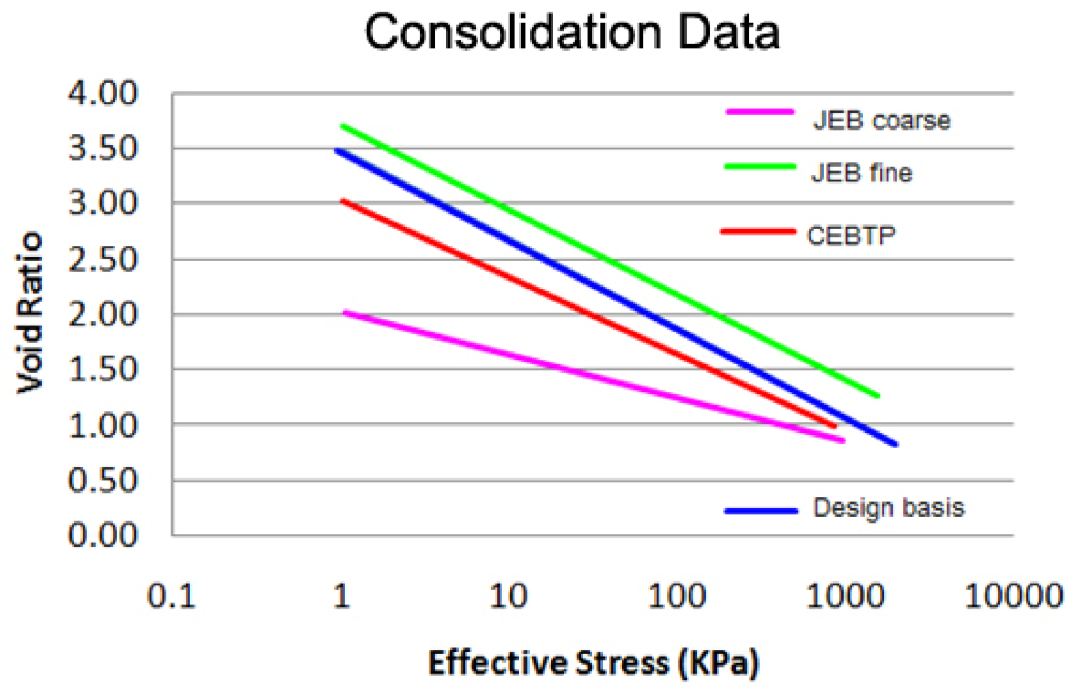
8.3.1 Geotechnical Properties

Geotechnical testing was conducted on laboratory produced tailings from the various ores to be processed at the Kiggavik mill. Additional tests were completed on tailings produced from the mill pilot plant testing program. The objectives of these tests were to estimate permeability, void ratio, water content and dry density during consolidation under increasing loads. Grain size distribution analysis was also conducted. Laboratory data was cross-referenced with field data from measurements at the McClean Lake Operation tailings management facility and the results were used to estimate field properties at Kiggavik.

Geotechnical properties of tailings that are of primary interest for predicting post-decommissioning effects include the density and hydraulic conductivity. The relationship between applied load and tailings density, or void ratio, was used to predict the volume that tailings will occupy within the decommissioned TMFs. The relationship between tailings hydraulic conductivity and void ratio, or effective stress, was used to predict the final hydraulic conductivity of consolidated tailings within the decommissioned TMFs.

Consolidation is dependent on hydraulic conductivity and it is known from observations at existing TMFs that it can range significantly – depending on particle size of the tailings from the mill as well as depositional segregation of coarse and fine materials. For the Kiggavik TMFs, the average solids content of the tailings in pit is predicted to increase from 40% to 70% during the consolidation process.

Figure 8.3-1 shows a comparison of measured, observed and design base properties. Based on laboratory consolidation test results, the hydraulic conductivity of the consolidated tailings is estimated to range between $4x10^{-9}$ m/s and $1x10^{-8}$ m/s. However it is considered that laboratory tests may not fully capture the complex consolidation processes and potential segregation between coarse and fine zones within a TMF. For that reason, the design base values were chosen to be somewhat conservative in their predicted time response for consolidation. Furthermore, observations at existing TMFs show that the actual field conductivity varies spatially and over time (as a function of consolidation) and, in general, is difficult to capture with great certainty in a model at the design stage. As a result, approximations and best judgment were applied based on a combination of measured and observed values and on the anticipated process at the site. As such a design value of $5x10^{-8}$ m/s was used for the tailings hydraulic conductivity in modelling of the consolidated tailings mass following cover placement (Technical Appendix 5J).



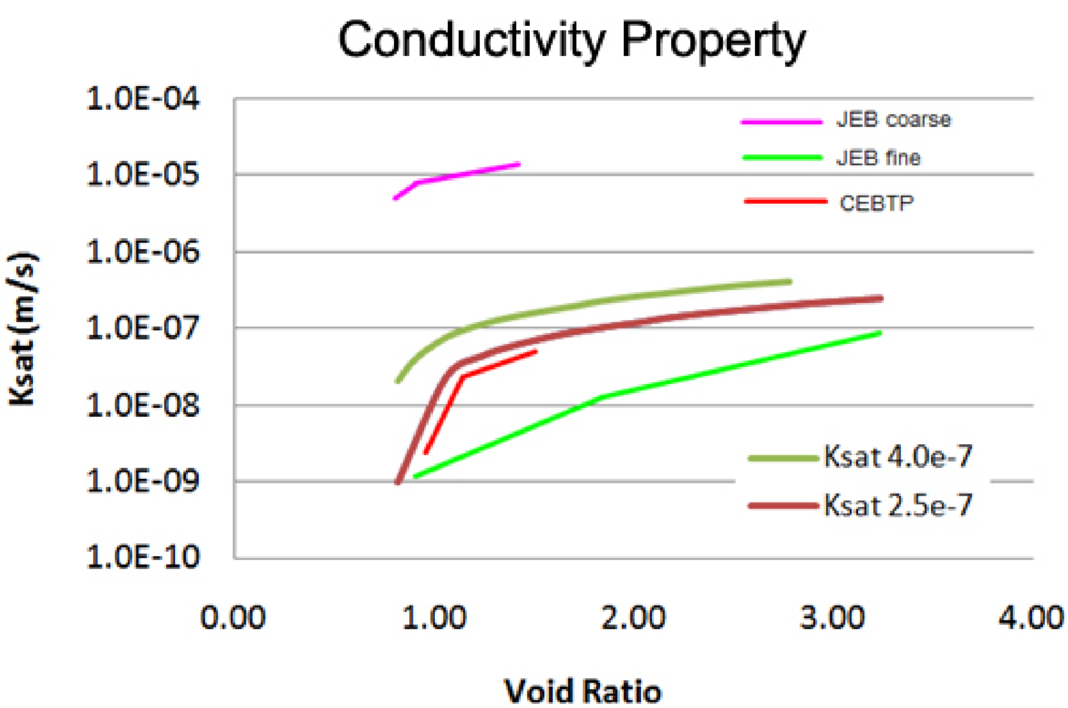


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FIGURE 8.3-1
VOID RATIO VERSUS EFFECTIVE STRESS AND CONDUCTIVITY

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8.3.2 Geochemical Properties

An evaluation of the data generated as part of the pilot plant testing program of Kiggavik Project ore samples was performed to determine the nature and extent of geochemical reactions that could potentially control uranium concentrations in the neutralized tailings. The analysis was first conducted for a worst case scenario with elevated uranium concentration in the tailings pore water under malfunction conditions. Typical operating conditions were then considered to produce tailings according to the flowsheet presented in Section 8.2. A series of geochemical models were constructed to analyze the test results and to estimate uranium and trace metal concentrations expected to be sent to the TMF from the tailings treatment circuit in the proposed Kiggavik mill.

A series of aging tests were then performed on these tailings to assess the changes in potential constituents of concern concentrations as a function of time. The assessment of the malfunction case test results focused on uranium since preliminary results suggested that other constituents, such as arsenic and nickel, are not expected to be of concern. The behaviour of all potential constituents of concern was assessed through the normal operational case laboratory tests.

From the aging tests a series of geochemical models were constructed to analyze the test results and to estimate the portion of chemical constituents available for transport out of the tailings mass. A long term tailings pore water concentration was predicted for each key constituent of concern. Table 8.3-1 summarizes the predicted long-term tailings pore water concentrations used to calculate mass flux for the decommissioned Kiggavik TMFs.

8.3.3 Thermal Properties

Thermal properties of the tailings under fully frozen and fully thawed conditions were estimated over a range of likely saturated water content. The measured values were used to back-calculate the likely quartz content and mineral specific heat capacity and then those values were applied to a range of water content and void ratios as may be experienced by the tailings. This is considered as a very accurate method of developing thermal model input properties.

The influence of dissolved ions on the freezing point depression was also estimated. Freezing point depression is a phenomenon by which water in porous materials does not freeze at 0°C. There are two main mechanisms that contribute to lowering the freezing point below zero. First, capillary forces in porous material develop surface tensions between water, ice and soil particles and this tension must be overcome by reducing the temperature in order to change phase. Secondly, dissolved ions in the pore-water, such as the addition of salts, can further reduce the freezing point depression. Calculation results show that the ion related freezing point depression is expected to be very limited, at about 0.18 degrees.

Table 8.3-1 Predicted Long-Term Tailings Pore Water Concentrations

Constituent		Units	Concentration in Tailings Pore Water
Aluminium	Al	mg/L	0.50
Arsenic	As	mg/L	0.02
Cadmium	Cd	mg/L	0.003
Chromium	Cr	mg/L	0.35
Cobalt	Co	mg/L	0.10
Copper	Cu	mg/L	0.40
Iron	Fe	mg/L	5.0 to 0.1
Lead	Pb	mg/L	0.04
Manganese	Mn	mg/L	10 to less than 1.0
Molybdenum	Mo	mg/L	0.20
Nickel	Ni	mg/L	0.40
Radium-226	^{226}Ra	Bq/L	10
Selenium	Se	mg/L	0.05
Uranium	U	mg/L	0.14
Vanadium	V	mg/L	0.70
Zinc	Zn	mg/L	3.0

8.3.4 Quantities

Approximately 11.5 million tonnes of tailings solids are expected to be generated from processing of existing Kiggavik Project ore sources based on the expected mill production schedule. For design purposes the containment volume was conservatively based on an average solids content of 40% (by weight) translating to a low dry density of 0.533 tonnes/m^3 while the average solids content of the tailings in pit is predicted to increase from 40% to 70%. Under this conservative assumption the containment requirement was estimated at 21 million m^3, approximately.

8.4 Operational Considerations

8.4.1 TMF Preparation

8.4.1.1 *Mining*

A number of pre-development activities, and activities to be conducted during mining, are required to prepare the TMFs prior to tailings disposal. These activities are described in Section 5 “Mining” and include:

- open pits rock slope designs (see Section 5.2.1.1);
- monitoring during mining (see Section 5.1.2.4); and,
- management of pit floor heave (see Section 5.2.4.2).

8.4.1.2 *Drainage*

The Kiggavik TMFs have been designed to rely on consolidation by upward drainage of excess pore water to the tailings surface only and a base drain has not been incorporated into the Kiggavik TMFs design. The excess pore water contained in the TMF ponds will be pumped as required to the Kiggavik water treatment plant for treatment.

Incorporation of a drainage system within the pits in a permafrost environment poses operational challenges. The efficiency of the underdrain option is dependent on the permeability of the drain. To be efficient, the drain needs to remain sufficiently permeable over time; that is the potential for permeability reduction factors such as chemical precipitation, sand plugging or freezing needs to be low. In permafrost conditions any underdrain installed with a pumping system in host rock risks freeze off.

8.4.1.3 *Pit Liner*

The Kiggavik TMFs have been designed to take advantage of the natural low hydraulic conductivity of the surrounding rock mass, regardless of the permafrost conditions, and a pit liner has not been incorporated into the Kiggavik TMFs design.

However it is recognized that a liner/barrier installed on the floor of the deepest part of Main Zone pit might be a reasonable TMF engineering option reinforcing the isolation of the tailings from the sub-permafrost aquifer.

The Kiggavik Main Zone environment presents specific features that would increase the efficiency of a base liner. This includes limited groundwater inflows under dewatered conditions due to the low hydraulic conductivity of the surrounding rock mass, including the sub-permafrost aquifer. As a result the liner would be installed under relatively dry conditions allowing for proper construction QA/QC. The liner would be constructed using suitable native materials and the hydraulic conductivity of the native materials would be lowered using an amendment material such as bentonite. In the Kiggavik area, potential sources of suitable native material include shallow overburden soils, metasedimentary and granitic rocks. Typically this material would be crushed and amended with bentonite to reach a low hydraulic conductivity, in the order of $1x10^{-9}$ m/s to $1x10^{-8}$ m/s, comparable to the hydraulic conductivity of the surrounding rock mass.

8.4.2 Tailings Pipeline

The Tailings Neutralization thickener underflow will be pumped to one of the three TMFs located south of the mill (Centre, East, and Main). Initially, only the East Pit will be available to receive tailings when the mill first begins operation.

Tailings will be pumped from the Tailings Neutralization circuit through a dual contained High Density Polyethylene (HDPE) pipe to the pump house, located south of the purpose built pit (PBP), and then directed towards the active TMF. The pipe will be dual contained, insulated, with a leak detection system and will be about 2 km in length. To prevent freezing, the insulated pipe will also be equipped with an electric heat tracing system.

The pipeline will run along a trestle and then along the ground, with permafrost protection measures in place if required. At road crossings the pipeline will go underneath the road through larger HDPE pipe. The current design considers that the pipeline will discharge through a tremie at a barge on the TMF. The barge will be complete with a walkway to shore, pumps, bubbler system to maintain open water around the barge, and will be enclosed. The barge will be movable to aid in the even deposition of tailings. Alternatives to the barge option, allowing for increased operational flexibility, will be assessed and may be proposed at the time of licensing application.

8.4.3 Tailings Thickener Overflow Pipeline

The Tailings Thickener overflow, will periodically travel by pipeline from the mill to the TMF. The flow will be pumped through the pump house to the barge, or alternative, directly to the TMF pond. The pipeline will be supported with the same system as the tailings line. The pipeline will be an HDPE dual contained pipe with insulation, heat trace and leak detection.

8.4.4 Benefit of the Water Cover

Thermal analysis of tailings deposition was modeled in a series of staged models which considered the change in elevation of tailings over time as well as the alternating surficial placement of tailings in winter and summer seasons. The modeling shows that over time, in the absence of mitigation measures, layers of frozen and unfrozen tailings could develop within the TMF. This result is supported by observation of an existing in-pit TMF in northern Saskatchewan which has less severe winter climate conditions. Based on these analytical and observed findings, deposition of tailings with the possibility of their freezing and affecting consolidation is deemed important to avoid.

Sub-aqueous deposition of tailings is one possible means to prevent the freezing of tailings during winter deposition. A water cover holds high latent heat which must be removed prior to temperatures at the base of the water dropping below zero Celsius, whereupon freezing of tailings would occur. Analysis was carried out to determine the minimum depth of water cover to maintain in place on a year round basis such that the temperature at the base of the water cover does not drop below the freezing point. Results show that with a 5 m deep water cover in place, the base temperature is buffered from the climate at surface and maintains a year round temperature of approximately 3°C, which corresponds with observed under ice lake temperature values in arctic environments.

Based on this analytical result and on observed arctic lake temperature profiles, it was concluded that a 5 m deep winter water cover would be maintained throughout the operational period of tailings deposition and early consolidation.

8.4.5 Tailings Deposition and Reclaim System

The tailings will be routed to a deposition system, which will be used to deposit the tailings subaqueously, using the tremie technique, under the surface of the TMF pond. The tailings will be deposited subaqueously near the surface of the previously deposited tailings on the bottom of the pond. As the tailings level builds up, the TMF water level and deposition pipe will be raised to maintain the proper tremie depth. To prevent winter freezing the pond will contain at least five metres of water cover.

Reclaim from the TMF will be pumped to the water treatment plant for treatment. The reclaim pumps will be located on the opposite site of the deposition system to prevent disruptions to reclaim collection from tailings discharge. Each TMF will have 2 reclaim pumps installed, one in operation with a second standby pump. Reclaim water may be pumped from more than one pit at a time, dependent on the pumping needs. The reclaim waters will be pumped to a tank in the reclaim pumphouse via a dual-contained, insulated, and heat-traced HDPE pipeline. The reclaim tank discharge will be pumped to the water treatment plant via a dual-contained, insulated and heat-traced HDPE pipeline. The pumping rates from each TMF will be controlled to maintain the minimum water cover in each TMF, and to provide a consistent flow to the water treatment plant that is within the plant design capabilities.

8.5 Tailings Monitoring

Contingency plans are intended to address unforeseen circumstances which could result in a significant increase in predicted environmental impacts. Extensive investigations into the chemical and physical properties of tailings have been undertaken at Kiggavik and will continue to be undertaken as part of a Tailings Optimization and Validation Program (TOVP), similar to the program that was initiated at McClean Lake Operation which has proven to be a successful audit program for the behaviour of the tailing produced at that site in northern Saskatchewan.

9 Water Management

9.1 Introduction

The proposed water management strategy for the Project is consistent with AREVA's experience in Northern Saskatchewan and with recent mining projects developed in sub-arctic conditions.

Kivalliq residents have expressed concerns about the environmental impacts of freshwater withdrawal (EN-CI KIA Apr 2007[97], EN-RB OH Nov 2010[98]) and effluent discharge (EN- BL HTO Mar 2009[99]). The overall objective of the strategy is to minimize both the intake of freshwater from lakes and the release of treated effluents to surface water receptors and to maintain the concentration of COPC's in the effluent at acceptable levels. Project design and water recycling are the key factors used to achieve this objective. Conservative assumptions have been used at this stage of design; continuous improvement will be used during operation to further reduce water consumption and discharge.

For the purpose of water management, contact water is defined as any water that may have been physically or chemically affected by site activities. Consistent with the philosophy to divert clean surface drainage prior to contact with the site, this definition does not include freshwater diversion around site infrastructure which will be accomplished using freshwater diversion channels prior to reporting to natural drainage pathways. Contact water includes:

- surface runoff from the mill terrace and mining areas;
- surface runoff from lined ore stockpiles;
- surface runoff from lined Type 3 mine rock stockpiles;
- water used in the mill process;
- water expulsed during tailings consolidation;
- tailings thickener overflow;
- water inflows into mines; and
- surface runoff from unlined Type 1 / 2 mine rock stockpiles.

[97] EN-CI KIA Apr 2007: *How much water will they use?*
[98] EN-RB OH Nov 2010: *Worried about the area where they get fresh water.*
[99] EN- BL HTO Mar 2009: *So the water that you are going to be discharging into Judge Sissons Lake, it's going to be treated?*

Residents have expressed concerns with runoff from ore and waste rock stockpiles (EN-BLC Mar 2009[100], EN-RI OH Nov 2010[101]). All contact water will be intercepted, contained, analyzed and treated when required. With the exception of runoff from the Type 1 / 2 waste rock piles at the Kiggavik and Sissons sites, all contact water will report either to one of the open pits and subsequently to the site water plant, or directly to the site water treatment plant. All water released to the environment will meet the discharge quality criteria; furthermore discharge quality objectives have been selected to minimize potential environmental effects associated with the release of treated effluent.

Non-contact water includes runoff originating from areas unaffected by site activities and that will not come into contact with mining and milling areas. Non-contact water will be diverted away from site activities to the surrounding natural drainage systems and is not subject to monitoring under MMER.

The overall water management strategy is based on the following concepts:

- minimize contact water by diverting local drainage prior to contact with the site and the use of snow fences to minimize on-site drifts;
- contain all contact water and provide contingency storage;
- contain all mine water;
- minimize freshwater usage in Mill processing; and,
- maximize recycling of on-site contact water.

Water management strategies for the Kiggavik and Sissons sites during operations are depicted in Figures 9.1-1 and 9.1-2, respectively.

At the Kiggavik site, the milling process is the primary consumer of freshwater and dictates the type of effluent treatment required. Other key components of the water balance are surface runoff and water expulsed as a result of tailings consolidation in the Tailings Management Facilities (TMFs).

At the Sissons site, freshwater is required primarily for potable use. Surface drainage and recycled water will be used for most industrial purposes.

[100] EN-BL CLC Mar 2009: *What will happen with water from wasterock?*

[101] EN-RI OH Nov 2010: *Will water from the ore pile be contained?*

The following sub-sections outline operational water requirements, freshwater sources and pumping, and water treatment plants for the Kiggavik and Sissons sites. Water management during construction is addressed in Section 12.9.7. Sewage will be treated in a separate sewage treatment process described in Section 14. A water management plan has also been developed and is included in Technical Appendix 2I.

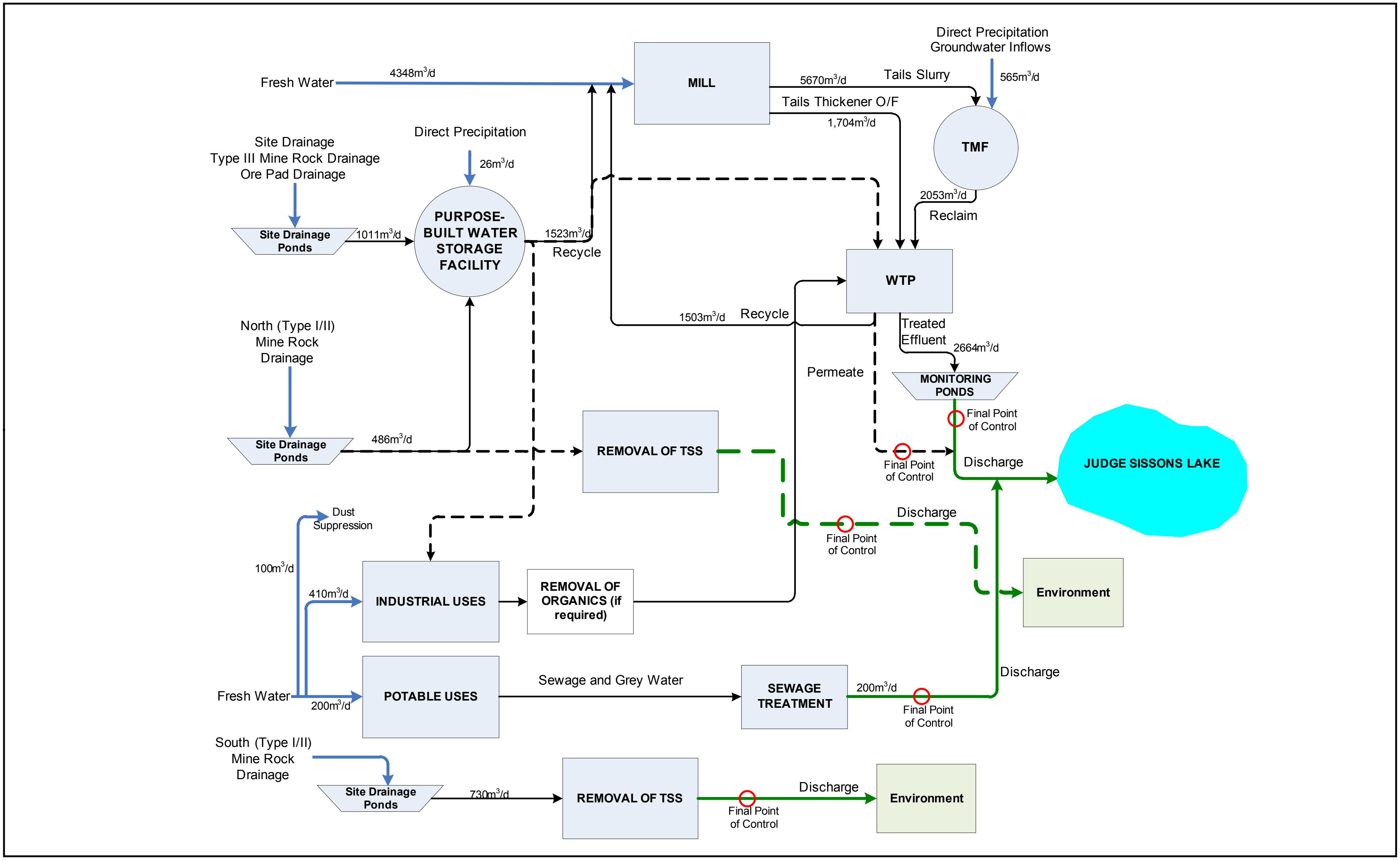

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FIGURE 9.1-1
KIGGAVIK SITE WATER MANAGEMENT - OPERATIONS

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File: Q:\SHEQ\GIS\KIGGAVIK\2014\EIS\Volume 2 - Project Description\Maps\MXD\Figure 9.1-1 Kiggavik Water Management Operations.mxd

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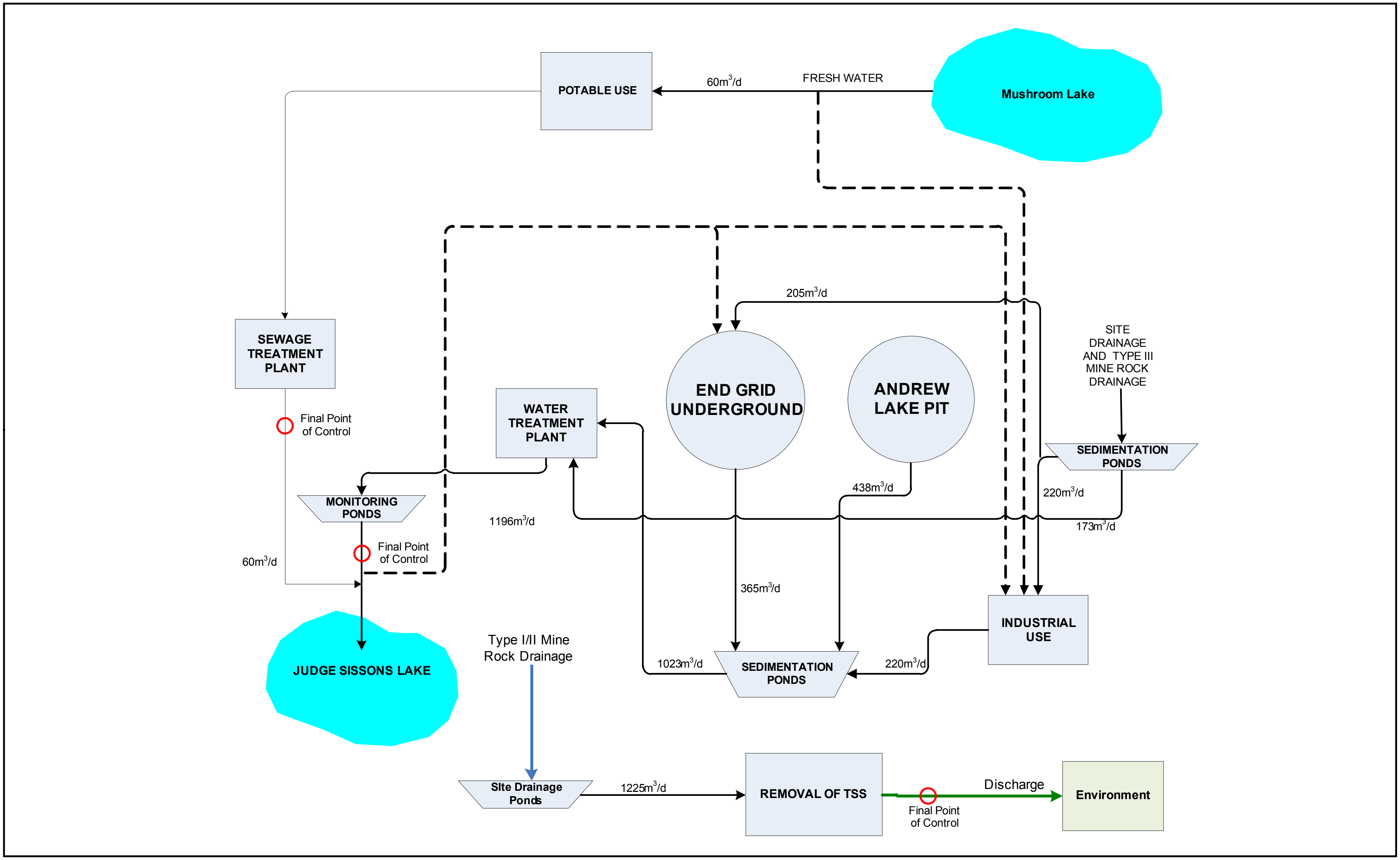

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FIGURE 9.1-2
SISSONS SITE WATER MANAGEMENT - OPERATIONS

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File: Q:\SHEQ\GIS\KIGGAVIK\2014\EIS\Volume 2 - Project Description\Maps\MXD\Figure 9.1-2 Sissons Site Water Management Operation.mxd

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9.2 Project Water Requirements

At the Kiggavik site, water is required for the following uses:

- potable uses;
- mill process and reagent preparation;
- mine shop and wash bays;
- dust suppression; and
- fire suppression

Some of these needs can be met using contact water rather than freshwater. Requirements for freshwater during mill operation are expected to range from 2,000 –8,000 m^3/day, depending on:

- the availability of permeate from the Kiggavik water treatment plant; and
- the availability of site recycle water from the water storage pit.

The mill process will be the primary consumer of water and the use of recycled water will be maximized. In the extreme case where no permeate or stored site drainage is available for use in the mill, a maximum of 8,000 m^3/d of raw water would be required to operate the Kiggavik site.

At the Sissons site, water is required for the following uses:

- potable uses;
- technical water for underground drilling;
- mine shop and wash bays;
- dust suppression; and
- pit re-flooding (decommissioning).

The industrial needs at Sissons will be met using recycled contact water, while freshwater will be required for potable uses.

At the Baker Lake dock site, freshwater will be required for potable use. Water for potable use may be hauled from the town supply.

Additional fresh water will be required for winter road ice building (flooding), dust suppression along the project area, and exploration drilling.

The overall volume of water required is summarized in Table 9.2-1.

Table 9.2-1 Estimated Overall Project Water Requirements (includes recycled water)

Use		Source	Estimated Water Requirements by Phase (m^3/year)		
			Construction	Operations	Decommissioning
Kiggavik Site	Potable (camp, mill, mine shops)	Fresh	73,000	73,000	18,250
	Mill, WTP and reagents	Fresh and recycled	15,330	2,691,510	31,025
	Mine shop and wash bays	Fresh and recycled	18,250	149,650	18,250
	Dust suppression (pits, roads)	Fresh	10,950	36,500	3,650
	Fire Suppression	Fresh	0	0	0
Sissons Site	Potable (camp, mine shops and underground)	Fresh	21,900	21,900	7,300
	Technical water (underground)	Recycled	0	74,825	0
	Mine shop and wash bays	Recycled	18,250	73,000	18,250
	Batch Plant	Recycled	0	7,300	0
	Dust suppression (pits, roads)	Fresh and Recycled (pit only)	10,950	18,250	3,650
	Pit re-flooding	Fresh (from spring freshet)	0	0	6,300,000
Access and Baker Lake Dock Site	Potable	Fresh	365	365	365
	Winter road flooding	Fresh	75,000	75,000	75,000
	Dust suppression (portion all-season road near Baker Lake community)	Fresh	1,200	12,000	2,400
Exploration Activities	Drilling	Fresh	6,500	19,500	0

9.3 Treated Effluent Discharge Criteria

Water treatment options have been developed to ensure compliance with the Metal Mining Effluent Regulations (MMER) requirements, and to maintain ecological risk assessment objectives to minimize the potential environmental effects of treated effluent discharge (see Technical Appendix 8A). AREVA will comply with treated effluent discharge criteria developed during CNSC and Nunavut Water Board licensing processes. In addition, best management practices, including the development of administrative and action levels, will be incorporated into the management of treated effluent discharge and to ensure ecological risk assessment objectives are achieved.

9.4 Freshwater Diversions

Two key methods are used to divert freshwater from the site and thereby minimize the volume of contact water:

- freshwater diversion channels; and
- snow fences.

Freshwater diversion channels are unlined channels that have been designed to isolate and divert clean runoff which drains toward site facilities (see Technical Appendix 2E). The runoff would be intercepted and directed around the core facilities areas and returned to the channels which carried the flow from these areas prior to development. LiDAR topographic surfaces and the GIS application ARC-Hydro was used to define contributing drainage areas and identify flow pathways. The US Army Corps of Engineers model HEC-HMS was used to derive peak flows and to accumulate runoff in the drainage areas and through the channel sections to the outlets. The channels are designed to carry runoff from a Probable Maximum Precipitation (PMP) which has been calculated for the Kiggavik area at 184 mm in 24 hours (Technical Appendix 4A). The use of the PMP in design is conservative and the proposed designs will be more than adequate for managing heavy rainfall and maximum snowmelt runoff. These designs will be optimized at the time of licensing application.

The freshwater diversion channel design strategy is to minimize excavation depths and use berms to control flow where possible. A minimum channel invert gradient of 0.2% has been adopted to facilitate drainage and reduce channel cross-sectional areas. Channels oriented perpendicular to runoff slopes will be shallow cut on the upslope side and bermed on the downslope side. Channels will be sloped and riprapped from erosional protection. Channel base widths will vary with longitudinal gradient, and with distance downstream as increasingly larger conveyance capacity is required. Construction approaches for the diversion channels are provided in Section 12.

Based on the results of hydrological investigations, the maximum peak flow is normally associated with spring snowmelt and occurs over a period of days or weeks in late May or June, with the duration of peak flows related to drainage area size and storage within the watershed. High flows may also be associated with extreme rainfall events. As peak discharges occur when areas below and adjacent to the channels remain frozen, the likelihood of introduction of large amounts for sediment into the channel is greatly reduced. Following the spring freshet, flow in the channels will drop substantially and in some cases be dry over much of the summer.

There is potential for some sediment transport where sediment laden runoff from adjacent disturbed areas could report to the channel. However, Best Management Practices for erosion and sediment control (Technical Appendix 5O) will be implemented with the intention of trapping sediment close to the source (i.e., silt fences, check dams in flow pathways, revegetation, etc.) during construction and operations where required. Where flow velocities exceed the erosion resistance, armouring in the diversion channels using riprap and geotextile where required will reduce the potential for erosion and sediment transport within the channels.

Figures 9.4-1 and 9.4-2 indicates the proposed channel alignments and flow directions for the Kiggavik and Sissons sites, respectively. The freshwater diversion channels will report to natural pre-development depressions or stream channels.

Snow fences will be installed at the Kiggavik and Sissons sites to limit snow accumulation on site and reduce water runoff from snow melt within the site. The snow fences will be located north of site infrastructure to reduce wind speeds from the prevailing winds. The reduction of wind speed will shorten the distance travelled by snow into site. The collected snow melt will be diverted during spring runoff.

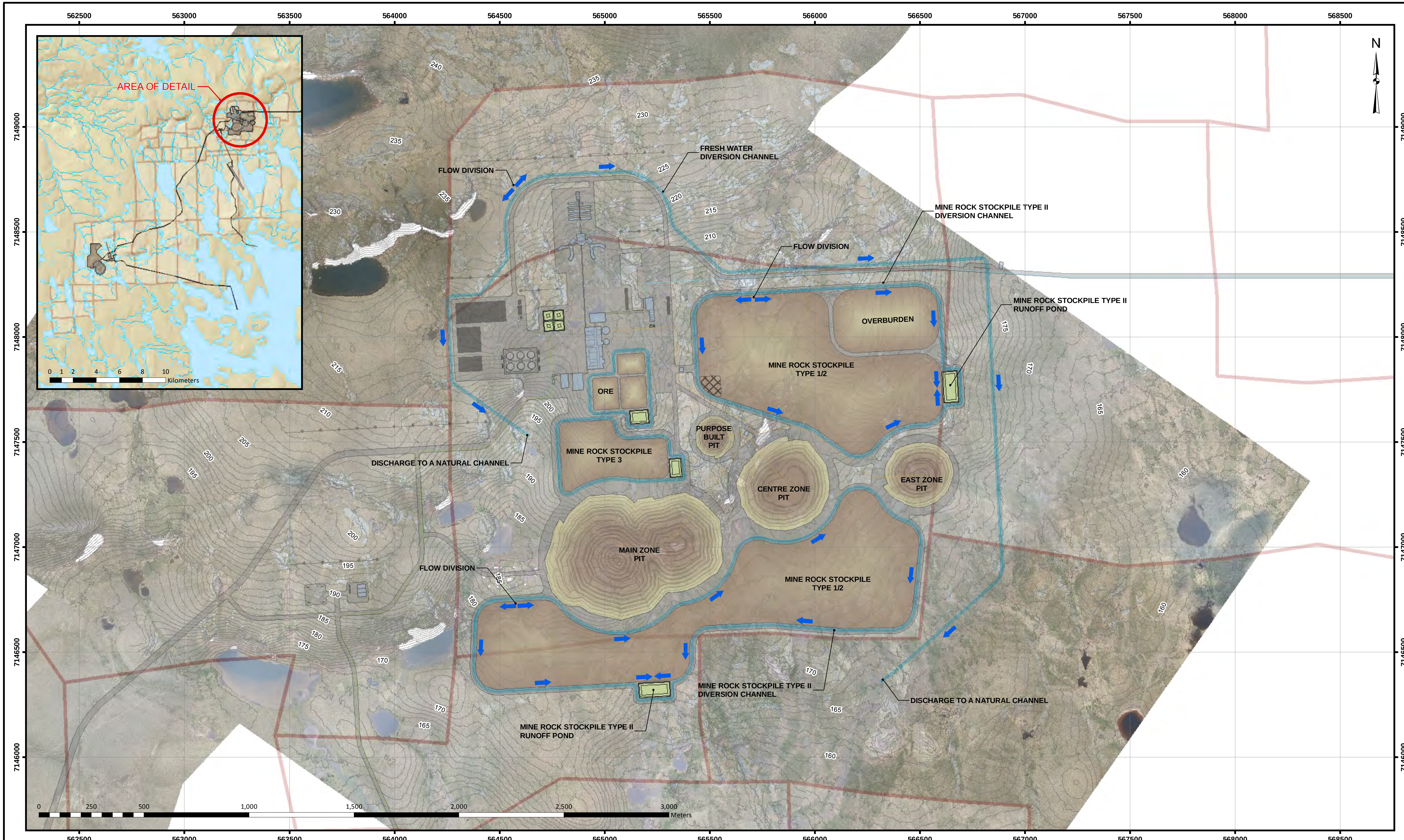

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FIGURE 9.4-1
KIGGAVIK WATER DIVERSION CHANNELS

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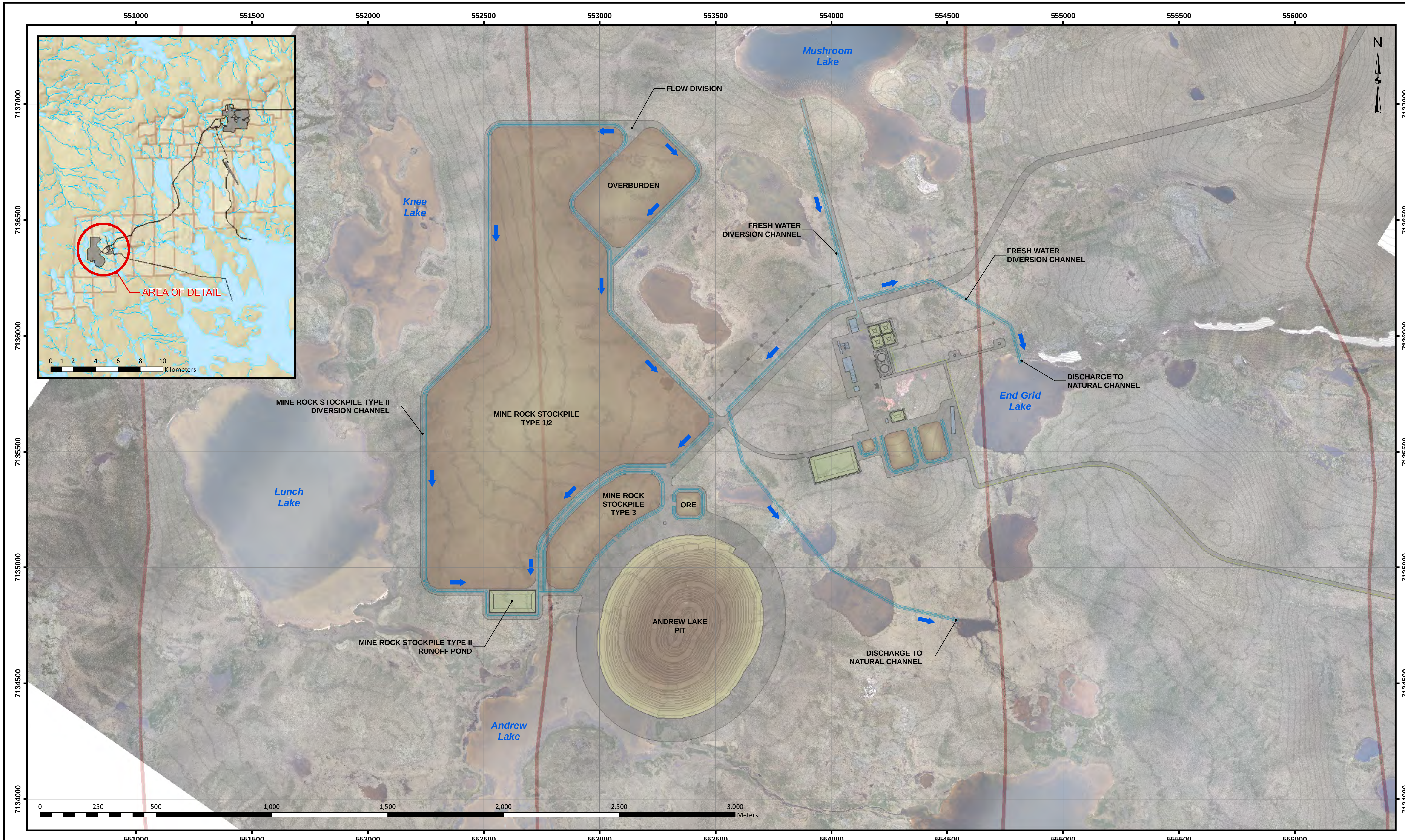

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FIGURE 9.4-2
SISSONS WATER DIVERSION CHANNELS

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9.5 Kiggavik Site Water Management

9.5.1 Freshwater Intake

The proposed freshwater pumping system for the Kiggavik site will primarily draw water from Siamese Lake. Siamese Lake is located approximately 8 km east of the site (Figure 9.5-1). Given average ice depths of 2 m, the screened intake will be located at a minimum depth of 4 m requiring the intake to be located approximately 400 meters off-shore. The main pumping station will be located along the shore.

The freshwater pipeline will be approximately 10 km long and will include allowances for elevation changes and snaking to reduce the risk of a pipeline break. The pipeline will be insulated, heat traced, and placed along a fill pad to prevent melting of the permafrost. A power line and maintenance road will be required in order to supply access and electricity to the heat trace and the pumping station. This road will also be used as the final portage of the winter road between Baker Lake and Kiggavik.

The freshwater pipeline will discharge directly into the Raw Water Tank in the mill for use in the mill processes. A backflow preventer will be used to ensure water from the Raw Water Tank cannot enter the freshwater line. A separate line will be teed off prior to the tank to deliver freshwater to the potable water membrane treatment system and the mine shops.

It is expected that the volume of freshwater drawn from the lake will vary based on permeate availability, seasonal variation, rate of tailings consolidation, and site activities. The system is designed for a maximum withdrawal rate of 8,000 m^3/day.

No treatment of freshwater to be used in the mill or mine shops is required.

Potable water will be treated to meet Canadian Drinking Water Guidelines and the Nunavut Public Health Act. It is anticipated that a membrane treatment plant, using either microfiltration or ultrafiltration, followed by a chlorination stage, will be required in order to meet these standards. During construction and the latter stages of decommissioning, a portable UV treatment system may be used.

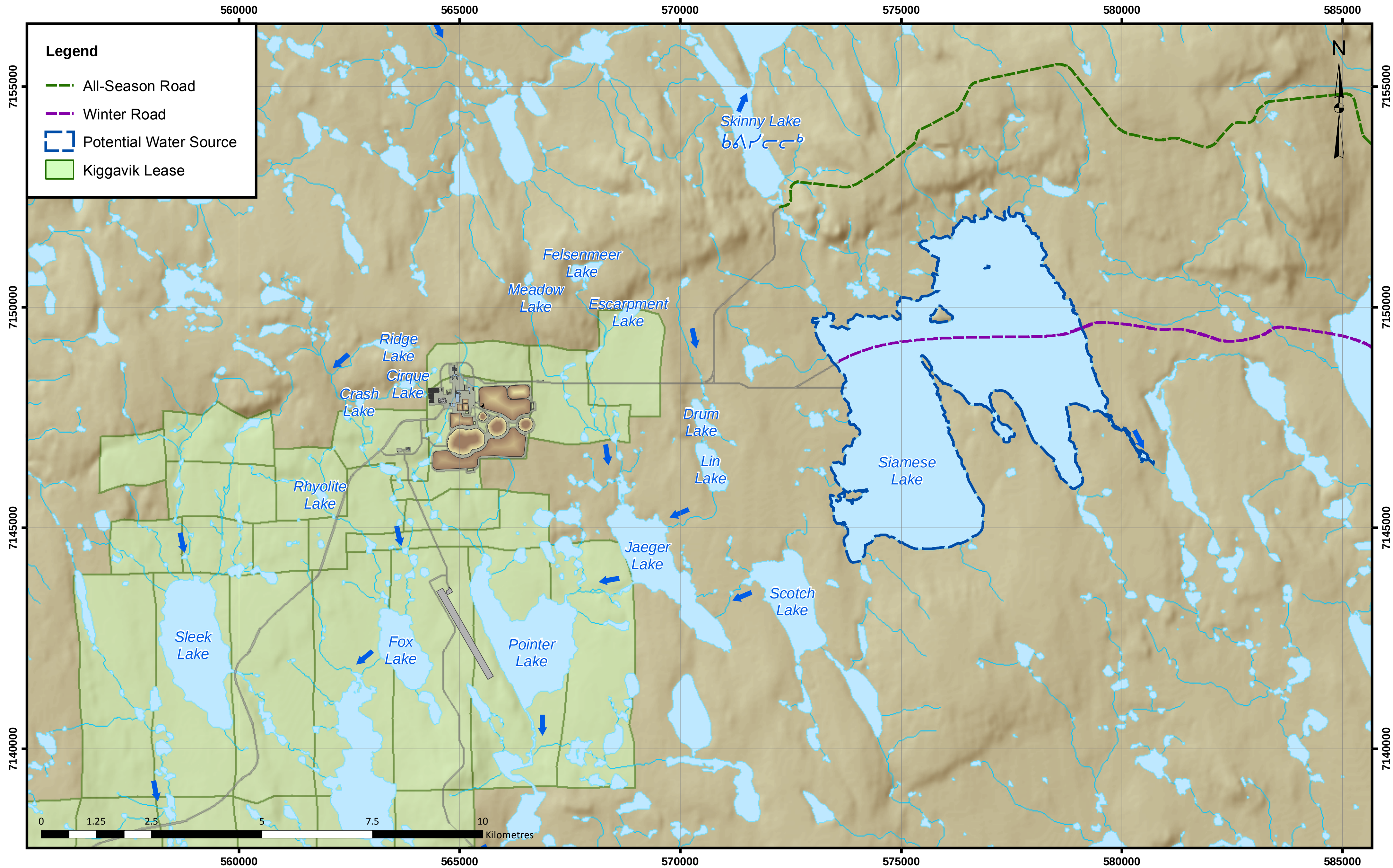

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FIGURE 9.5-1
KIGGAVIK FRESH WATER SOURCE – SIAMESE LAKE

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9.5.2 Management of Contact Water

Key components of Kiggavik site drainage management include:

- containment of all drainage that has contacted the site;
- treatment of drainage as required;
- storage and recycling of drainage to mill processes where practicable; and
- contingency storage through site design.

Contact water is defined as any water that may have been physically or chemically affected by site activities. Consistent with the philosophy to divert clean surface drainage prior to contact with the site, this definition does not include freshwater diversion around site infrastructure which will be accomplished using freshwater diversion channels. Contact water includes site drainage, Type 1 / 2 mine rock drainage, and mine water. It has been assumed that all site drainage, which includes drainage from the mill terrace, ore pad, and Type 3 mine rock pad, will either be recycled to the mill or pumped to the WTP. Mine water will also be either recycled or treated.

Drainage from the Type 1 / 2 mine rock pile will be stored in the purpose-built water storage facility or treated passively in unlined sedimentation ponds and released to the environment. The Type 1 / 2 waste rock piles are operational areas as defined by the federal Metal Mining Effluent Regulations; therefore, the monitoring methods, volumes, parameters tested, and QA/QC will meet the requirements of the Metal Mining Effluent Regulations. A contingency for discharge that does not meet MMER requirements would be to curtail discharge, and direct the Type 1 / 2 waste rock pile runoff to the site water treatment plant.

The monitoring locations, methods, volumes, parameters tested, QA/QC, and statistical rationale for monitoring contact water upstream of the water treatment plants will be determined by the project metallurgy department to ensure monitoring program requirements meet the objectives established for process control at each site. At each Final Point of Discharge, the monitoring methods, volumes, parameters tested, and QA/QC will meet the requirements of the *Metal Mining Effluent Regulations*.

9.5.2.1 *Site Drainage*

Site drainage will include runoff that contacts the mill terrace, ore pad, and Type 3 mine rock pad. This water will be contained using lined pads and ditches and collected in lined sedimentation ponds. Where practicable, this water will be pumped to the water storage pit for storage and use in the mill. Any non-recyclable water will be pumped to the WTP for treatment prior to discharge. Any pond sludges collected from the cleaning of the sedimentation ponds will be fed to the tailings neutralization circuit for treatment and ultimate disposal in the TMF. Figure 4.4-1 presents the pond locations.

9.5.2.2 ***Drainage Around Permanent Mine Rock Piles***

Part of the runoff management strategy within the site boundary consists of the construction of unlined drainage channels around the mine rock piles to collect runoff water for passive treatment in unlined sedimentation ponds to reduce potential suspended sediment loads prior to releasing flow to the natural receiving drainages. Runoff conveyance channels have been designed to manage the PMP, and lesser precipitation events, using the same methods as the freshwater diversion channels (Section 9.4). In most cases the pile exterior will form one side of the channel while a berm will constrain the flow within the desired alignments. Where topography dictates, channels may need to be excavated over some sections. These channels will drain according to topographic controls and discharge to unlined sedimentation ponds. Outflow from the sedimentation ponds will be to natural channel flow pathways or existing channels, with the exception of the north pile at the Kiggavik site, which will drain to the east freshwater diversion channel. Where channels encounter roadways, culverts will be used to facilitate cross-drainage.

9.5.2.3 ***Mine Water***

At the Kiggavik site, mine water will consist of direct precipitation and limited groundwater inflows entering the East Zone, Centre Zone and Main Zone open pits. During mining and TMF preparation, this water will be collected in pit sumps and pumped either to the WTP for treatment or to the water storage pit for recycling to the mill. Once the pit has been transformed to a tailings management facility, this water will contribute to the total volume of tailings reclaim water, which is pumped to the WTP.

9.5.2.4 ***Water Storage Pit***

Construction and operation of a water storage pit is proposed as a means of managing spring freshet. This pit will be located to reduce pumping distances to the mill, to take advantage of gradients and to provide contingency containment in case of failure of the primary site containment structures. During normal operation, recyclable water will be diverted to this pit for storage prior to use in the mill.

Water from the north Type 1 / 2 mine rock pile may be diverted directly to this pit using diversion channels. All other sources will be pumped to the pit as needed. The design storage volume of the pit is 350,000 m^3. Preparation of the pit includes placement of a pump and access. The level of the water is expected to be maintained approximately 5 m below the pit crest.

9.5.3 Water Treatment during Operations

Water to be treated at Kiggavik during the operational phase includes:

- open pit mine water;
- site drainage;
- reclaim water from the tailings management facilities;
- water from the mine shop / wash bay
- tailings thickener overflow; and
- domestic wastewater.

9.5.3.1 *Water Treatment Options Considered*

The following options were considered for the treatment of mill and site run-off water at the Kiggavik site.

- operation of the chemical treatment portion and discharge during the summer only with RO reject stored in an open pit during the winter;
- chemical treatment;
- two-stage reverse osmosis, concentration of RO reject in an evaporator, brine storage in an open pit;
- single stage reverse osmosis with chemical treatment of the RO reject;
- softening pre-treatment, followed by reverse osmosis with chemical treatment of the RO reject;
- returning a portion of the RO reject to tailings neutralization; and,
- ion exchange of the chemical water treatment plant effluent for additional Cd and/or Se removal.

9.5.3.2 *Water Treatment Process*

This section describes a base case water treatment option meeting the discharge criteria. It is anticipated that this option will be optimized as a result of on-going tests and studies. The design will be updated at the time of licensing application.

The process consists of a single reverse osmosis (RO) stage and a chemical precipitation stage. The RO stage is preceded by a simple chemical precipitation/flocculation and ultra-filtration pre-treatment. The chemical precipitation stage, for the RO reject, is a 3-stage treatment process for precipitation of metals. Permeate from the RO system will be recycled to the mill for reuse or combined with the chemical WTP effluent for discharge. A process block diagram is shown in Figure 9.5-2.

Pre-treatment

The reclaim water from the TMF, tailings thickener overflow, and runoff enter the precipitation tank via separate pipelines. This first treatment stage removes most of the total suspended solids (TSS). If heavy metal removal is required to precipitate heavy metals, lime is added at a pH of 10-11. Coagulant such as ferric sulphate may be added. The water flows into two reaction tanks, promoting the completion of the chemical reaction and the formation of crystalline solids (flocs). A clarifier is used to promote the flocs/water separation and ensures a turbidity of the clarifier overflow less than 10 NTU. Polymer is dosed before the clarifier in order to improve the conglomeration of the suspended flocs and thus to enhance the settling rate of flocs in the clarifier. The clarifier underflow (U/F) sludge will be extracted and returned to the mill tailings neutralization circuit. The pH of the clarifier overflow is adjusted with sulphuric acid in a pH adjustment tank to reduce the pH of the water to a pH of 8 – 9 prior to ultrafiltration.

A softening stage may be added to the pre-treatment if additional sulphate removal is deemed necessary.

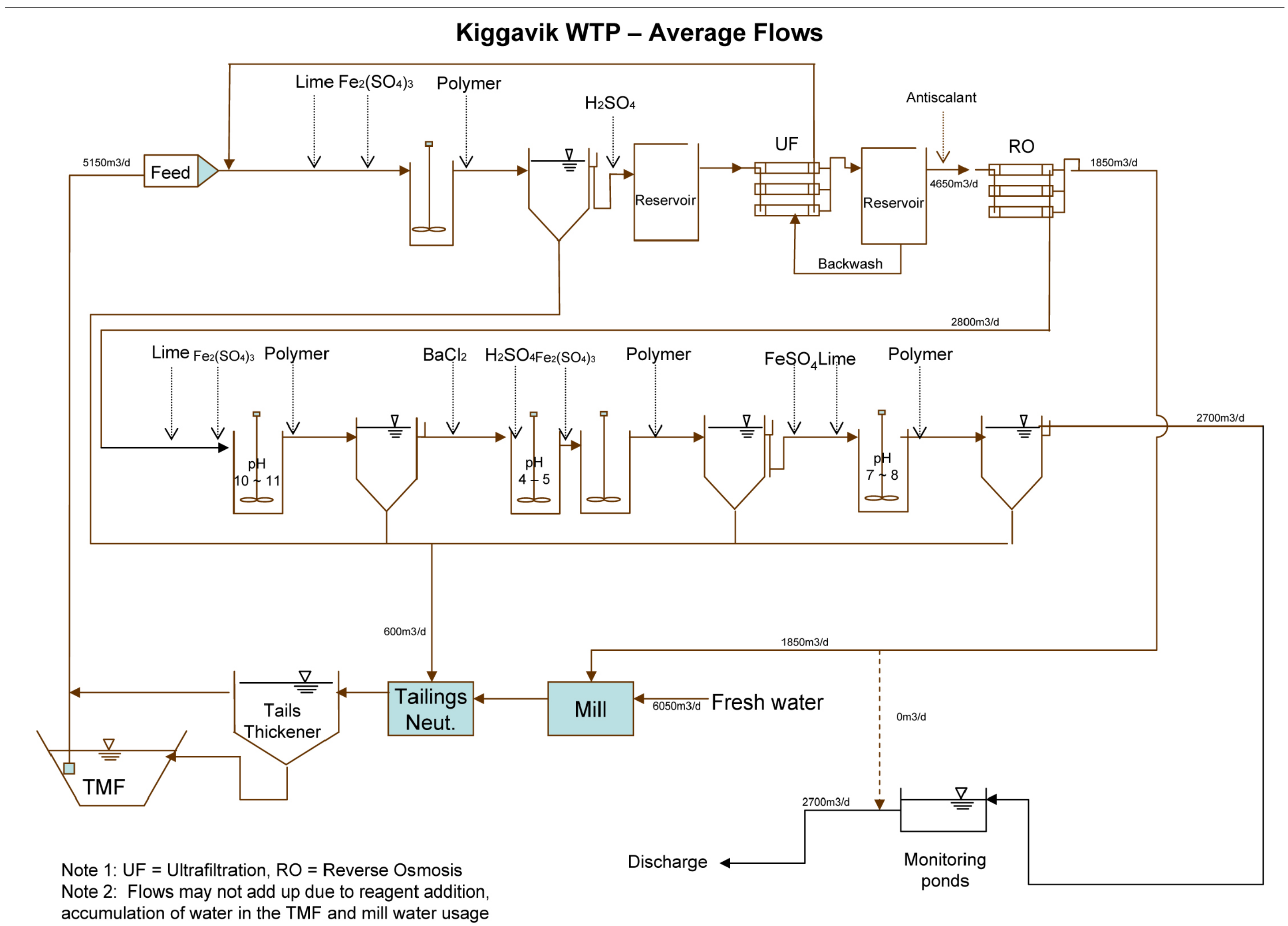

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FIGURE 9.5.2
KIGGAVIK WATER TREATMENT PLANT BLOCK DIAGRAM

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Filtration

The pH adjusted water will be filtered to remove any residual suspended solids and maintain the Silt Density Index (SDI) to a level which minimizes RO membrane fouling. The method of filtration is currently envisioned to be ultrafiltration (UF), which is described below, although other suitable filtration technologies such as multimedia filtration may be used.

The clarified water from the pH adjustment tank discharges to the UF trains by a low pressure feed pump. The filtrate from the UF flows into the RO buffer reservoir. A portion of this filtrate will be used for backwash of UF membranes.

Regular backwash (normally every hour) of the UF units is required because the permeability of the membranes declines due to solids build-up on the membranes. The backwash is not always 100% effective and after a certain period, the trans-membrane pressure (TMP) will increase. When the TMP reaches a design limit (e.g. 2 bars), a chemical cleaning in place (CIP) step will be required, followed by a regular backwash.

The wastewater of backwash will be returned to the plant feed.

Reverse Osmosis (RO) System

The filtrate from the UF stage feeds the RO system via high pressure pumps. Acid is added to the RO permeate as required for pH adjustment to approximately 7. The RO is a single-stage RO with a design recovery of 40%. The RO recovery was set at 40% based on a conservative estimate of the expected feed water quality to the RO. The RO feed water quality is expected to be saturated with gypsum in solution. In order to operate above the solubility limit, antiscalant is added to the RO feed. This is effective in reducing the scaling, however, there are limitations with the amount of total dissolved solids which can be present in the RO reject. Based on operational experience, and RO modelling software, the maximum recovery with the projected feed water quality was 43%. To be conservative, a recovery of 40% was selected. The estimated metals rejection is 99%. The RO system is anticipated to produce approximately 1,900 m^3/d of RO permeate, which contains low levels of dissolved solids. Permeate from the RO system is forwarded to a storage tank for use as reagent make-up water in the water treatment plant, recycled to the mill for reuse or discharged. The RO system will be monitored for feed and discharge pressures to provide an indication of the membrane performance. The permeate will be monitored for conductivity and pH, which may provide a preliminary indication of permeate quality. Any permeate which is deemed unsuitable for discharge due to off-spec pH or suspected contamination will be recycled. An autosampler will be in place on the permeate discharge line to collect a 12 hour composite sample for analysis. Reject from the RO will be treated by the chemical water treatment plant..

The RO portion of the WTP will be composed of multiple RO skids. The RO system will be designed so that it can operate at full capacity with one skid offline. The RO system will be cleaned one skid at a time. This skid will be taken offline and disconnected from the WTP process during cleaning. This will allow the WTP to continue operating at its normal capacity when one skid is being cleaned.

The RO membranes will be cleaned periodically (approximately once every 3 months) as required using a Clean in Place (CIP) step. During the CIP step, the RO skid will be isolated to prevent cleaning solution from mixing with the permeate and reject. Cleaning solution will be circulated to the membranes to remove any scaling or other material that has accumulated on the membranes. The cleaning waste will be either sent to the tailings neutralization circuit or to chemical treatment.

3-Stage Chemical Precipitation

The chemical precipitation process is designed to precipitate heavy metals, with 3 treatment stages as follows:

- high pH precipitation stage to precipitate nickel, uranium and others metals at a high pH (10 – 11) condition;
- low pH precipitation stage to precipitate radium, arsenic, molybdenum, selenium and other metals at a low pH (4-5); and,
- neutral pH stage to ensure the concentration of radium, arsenic and molybdenum in the final discharge, as well as the water pH, are in compliance with the discharge limits.

Each stage consists of reagent addition, two reaction tanks, with a minimum retention time of 40 minutes each and 1 clarifier. A retention time of 40 minutes was chosen for the reaction tanks to ensure sufficient time for metals precipitation. The third stage has only one reaction tank since this stage is mainly for pH adjustment.

The first stage of chemical treatment involves the precipitation of heavy metals at an elevated pH. The feed streams enter the first of two mechanically agitated reaction tanks. Lime slurry is added to the tanks to raise the pH to approximately 10 – 11 for precipitation of heavy metals. The overflow from the first tank enters the second tank. The second reaction tank overflows to the high pH clarifier. The clarifier separates the solid precipitates from the partially treated water. Flocculant is added to the clarifier to aid in solid/liquid separation. The underflow from the clarifier is pumped to the combined sludge tank.

Overflow from the hydroxide precipitation clarifier reports to the first of two low pH reaction tanks. Overflow from the first tank enters the second tank. Overflow from the second tank enters the third tank. Barium chloride is added to the first two reaction tanks to precipitate radium. Sulphuric acid is added to the first and second reaction tank to reduce the pH to approximately 4 – 5. Ferric sulphate is added to the second reaction tank to precipitate As, Mo, Se and other transition metals. The overflow from the second reaction tank reports to the neutral pH clarifier. The clarifier separates the solid precipitates from the partially treated water. Flocculant is added to the clarifier to aid in solid/liquid separation. The underflow from the clarifier is pumped to the combined sludge tank.

Overflow from the low pH clarifier reports to the neutral pH precipitation tank. Barium chloride will be added, if required, to precipitate radium. Ferrous sulphate is added for additional selenium removal. Lime is added to the second tank to raise the pH to approximately 7 -8 and to precipitate any residual metals remaining in solution. Overflow from the reaction tank enters the neutral pH clarifier. The clarifier separates the solid precipitates from the treated water. Flocculant is added to the clarifier to aid in solid/liquid separation. The underflow from the clarifier is pumped to the combined sludge tank. Overflow from the clarifier is pumped to one of the lined monitoring ponds. As the treated water is pumped to the monitoring ponds, a sample is taken periodically. The composite sample is sent to the chemistry lab for analysis to ensure the water meets the effluent discharge criteria prior to being discharged. Periodic toxicity testing will be done to meet MMER requirements

Sludge Management

The underflow from the clarifiers will be pumped to an agitated combined sludge tank. Depending on the amount of sludge produced, the sludge may be pumped on either a continuous or intermittent basis. Provisions will be in place to clean out the sludge lines with either air or water to prevent plugging of the lines. The sludge from the sludge tank will be pumped at a controlled rate to the mill tailings neutralization circuit for stabilization and ultimate disposal with the tailings. A centrifuge or other means of sludge dewatering may be used to increase the solids density of the sludge reporting to tailings neutralization. Any liquids from sludge dewatering will be treated in the water treatment plant. Details of the tailings neutralization circuit can be found in Section 8.

9.5.3.3 *Treated Effluent Quality and Quantity*

The WTP will have two effluent streams; RO permeate and chemical WTP effluent. The RO permeate will either be recycled to the mill for use in mill process, or combined with the chemical WTP effluent in the effluent pumphouse for discharge to Judge Sissons Lake.. The preference will be to recycle the RO permeate. RO permeate quality was based on operational experience for the performance of the RO pre-treatment, industry standards and the results of RO membrane simulations for the RO performance. If the RO permeate is not recycled, the water will be analysed for compliance to determine its suitability for discharge. If the RO permeate is not suitable for discharge, the cause of non-compliance will be investigated and the water will be temporarily diverted to the tailings management facility.

The chemical WTP effluent will be discharged to the environment. Average total treated effluent discharge from the Kiggavik WTP will be approximately 2,707 m^3/day, while maximum flows are estimated at 3,000 m^3/day.

Effluent quality was based on chemical solubility modelling and operational experience at the McClean Lake and Key Lake sites for treating similar types of water. The chemical treatment system for Kiggavik is similar to the McClean Lake and Key Lake water treatment processes, and the expected feed water quality is similar. Use of McClean Lake and Key Lake data provides a reasonable estimate of the effluent water quality.

Sewage is not expected to have significant levels of contamination, and therefore was not considered in the prediction of effluent quality. The projected discharge concentrations and annual mass loadings from treated effluent discharge from the Kiggavik water treatment process are shown in Table 9.5-1. This table includes estimated loadings of a wide range of parameters including those required to be monitored under MMER requirements. Cyanide is not used in the uranium milling process and the effluent is not expected to contain any cyanide.

Table 9.5-1 Projected Treated Effluent Quality and Loadings

Component	Discharge Concentration (mg/L)	Annual Loading (kg/y)
SO_4	2,199	2,407,905
Cl	237	259,515
As	0.021	23
Mo	0.200	219
Ni	0.020	22
U	0.002	2.2
Se	0.010	10.95
Si	5.8	6,351
Fe	8.3	9,089
Al	0.046	50
Cd	0.007	7.7
Co	0.007	7.7
Cu	0.002	2.2
Cr	0.007	7.7
Pb	0.002	2.2
Zn	0.003	3.3
Ca	470	514,650
Mg	29	31,755
Na	94	102,930
K	880	963,600
NH_3**	17.6	19,272
TSS	<15	N/A
pH	7 - 8	N/A
Ra-226*	0.008 Bq/L	8,760 kBq/y
Th-230*	0.011 Bq/L	12,045 kBq/y
Pb-210*	0.052 Bq/L	56,940 kBq/y
Po-210*	0.007 Bq/L	7,665 kBq/y

NOTE:
*Note that radionuclides concentrations are in Bq/L and loading is Bq/y

9.5.3.4 ***Treated Effluent Monitoring and Discharge***

The treated effluent is pumped to the lined monitoring ponds (Figure 4.4-1), which serve as the final treated water quality checkpoint before discharge. A composite sample of the water pumped to the monitoring ponds will be taken using an autosampler for chemical analysis to confirm effluent quality prior to final discharge to the environment. Should the effluent quality of the monitoring pond not meet discharge criteria, the pond will be recycled to the TMF or WTP and the cause will be investigated to reduce the likelihood that additional monitoring ponds do not meet the discharge criteria.

Lined monitoring ponds will be constructed, each with 12 hour effluent storage capacity. Under normal operation, one pond will be filling, one pond will be discharging, and the other pond will be awaiting laboratory analysis to confirm discharge criteria have been met. The addition of a fourth monitoring pond may be considered in order to provide operational flexibility. Treated effluent not meeting discharge criteria will be recycled to the TMF or WTP for further treatment.

Once it has been confirmed discharge criteria have been met, the monitoring pond discharge will be pumped to the monitoring pond pumphouse, where it will be combined with any RO permeate overflow. An autosampler will collect a sample of the monitoring pond discharge for confirmation of the initial analysis and that the pond meets specifications. RO permeate will also be sampled to confirm it meets specifications. The monitoring pond discharge and RO permeate that is discharged will be combined in an effluent discharge tank and pumped to Judge Sissons Lake. Grab samples for pH measurement will also be taken to confirm the pH is suitable for discharge. An additional autosampler will collect a sample of the combined discharge to track the effluent discharged to Judge Sissons Lake.

The effluent will be discharged to Judge Sissons Lake on a year-round basis. The water will be transported via an insulated, single-walled, heat-traced pipeline approximately 12 km in length. The pipeline design includes a berm and unlined containment ponds located at low points along the corridor. The berm will be designed such that the pipeline can be crossed by wildlife.

AREVA has experience operating effluent discharge lines during Canadian winter conditions. AREVA's McClean Lake Operation in Northern Saskatchewan has two effluent discharge pipelines from its Water Treatment Plants. The water treatment plants have operated both year-round and on a batch basis and effluent has been piped during all weather conditions, including northern winters. These lines are insulated and heat-traced. The discharge pipelines are approximately 5km and 7km in length. The pipelines have been operated successfully for 10 to 15 years without any major issues.

Containment ponds will be located at the low points of the pipeline to hold water in the event of a pipeline leakage or breakage, or if the pipeline needs to be drained. The containment ponds will be designed for at least two pipe volumes of the section of pipe that could potentially drain into the pond.

Discharge of treated effluent into Judge Sissons Lake will be accomplished using a diffuser system to promote treated effluent mixing. The diffuser will be located approximately 500 m from the shoreline of Judge Sissons Lake where a water depth of at least 5 m exists. The diffuser will consist of a weeping tile pipe.

The discharge of treated effluents to Judge Sissons Lake on a year-round basis will also necessitate a service road. The road will be 12 km in length and 5.1 m wide. A power line will be constructed alongside the road to provide power to the discharge pipe's heat trace system.

9.5.3.5 *Contingency Measures for Discharge*

Preliminary contingency measures for discharge during upset conditions have been developed. Table 9.5-2 shows the upset condition and the contingency measures to be taken.

Table 9.5-2 Contingency Measures for Discharge

Upset Condition	Contingency Measures
Liner break in monitoring pond	• Operate with 2 of 3 monitoring ponds • Reduce feed flow rate if required
Major leak in pipeline	• Temporarily suspend discharge of effluent • Temporarily divert treated effluent to TMF
Monitoring pond does not meet discharge criteria	• Recycle pond contents to front-end of water treatment plant or empty pond contents to TMF
Water treatment plant not operating and treated sewage effluent requires discharging	• Storage of treated sewage effluent, followed by intermittent batch discharge.

9.6 Sissons Water Management

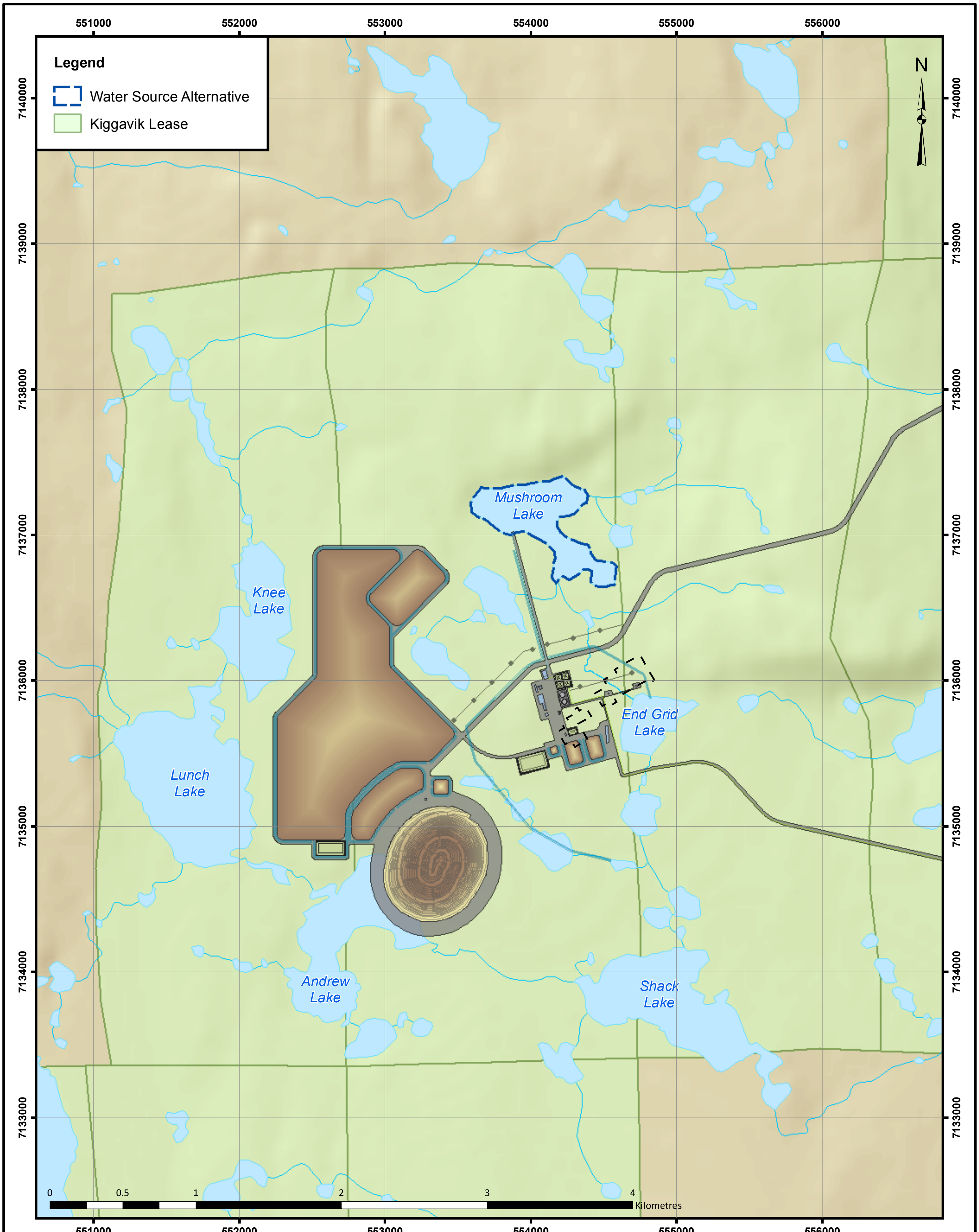
9.6.1 Freshwater Intake

The proposed Sissons freshwater pumping system will draw water from Mushroom Lake, which is located approximately 2 km north of the site (Figure 9.6-1). Given average ice depths of 2 m, the intake will be located at a minimum depth of 4 m requiring it to be located approximately 400 meters off-shore. The intake will be screened in accordance with DFO guidelines. The main pumping station will be located along the shore.

The freshwater pipeline will be approximately 2 km long and will include allowances for elevation changes and snaking, to reduce the risk of a pipeline break. The pipeline will be insulated, heat traced, and placed along a fill pad to prevent melting of the permafrost. A power line and maintenance road will be required in order to supply access and electricity to the heat trace and the pumping station.

Freshwater will be pumped via an electrically heat-traced line to a storage and potable treatment system and then to the mine dry and kitchen facilities for potable use. Potable water will be treated in a separate membrane treatment system with no connection to any mine water or sewage streams.

Freshwater requirements for the site are estimated at 60 m^3/day, consisting primarily of potable use.



Projection: NAD 1983 UTM Zone 14N
Creator: CDC
Date: 09/01/2011 Scale: 1:35,000
File:
Data Sources: *Natural Resources Canada, Geobase®, Nation Topographic Database, AREVA Resources Canada Inc.*

FIGURE 9.6-1
SISSONS FRESH WATER SOURCE – MUSHROOM LAKE

ENVIRONMENTAL IMPACT STATEMENT
VOLUME 2

Kiggavik Project

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