INFLOWS IN URANIUM MINES OF NORTHERN SASKATCHEWAN: RISKS AND MITIGATION

R. Bashir [1,2] and *J.F. Hatley[3]

[1]*Golder Associates*
1721 8th Street East
Saskatoon, SK, Canada, S7H 0T4

[2]*Department of Civil & Geological Engineering*
57 Campus Drive
University of Saskatchewan
Saskatoon, Saskatchewan, SK, S7N 5A9

[3]*Cameco Corporation*
2121-11th Street West
Saskatoon, SK, S7M 1J6
*(*Corresponding author: James_Hatley@cameco.com)*

ABSTRACT

Northern Saskatchewan comprises an area of about 350 000 km^2 and boasts some of the world's largest known high-grade uranium deposits. The successful mining of these deposits; however, can not be accomplished without overcoming technical challenges. Not only is the high grade of the uranium ore challenging, but mining methods needed to be developed to deal with groundwater at very high pressures, and ground conditions that vary substantially from excellent to wholly unconsolidated clays and sand. This paper discusses the various mechanisms of inflows at the three of the Cameco Corporation Mines in Northern Saskatchewan. The risk of inflows is quantified in terms of unique challenges from hydrogeologic conditions, rock mass integrity, and uncertainty in geologic conditions. Mitigation strategies in case of an inflow are also briefly described. The paper concludes by addressing the impact of institutional rules (corporate standards) on minimizing, monitoring, and controlling the probability and/or impact of unexpected inflows.

MET SOC
CIM ICM
Uranium 2010 - "The future is U"
Proceedings of the 3rd International Conference on Uranium
40th Annual Hydrometallurgy Meeting
Saskatoon, Saskatchewan, Canada
Edited by E.K. Lam, J.W. Rowson, E. Özberk

INTRODUCTION

The Province of Saskatchewan is home to the world's largest producers of fertilizer KCl and of uranium trioxide U_3O_8. With its world class mines of potash in the south and uranium in the north, Saskatchewan is indeed a major mining centre in Canada. Uranium deposits in Athabasca basin are of hydrothermal origin located along ancient fractures at the unconformity between the Archean basement and the overlying sandstones of Proterozoic Age (Figure 1). The Province is also blessed with an abundance of surface water and ground water. Most of the potash mines and the underground uranium mines have experienced significant unplanned inflows into their operations. Both potash and uranium mines have experienced significant challenges during the process of shaft sinking related to mining through completely different rock units and ground water regimes. This paper discusses the recent lessons learned by the mining operations staff of Cameco Corporation in the challenging Athabasca Basin located in northern Saskatchewan.

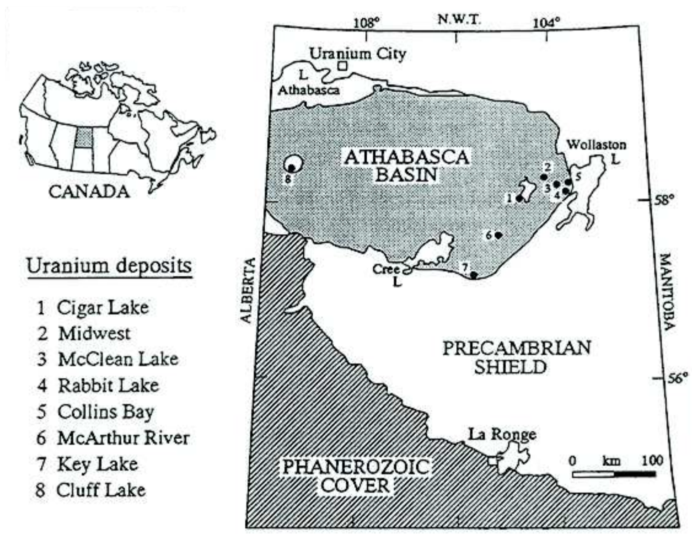

Figure 1 – Map showing the uranium deposits of the Athabasca Basin, northern Saskatchewan, Canada

TYPES OF MINE INFLOWS

Two of the primary challenges in mining the unconformity type deposits in Athabasca Basin are the control of groundwater and the ground support in areas of weak rock. These challenges occur concurrently in the immediate area of massive mineralization, in areas where the rock is fractured and faulted, and in the overlying sandstone. Hydraulic conductivity in the vicinity of the orebody is controlled largely by the presence of open fractures. The sandstone and the unconformity are known areas where water in significant quantities is present with pressures up to 4 to 5 MPa dependent on depth. These areas can produce significant flows if intersected by the mine development in the basement rock. The mine development to date has attempted to minimize the amount of water that can be encountered underground. This is accomplished through extensive cementitious grouting, careful placement of the underground excavations away from known groundwater sources (unconformity and sandstone above) whenever possible, as well as artificial ground freezing.

There are two general means by which water can enter the underground mine workings; via cased or uncased boreholes; and via geological structures. Geotechnical investigation holes are drilled into any planned mining areas prior to mining. These holes can intersect areas which have substantial amount of water and pressures in excess of 4 MPa. Loss of control during or after drilling can result in these holes being conduits for significant amounts of water. Flow rate potential via boreholes is limited by hole characteristics such as pressure, diameter, length, and smoothness and can be quantified rather accurately. In addition to the holes drilled from underground for geotechnical characterization, there is also a potential for encountering surface holes during underground development. These surface exploration holes might or might not be grouted and, with their passage through some 400 m of water saturated sandstone, can be a significant source of water. Moreover, in some instance, exploration holes were drilled from the bottom of a lake and can act as conduits to a large body of water.

The other mechanism of inflow comes from encountering a geological structure that is hydraulically connected to a significant source of water such as water saturated sandstone. The inflow rate in such a case is limited only by the local and regional hydrogeology. Potential inflow volumes from this mechanism could be much larger than open boreholes and their exact rate is difficult to quantify.

Previous to the McArthur River Operation 2003 inflow, maximum dewatering rates at operations were matched to typical borehole sizes that could be drilled from the mine workings or could be intersected by developing into ungrouted or incompletely grouted diamond drill holes. No plans were made to develop laterally into the water bearing areas such as the unconformity and sandstone, and cementaceous probe and grout covers were known to be effective pre-excavation methods of reducing or eliminating significant inflows. Prior to 2006 four shafts had been successfully sunk through the Athabasca sandstone with the use of (vertical) cementitious probe and grout covers.

McARTHUR RIVER OPERATION

The McArthur River Operation is underground uranium mine located in the eastern part of the Athabasca Basin in northern Saskatchewan, Canada, 80 km northeast of Key Lake, 270 km north of La Ronge, and 40 km southwest of the Cigar Lake deposit. McArthur River Operation mines high grade uranium ore using a unique non-entry raisebore mining method. The ore is ground and processed as slurry underground and pumped to surface where it is loaded into special containers and shipped to Key Lake for milling.

The mining of the McArthur River deposit faces a number of challenges including control of groundwater, weak rock formations, and radiation protection from very high grade uranium. Based on these challenges, it was identified during initial mining studies that non-entry mining methods would be required to mine the deposit. The raisebore mining method was selected as an innovative approach to meet these challenges and was adapted to meet the McArthur River conditions This method has been used to extract all the ore at McArthur River since mine production started in 1999 (McArthur 2008).

Hydrogeological Setting

The upper bedrock at McArthur River consist of 480 to 560 m of Athabasca Group sandstones which unconformably overlie crystalline Archean and Aphebian basement rocks. The mineralisation at the McArthur River Operation is associated with a major thrust fault zone known as the P2 fault, and the majority of the mineralisation occurs in a southeast-dipping thrust at the contact between the Athabasca Sandstone and underlying basement rocks. The thrusting has resulted in a large wedge of basement rock overhanging the younger sandstone along the P2 Fault.

Six major hydrostratigraphic units have been defined during the hydrogeological investigations. From stratigraphically highest to the lowest, these include: (1) Overburden, (2) Sandstone, (3) Fanglomerate (and/or conglomerate) with a basal paleo-weathered zone, (4) Unconformity, (5) Mineralised zone, and (6) Basement rocks (Figure 2).

The overburden is a relatively thin unit, averaging about 10 to 15 m thick with a maximum thickness in the immediate area of about 50 m in the drumlin to the east of the mine (Figure 2). Its hydraulic conductivity ranges from 0.1 to 1.0 m/day. The underlying Athabasca sandstone, which is about

550 m thick in the mine area, has been locally differentiated into 4 sub-units: the MFd, MFc, MFb, and MFa (where MF refers to the Manitou Falls formation). The sandstone is well indurated and cemented and has very little primary permeability (except where locally de-silicified). Fractures, however, induce a bulk hydraulic conductivity in the range of 0.02 to 0.5 m/day. Figure 3 shows the horizontal hydraulic conductivity (K_h) with depth in the sandstone measured by packer and drillstem tests. Simple statistical analyses indicate that there is no significant decrease in K_h values for the entire sandstone except in the bottom 50 m. The K_h of the bottom 50 m of the sandstone appears to be about one order of magnitude less permeable than in the overlying sandstone. The fanglomerate and paleo-weathered zone is a relatively thin unit with a thickness ranging from about 10 m to 30 m. Its hydraulic conductivity values range from about 0.002 to 0.08 m/day. The underlying unconformity zone is generally less than 10 m thick. Two measured values of hydraulic conductivity for this unit averaged 0.005 m/day. The basement rocks are much less fractured than the sandstone, and existing fractures are generally infilled with clay-like gouge material. Measured values of hydraulic conductivity for the basement rocks are in the range of 0.002 to 0.05 m/day.

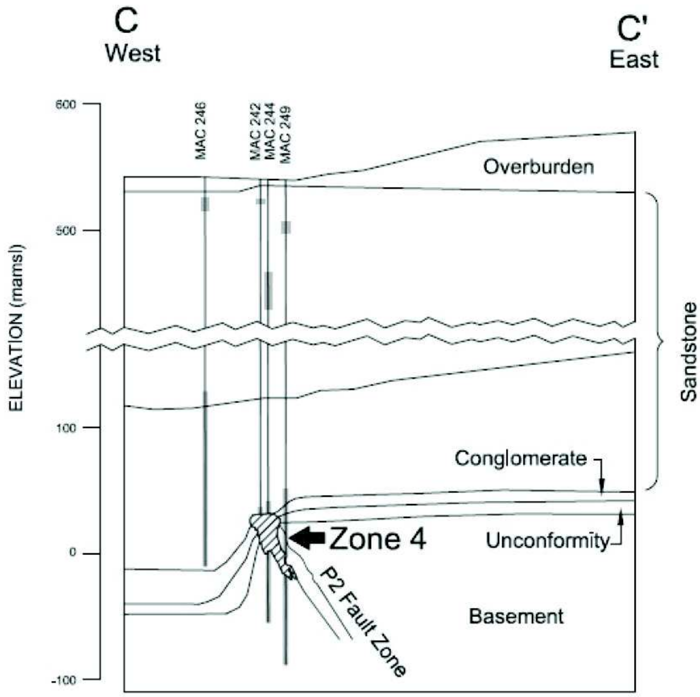

Figure 2 – Cross section of stratigraphy and orebody

Mine Inflow of 2003

On April 6, 2003 a ground fall occurred in the 7320 East Freeze Drift on the 510 m sublevel resulting in a large inflow of water. The initial inflow estimate was approximately 1050 - 1,100 m^3/h including the existing background inflow of 225 m^3/h. The installed pumping capacity at the time inflow was in the range of 450-500 m^3/h. Mining operations ceased immediately, and water in excess of pumping capacity was stored at various locations underground. Additional pumping capacity was installed. Underground water handling modifications were also made by addition of a number of low-head and high-volume pumps to move water strategically around the mine. Figure 4 shows a detailed approximation of the inflow and outflow rate from the mine.

The 7320 East Freeze drift on the 510 m sublevel was being excavated in proximity to the unconformity and P2 fault in preparation for freeze drilling of Zone 2. The drift was to be advanced below the unconformity and experience indicated that the 8 to 10 metres of rock separating the drift from the unconformity would be a sufficient buffer. It was not expected that the 5 MPa of water pressure in the sandstone layer above the unconformity would come into play. Unfortunately, this assumption proved incorrect. Prior to the drift development, McArthur River Operation spent three months drilling 28 holes averaging about 35 metres each into the area and the most amount of water encountered was 10 m^3/h in one of the holes, which was well away from the planned development. Mine development relies heavily on the results from probe drilling ahead of development, and hundreds of probe and grout drilling covers at McArthur River Operation have proven successful. Unfortunately, this drilling may not, at times, hit the water bearing structures. But, in reality, it is the best method to predict what lies ahead.

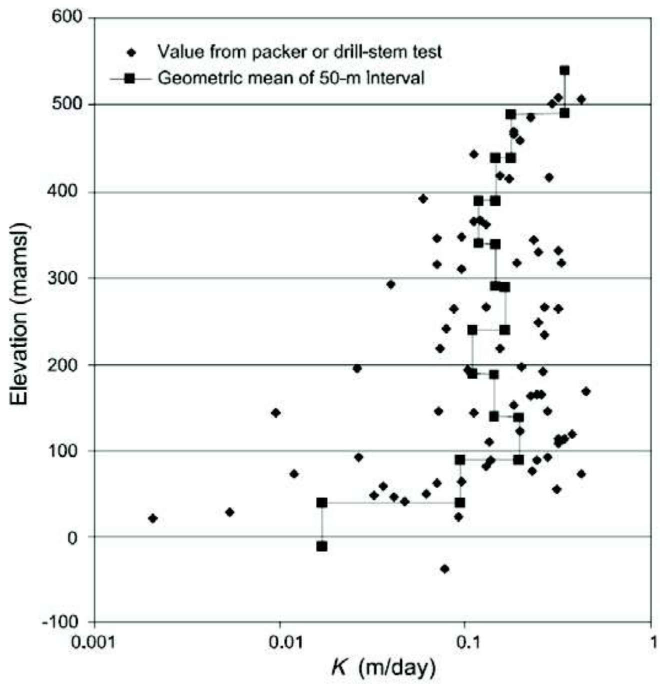

Figure 3 – Hydraulic conductivity vs. depth in sandstone

The drift was originally designed to be 6.5 m wide by 6.5 m high. However, due to deteriorating ground conditions, and to reduce the amount of radon laden water entering the drift, it was decided to install an Armtec arch in the opening. This resulted in a change to the size of the drift to 7.5 m wide by 7.5 m high to accommodate the installation of the Armtec arch. Normal ground control support continued with 2.4 m long Dywidag rockbolts, 1.8m long split set bolts and screening, 75 mm primary and 50 mm secondary shotcrete, and three layers of IBO Ankor Bolts spiling. Utilizing this support system had resulted in no significant issues elsewhere in the mine, including drifts that were in proximity to the unconformity and P2 fault. Cable bolting was not utilized due to the proximity of the unconformity which was estimated to be 8-9 m above the excavation. The Armtec arch installation lagged behind the drift excavation due to fears of blast damage if the culvert was installed to close to the working face.

A number of theories have been hypothesized for the mechanism of ground failure and the subsequent water inflow. However, there is a general agreement that there would have been no significant water inflow if there had not been a ground fall. A plausible sequence of events could be summarized as follows based on one of many theories. The back and walls of the drift were converging due to local low rockmass strength / stress conditions on to the steel Armtec Arch - causing differential loading, stress concentration at top of the arch. The Armtec Arch bent and stress relief occurred as a new fracture set formed above the drift and extended into the unconformity. As the distance between the unconformity and back of the drift remains the same the general hydraulic gradient remains constant for the drift but locally in the new fracture set, the water pressure of 5 MPa is introduced, very close to the excavation surface. This would have happened in a very short period of time and the rockmass in this area would have tried to reach a more stable state by expanding to relieve pressure. The continued movement in the back and walls to relieve pressure would have resulted in a small amount of inflow with instantaneous release of high pressure. The flow, initially would have been small, but would have continued as more joints unraveled and "key" blocks fall out of the back, allowing for low pressure water to saturate more joints near the excavation. The relaxation would have also allowed the area bulk permeability to increase markedly as the clay material would have washed out of the fractures in the rock above. As more material washed out, the inflow increased. This would have continued until the system could not supply more water due to permeability constraints and/or depletion of the local storage. As the inflow would have eroded new channels over the next while a pyramid shaped void was formed by the missing key blocks resulting in increased inflow. The collapse of the entire drift followed extending all the way up to the unconformity possibly for a 20 metre strike length, resulting in development of a maximum inflow rate.

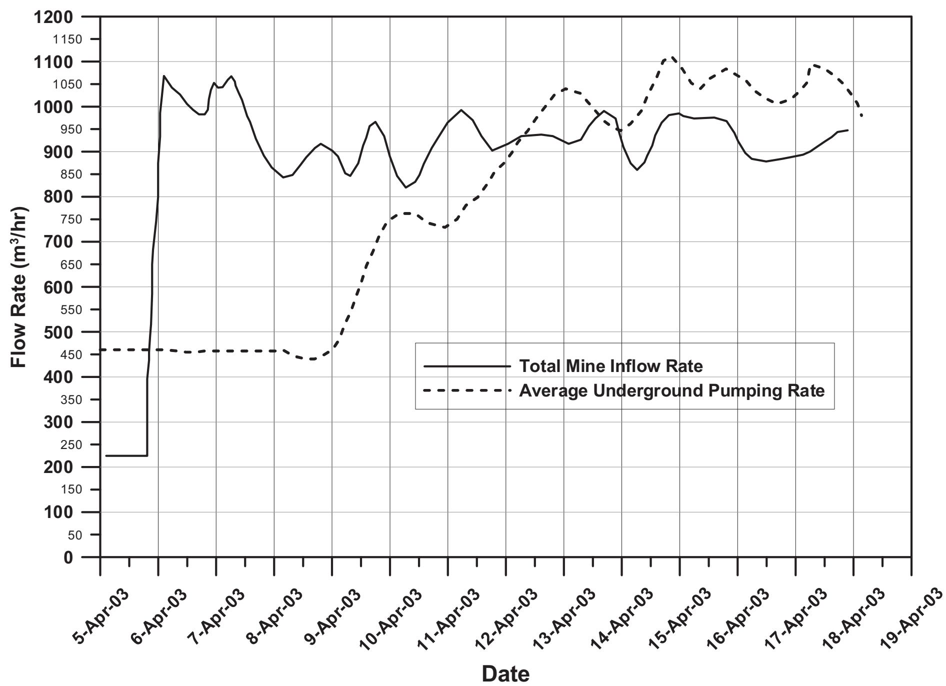

Figure 4 – Total mine inflow rate and average underground pumping rate

<u>Remediation</u>

All inflows are unique, and require careful planning to remediate. The location of the 2003 inflow at McArthur River required one hydrostatic bulkhead, and as a precautionary measure one high strength concrete fill and two lower strength concrete fills to insure integrity of the nearby tunnels before the inflow could be shut in and grouted back, see Figure 5.

There are a number of in-the-field, adaptive management decisions that are required to be made at the implementation stage of bulkhead building which were carried out by professionals including senior mine staff and mining contractor, see Figure 6. For McArthur River, the choice between structural bulkheads or monolithic bulkheads was suitable. Structural bulkheads would generally be 2 metres thick, notched into the surrounding rock and contain inner rebar mats on the dry face to counter bending forces. Monolithic bulkheads would be generally be 8 metres thick at this depth, would not be notched and would have no reinforcement. Shear strength of the rock/concrete interface is used to resist hydrostatic forces. The advantage of monolithic bulkheads is that they limit the possibility for the inflow to bypass the bulkhead via the surrounding rock. The disadvantage is the greater need for concrete which can be logistically difficult; however McArthur River Operation has the ability to place the pour requirements.

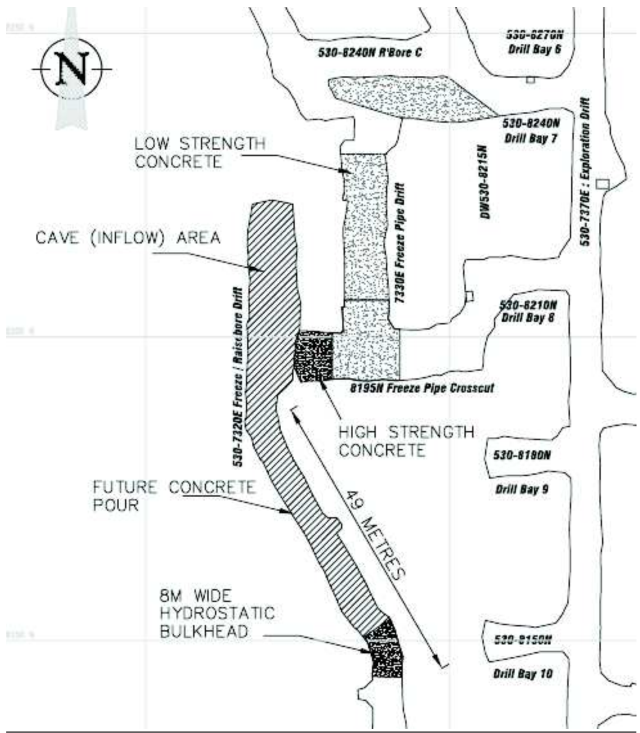

Figure 5 – McArthur River 2003 inflow hydrostatic bulkhead and concrete fill locations

Due to the volume of water in the 2003 inflow at McArthur River, a culvert was used to direct the flow away initially so flanged piping could be installed, see Figure 7. Two bulkhead walls were created, and in-between these walls an absolutely dry length was cleaned for a monolithic concrete pour or plug. During construction of the plug, water was controlled through valved flanged pipes, allowing time for the concrete to set without pressure on the bulkhead. Once the concrete set, the perimeter around the plug was sealed with cementitious grout by drilling from the dry side, through the centre of the bulkhead radiating out along the bulkhead and rock interface. Once the monolithic plug interface was sealed, a careful program of closing the valves and pressurizing the bulkhead commenced over many months. This program involved detailed water pressure and geotechnical stability monitoring of the bulkhead and the surrounding tunnels. Approximately one million kilograms of cement was used in cementitious grouting around the inflow cavity, and into the immediate area at the unconformity and above into the sandstone. This area, also referred to as Zone 2 Panel 5, was successfully redeveloped in 2009 using ground freezing, careful excavation (Yameogo, 2010), and applicable controls described in the lessons learned section of the paper.

Figure 6 - Adaptive management was required by experienced professionals and mine contractor

Figure 7- Initial timber bulkhead wall

Cigar Lake Mine

The Cigar Lake Project is an underground uranium mine under construction and is situated in Northern Saskatchewan, 70 km from McClean Lake and 660 km north of Saskatoon. The Cigar Lake orebody is situated about 430 meters below surface at the unconformity between metamorphic basement rocks and flat lying sandstone. The deposit is characterized by a series of geochemical alteration haloes geometrically arranged around the orebody, decreasing in intensity with increasing distance from the ore surface. These haloes comprise a clay-rich zone (mainly illite) of varying thickness (up to 12 m) immediately surrounding the orebody and mostly derived from the hydrothermal alteration of the host sandstones. Figure 8 shows the schematic geological cross-section of the Cigar Lake orebody and existing underground development.

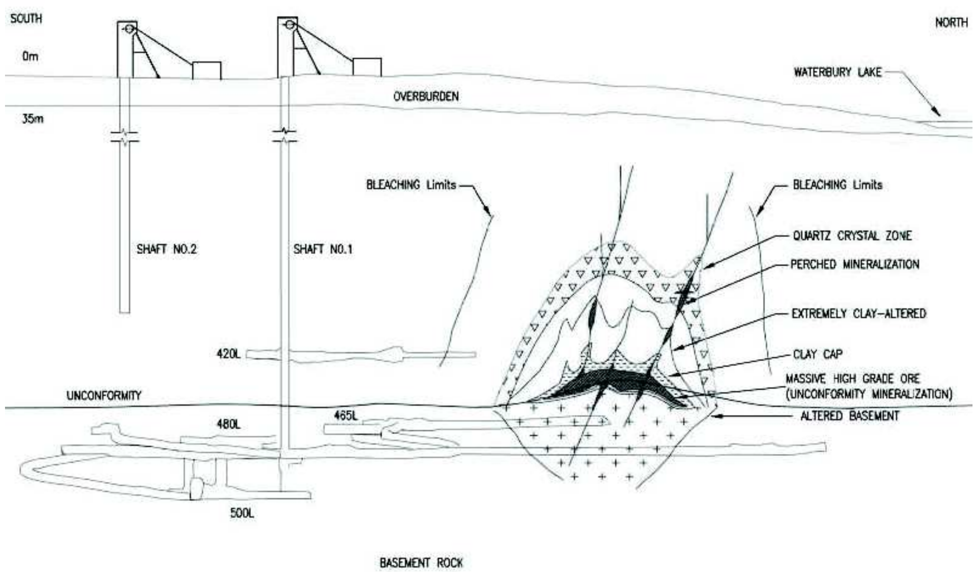

Figure 8 – schematic geological cross-section of the Cigar Lake orebody and existing underground development

Major technical factors influencing the selection of a mining method include ground stability, control of ground water, radiation exposure and ore handling and storage. The deposit is from 20m to 100m wide and about 2150m long. It is crescent shaped in cross-section and averages about 6m thick, with a maximum thickness of 15m. Ore at Cigar Lake will be broken with jets of pressurized water and removed in slurry form through steel piping. The ore will be pumped to surface and loaded into special containers and trucked 70 km to McClean Lake and Rabbit Lake for processing.

Hydrogeological Setting

One of the earliest descriptions of the hydrogeologic setting of the Cigar Lake Deposit can be found in Winberg & Stevenson (1994). They conceptualized the groundwater flow system at Cigar Lake to be composed of three flow regimes: a) superficial regime with predominant flow in the overburden and the upper part of the weathered sandstone, b) an intermediate flow regime with water recharging in the upstream end, partly discharging into Waterbury Lake and partly feeding the lower sandstone, and c) a lower semi-regional regime that comprises water in part recharged beyond the limits of their modelled system, and in part being fed by water percolating through the overlying strata mostly via discrete fracture zones. The semiregional groundwater flows primarily within the Lower Sandstone; final discharge for all regimes is Waterbury Lake as suggested by tracking of water particles from their local model into the regional model. Groundwater flow at the depth of mineralisation is horizontal, from south to north, with an average hydraulic gradient of approx. 1%

The sandstone in the Cigar Lake Mine area is relatively quite permeable, especially at depth. This is a rather unusual relationship of hydraulic conductivity with depth (the norm being a decrease with depth). A possible explanation this increase is that they are a result of the fluids that moved through them and above the unconformity as part of the ore-forming process. Figure 9 shows the horizontal hydraulic conductivity (K_h) values derived from the packer tests in the three coreholes drilled as shaft pilot holes. Assuming the ground-water system is essentially hydrostatic (i.e., there are no significant differences in hydraulic heads with depth) and the phreatic surface above the site is at a depth of about 20-30 m, the pore pressure at the unconformity at a depth of 448 is about 4.2 MPa

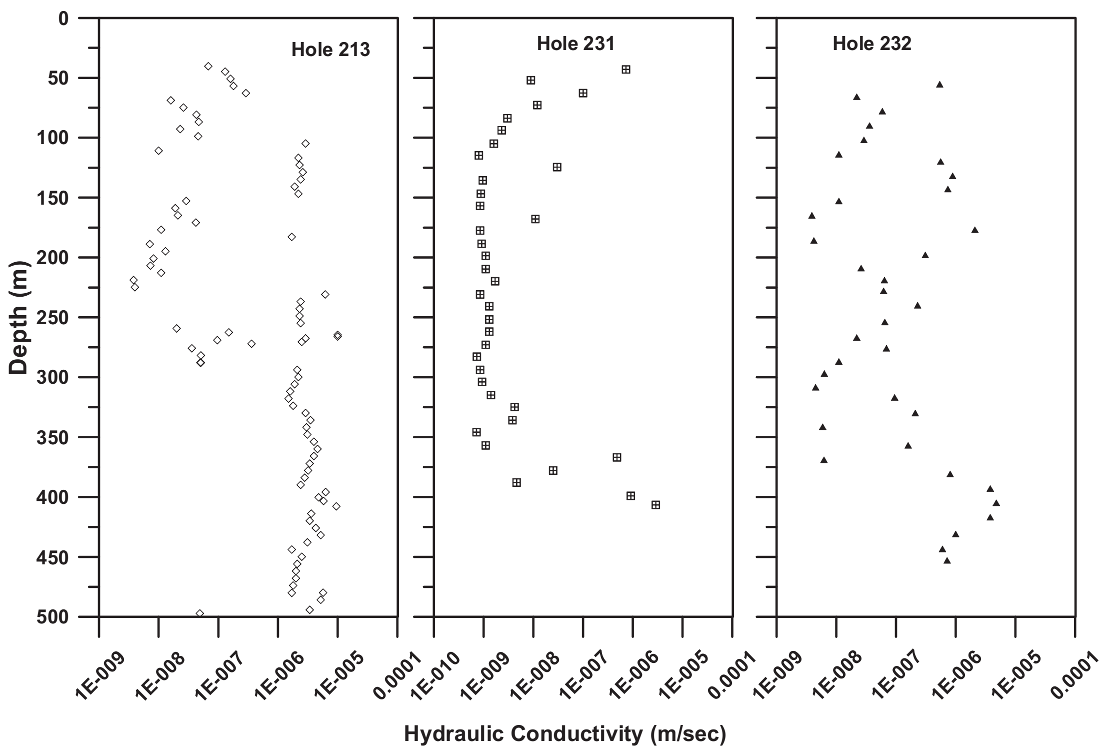

Figure 9 – Distribution of horizontal hydraulic conductivity with depth

Mine Inflows of 2006 & 2008

The primary risk associated with inflows at Cigar Lake Project is from mining activities, particularly:

- fall of ground that propagates to the overlying water bearing zones; and
- holes drilled from the basement rocks that connect with the water bearing zones.

The test mining program at Cigar Lake Project demonstrated the effectiveness of artificial freezing to control water inflows. However, in the development area, to the south side of the orebody, it was decided that ground conditions were satisfactory and it was an acceptable risk to develop a portion of the 465 production level in unfrozen ground (Cigar Lake, 2007). It was in this unfrozen section of development that the October 2006 water inflow occurred. The inflow event occurred on October 26th 2006 in the 465-944 drift on the 465 level of mine. The inflow event was hypothesized to be result of the of ground failure of a relatively thin (about 8 m) beam of weak, fractured rock exposed in the roof of the drift and being loaded from above by relatively high hydrostatic pressure (about 4.2 MPa). It is further hypothesized that the initial seepage through the incipient roof failure resulted in eroding the fracture infilling (clay and sand) resulting in further rock collapsing,. The roof gradually and then catastrophically chimneyed, up into and above the unconformity where it enabled water from the extensive sand aquifer above the unconformity to flow in at a rate limited only by the permeability and thickness of the overlying aquifer. It should be noted that another failure, but much less catastrophic, had occurred about 65 m away in the 465-743 XCN on October 6th 1999. In that case, the inflow rate was minimal – approximately 40 m^3/h. The collapsed "chimney" reportedly went up about 7 m, which is probably about 3 m below the unconformity and the source of a potential large volume of water. The cross-section of roof failures at 465-743 XCN and 465-944 DRE is shown in Figure 10.

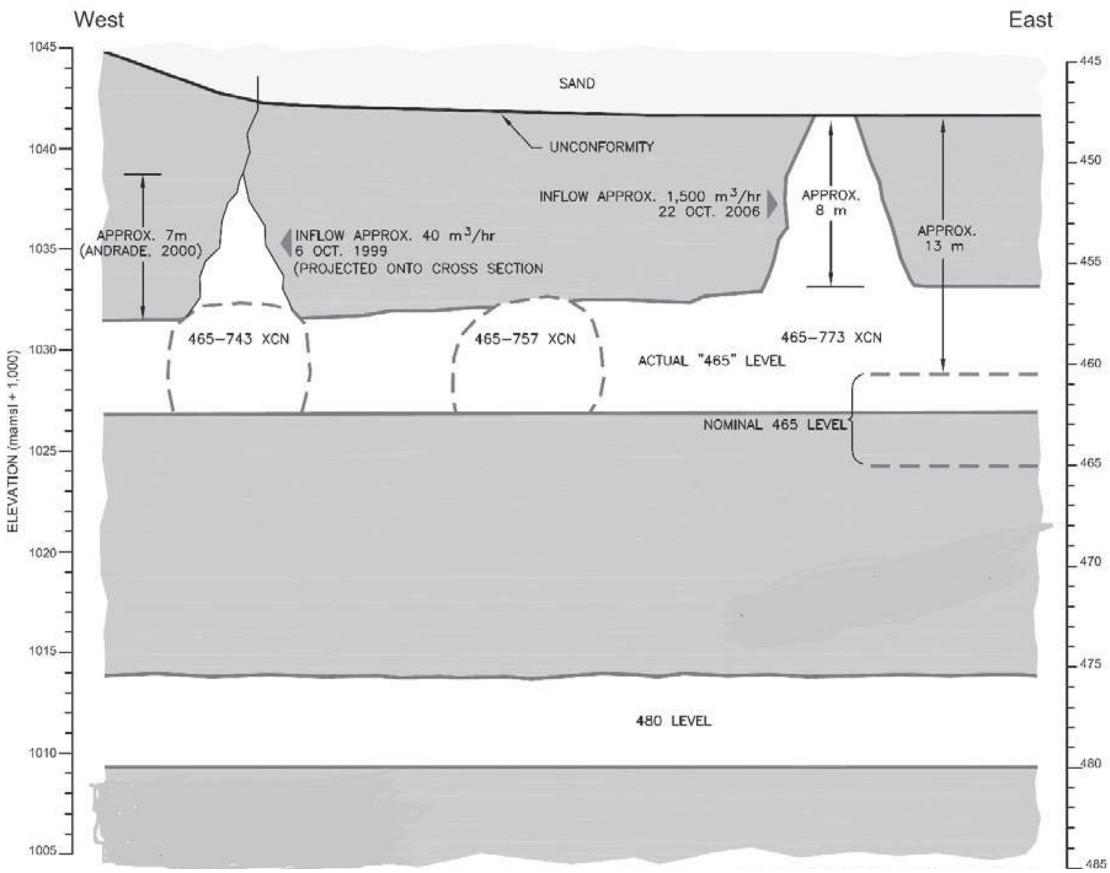

Figure 10 – cross-section of roof failures at 465-743 XCN and 465-944 DRE

<u>Mitigation</u>

During the first phase of remediation a number of holes were drilled 465 m down from the surface to the underground mine workings. Some of these holes were drilled to the source of water inflow and others to a nearby tunnel. A specially deigned concrete mix was poured into these two locations – one near the rock fall to seal off the inflow area and another in a nearby tunnel to provide reinforcement. A schematic of the remediation activities is shown in Figure 11. Four additional holes were drilled to 500m Level of the mine and were installed with borehole pumps to be used for dewatering the mine for the second phase of the remediation plan. This component of the remediation was a requisite for the dewatering strategy. This pumping system was be used to assist with mine dewatering, and continue to be available for use for emergency dewatering during the remainder of construction and operations.

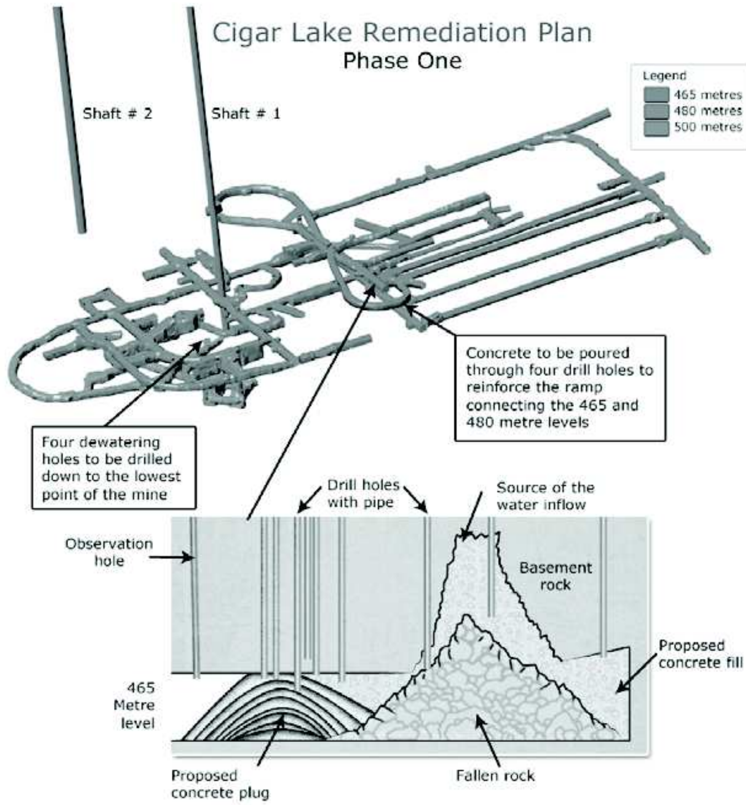

Figure 11 – schematic of the remediation activities

Following the installation of the concrete plug from surface a drawdown test was performed to test the effectiveness of the plug. The water level was pumped 100 m below ground surface in shaft # 1. The water level was held constant at this level and inflow to the mine workings under the imposed head was estimated by measuring the volume of water that needs to be pumped out to maintain the water level. The estimated amount of inflow to the mine workings was compared to a similar drawdown test done prior to the installation of the concrete plug. Figure 12 shows that results from pre and post grout drawdown tests. From this figure it can be seen that the installation of plug resulted in reduction of inflow to the mine workings by almost 89% under the imposed head. Following the drawdown test, a decision was made to dewater the mine in increments. The water from the mine workings was pumped out in increments with the water level held constant at for a period of time at predetermined intervals. The water level was held at these predetermined intervals to estimate the amount of inflow as the head on the workings was increased and to provide enough time for the excess pore pressure to dissipate. It should also be noted that the rate for dewatering the mine (the rate at which head was applied to mine workings/rate at which water level was lowered) was decided after a detailed geotechnical stability study by a third party geotechnical expert. Figure 12 also shows the inflow rates estimated during the dewatering attempt. The projected inflow to mine workings under fully dewatered conditions was slightly in excess of the pre inflow value.

The dewatering attempt was suspended on August 12, 2008 when the rate of the inflow to the mine significantly increased when the water level as held constant at 430 m below surface. . The location of this second inflow was later identified as a fissure located in a tunnel on the 420 m level. The 420 m level was developed many years ago to assess the practicality of developing a working level above the orebody. Further development on the 420 m level proved not to be feasible due to poor ground conditions. A concrete bulkhead was put in place and the remainder of the area was used for mine infrastructure and storage. On October 23, 2009, Cameco announced that the inflow on the 420 m level which forced suspension of dewatering on August 12, 2008 was sealed by remotely placing an inflatable seal between the shaft and the source of the inflow and subsequently backfilling and sealing the entire development behind the seal with concrete and grout. The 420 m level is not part of future mine plans and will be abandoned. Cameco plans to install a permanent bulkhead and fill the entire 420 m level with concrete backfill. The remediation for the August 2008 water inflow is shown in Figure 13.

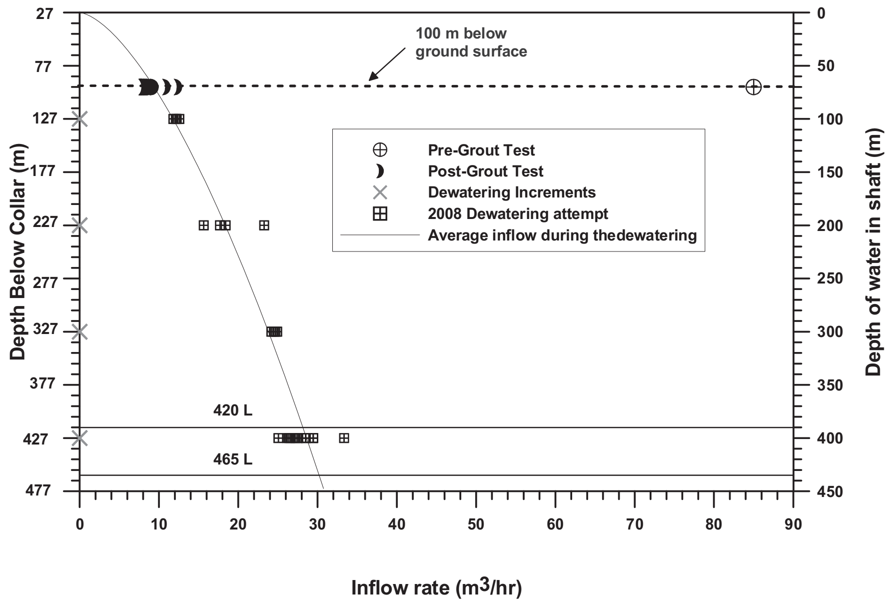

Figure 12 – Inflow rate to the mine working under various stages of remediation

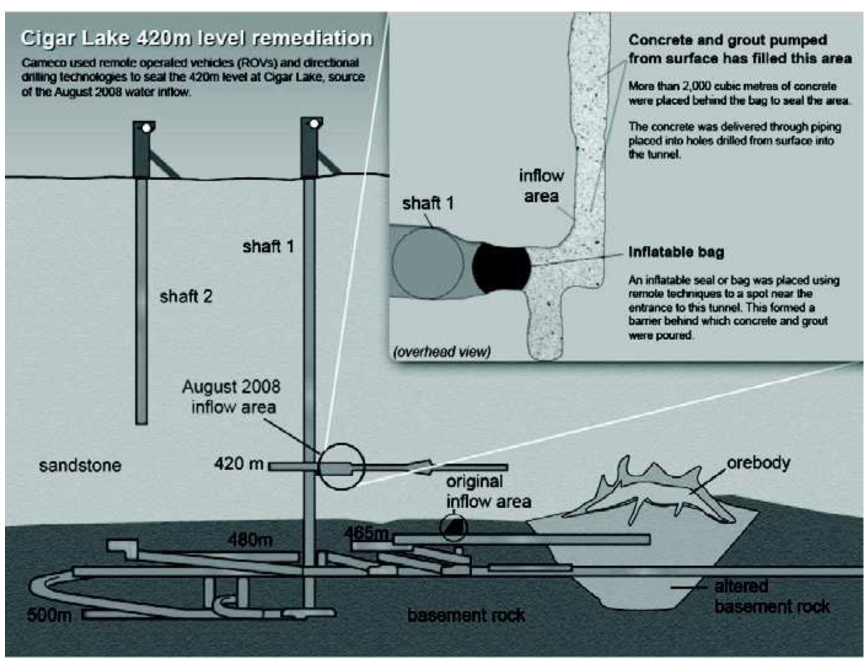

Figure 13 – Remediation strategy for inflow remediation on 420 level

Eagle Point Mine

Eagle Point Mine is part of Cameco’s Rabbit Lake Operation on the southeast shore of Collins Bay, which is an arm of Wollaston Lake. Eagle Point is located approximately 12 km northeast of the Rabbit Lake Pit (Figure 14). The Harrison peninsula, of which Eagle point is part, is generally low lying with swampy areas. Ground slopes gently from a low ridge near the centre of the peninsula towards both Collins Bay to the northwest and Ivison Bay to the southeast. The Eagle Point orebody subcrops under Collins Bay as well as under the shore area of Eagle Point. Depths of water over the orebody range from 0 m on shore to about 40 m. The total length of the deposit is approximately 5760 m, as measured along strike.

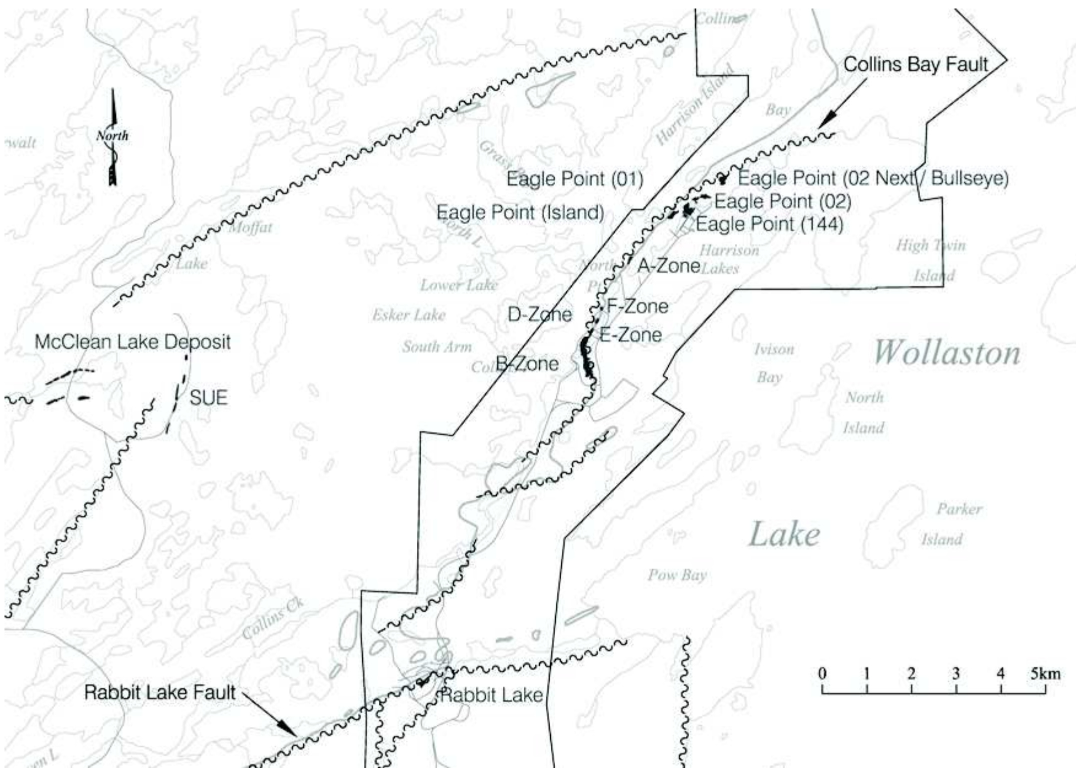

Figure 14 – Rabbit Lake Operation Site Map

<u>Hydrogeological Setting</u>

The formation can be divided into following hydrogeological units: (1) Surficial Sediments; (2) Bedrock Units; and (3) Structure; and (4) Mineralization and Alteration. Figure 15 shows the location of these units at the mine site. A brief description of these units is provided in the following paragraphs.

Total thickness of the surficial sediments in the Eagle Point area is estimated to range between about 6 and 14m thick. Average hydraulic conductivity of the total thickness of surficial sediments is estimated to range between 10^{-6} to 10^{-4} m/s. Surficial sediment can be further classified into following units: i) Soft Lake Sediments, ii) Upper Deglacial Sediments, iii) Lower Till, iv) Lower Gravel.

The soft lake sediment is typically an organic silt layer of only 1.0 m in thickness although it thickens to about three meters towards the centre of the lake. The hydraulic conductivity of this unit is fairly low and is estimated to range between 10^{-6} to 10^{-5} m/s. Upper Deglacial Sediments are possibly a reworked, sorted till and typically range from 3 to 5 m in thickness. They are composed of glacio-fluvial sand and gravel, underlying lacustrine silt and clay. Where the lacustrine material is present, the hydraulic conductivity of this unit is estimated to be 10^{-7} to 10^{-6} m/s. In areas where the silt and clay is absent, hydraulic conductivities could be as high as about 10^{-4} m/s. Lower Till unit ranges from a meter thickness to about 6-7 meters thick in the Eagle Point area. It is typically compact pebbly gravel with a silt/sand matrix and some silty clay-rich sections. The hydraulic conductivity of these sediments is estimated to be in the range from about 10^{-7} to 10^{-4} m/s. Lower gravel exists offshore to the southwest of the Eagle Point orebody. It has been deposited in a topographic low associated with the Collins Bay Fault. The hydraulic conductivity of this material is estimated to be in the range of 10^{-6} to 10^{-4} m/s.

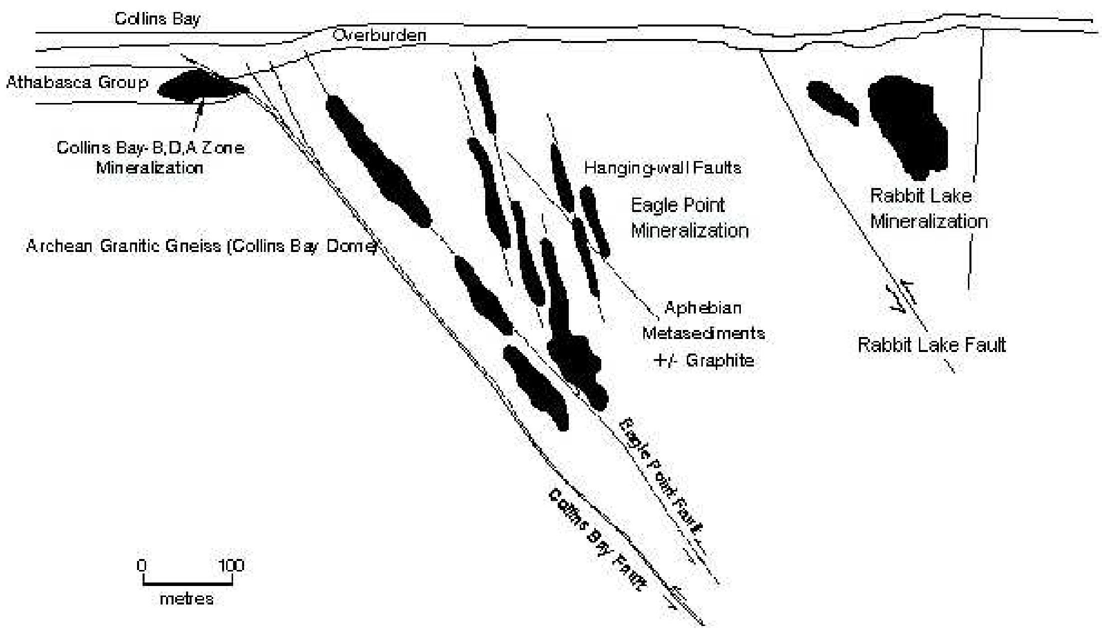

Figure 15 - Cross-section of different deposits at Rabbit Lake

The bedrock sequence at the Eagle Point Mine consists of a granitoid gneiss overlain by the Wollaston group pargneisses. Athabasca Fracturing in these units ranges from slight, to highly fractured in the fault zones. Alteration associated with fault and fracture zones is restricted to narrow bands around and along these structures. This indicates that, at the time the alteration occurred, permeability of the rock mass adjacent to these structures was sufficiently low to inhibit the passage of the solutions that caused the alteration. Alteration products, typically chlorite and illite in unmineralized zones, and chlorite, illite and hematite in mineralized areas, have probably substantially reduced the permeability of the faults since before the alteration occurred. A reasonable range of hydraulic conductivity of the shallower Aphebian rocks is 10^{-8} to 10^{-5} m/s and for deeper rocks to be 10^{-8} to 10^{-7} m/s.

The Athabasca sandstone is typically a gently dipping, thinly bedded sandstone and pebble conglomerate. Near fault zones, the sandstone is typically friable and fractured. The sandstone is quite permeable, having a hydraulic conductivity ranging between 10^{-8} and 10^{-5} m/s. However, as sandstone is only present on the downthrown side of the Collins Bay Fault, it is not present above the orebody. Thus, it is of little importance in terms of its effect on the inflows to the eagle point mine.

Possible Inflow Scenarios for Eagle Point Mine

The Eagle Point orebody is hosted in basement rock that allows longhole stope mining. The mining depth is such that hydrostatic water pressure is more easily handled in the event of an inflow when compared to McArthur River Operation and Cigar Lake Project. However part of the mining areas are under Wollaston Lake, so under a catastrophic failure of the crown pillar (rock left between the uppermost levels and the bottom of Wollaston Lake) the inflows could vastly exceed any pumping system capacity. The mine de-watering system requires the mine water to be pumped to the mill 15 km away for treatment before release. There is large volume storage capacity of more than 150,000 m^3 underground in mined out areas.

The most likely inflow scenario for Eagle Point Mine would be interception of an ungrouted (or poorly grouted) exploration borehole beneath Collins Bay. The risk for such a scenario exists during future mining activities in the 82-122 Levels at the 02 Next area and the 90-120 Levels at the 03 Zone, both beneath Collins Bay, and the 80-125 Levels in Block 1 of the 144 South Zone, immediately east of Collins Bay (Figure 16). Detailed risk assessments are done prior to mine development in these areas.

A direct connection to Collins Bay via geologic structure can also be considered as a possible inflow scenario. There is no evidence that this has occurred to date. The correct course of action is to maintain current probe drilling practices ahead of mine development activities.

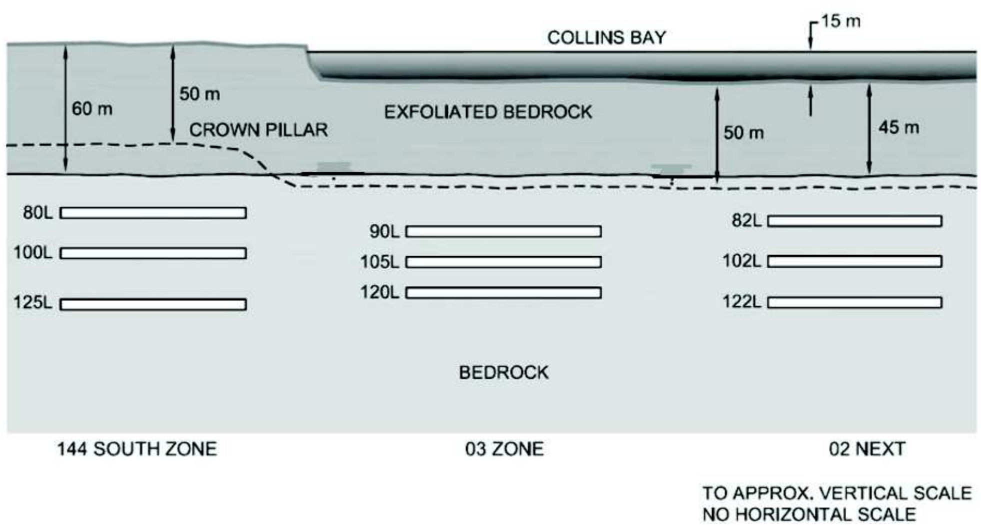

Figure 16 – Existing and proposed development on upper parts of Eagle Point Mine

Mine Inflow of 2007

An unplanned inflow began on November 26, 2007 at the Eagle Point Mine. Inflow location was from four discrete areas over an approximate 8 m strike length and 4 m height from the blasted footwall of the 105-533 stope just below the 90 Level in the "03 Zone". Initially the inflow rate was estimated to be up to 45 m^3/h. Actual flow rate was later confirmed to be approximately 110 m^3/h by building a weir underground. The origin of the inflow was thought to be the surface diamond drill hole EP-234. This hole collared at lake bottom with a structural connection from the hole serving as a conduit to the 90L overcut development at Long hole Stoping of 105-533 Stope in the "03 Zone". The hole was drilled as part of the surface diamond drilling program from 1980s. In those early years some of the holes drilled from the lake were grouted, however others were left un-grouted. Eagle Point Mine geology department maintains a database of all the surface holes in which their status, whether grouted or ungrouted is also mentioned.

A theoretical calculation using flow of fluid through a pipe employing Darcy-Weisbach Formula with friction factor similar to that of concrete pipe produced inflow rate similar to the one measured at the weir. Similar calculations are done routinely in Karst hydrogeology for conduit flows [1]. A Remotely Operated Vehicle (ROV) was used to search the lake bottom and was successful in positively identifying the hole responsible for the inflow

<u>Remediation</u>

Although the source of this inflow was also due to a borehole, the structural connection that created a conduit between the borehole and the footwall of the open stope was not consistent with previous experience. A conservative approach was used in developing the action plan to stop the inflow to ensure that crown pillar stability was not jeopardized. Only a small percentage of the storage (5%) was used in the month it took to shut off inflow. The specific exploration hole that contributed to the inflow was found by the use of a ROV scanning the lake bottom. The hole was plugged off from top of the ice (in the lake) by first lowering casing and then a packer, followed by grouting off the hole with cement. In addition a conservative step was taken by operation staff to pour four monolithic bulkheads underground to seal the inflow area from the rest of mine.

MITIGATION AND LESSONS LEARNED

To facilitate corrective actions, each of the major mine inflows were investigated by the TapRooT® method. In the case of the McArthur River 2003 and Cigar Lake 2006 inflows the TapRooT® investigation was carried out by independent third party. TapRooT® is a well established investigative tool which systematically and objectively examines an incident/accident. The objective of the TapRooT® investigation is to determine the root causal factors of an incident. This is achieved by developing a flow chart of events which have occurred; identifying conditions which have significant impact on events enabling the identification of causal factors. TapRooT® closely focuses on the performance of systems which are in place, including the systems which were in place but did not operate. The TapRooT® investigation concludes with development, evaluation and implementation of Corrective Actions. The seven steps in a TapRooT® investigation are shown in Figure 17.

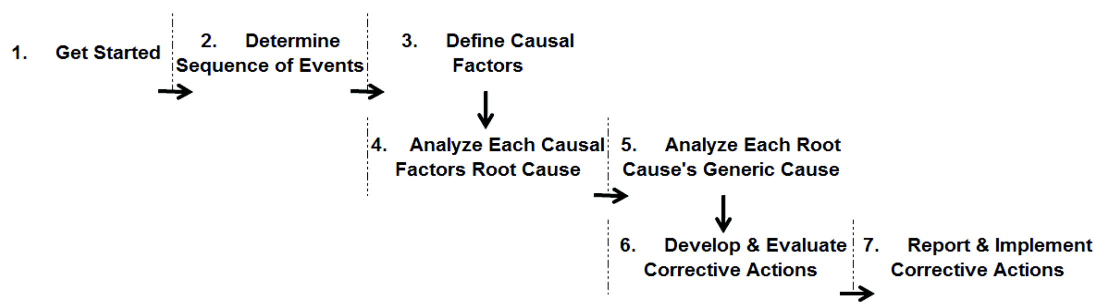

Figure 17 – Various steps in the TapRooT® investigation

Table 1 summarizes the select causal factors identified in the formal TapRooT® investigations for the inflows. A brief summary of some of the lessons learned from these inflows. It should be noted that in root-cause analysis it is typical to determine design, execution, and human factors in the investigation of serious incidences.

Table 1 – Causal factors identified in the formal TapRooT® investigations

	McArthur River 2003 Mine Inflow*	Cigar Lake 2006 Mine Inflow**	Eagle Point Mine 2007 Mine Inflow***
Design	Quantifying how much dewatering capacity needed	Better flood-related emergency preparedness	Reliability of drill hole grout status
	Alternative excavation and support methods for high risk development	Formal standards for mine development	Risk Assessment Omission (grout status of surface drill holes)
		Improved general risk awareness	
Execution	Timely application of ground support for high risk development	Formal standards for mine development	Change Management (of risk assessment)
	Need for comprehensive geotechnical assessment for design and monitoring	Critical equipment maintenance	
Human Factors	Roles and responsibility clarification	Roles and responsibility clarification	Oversight (no one responsible for overall site water balance)
	* 6 CFs in total	** 11 CFs in total	*** 4 CFs in total

In the past, the underlying premise has been that water bearing sandstone would not be encountered in mine development; therefore its larger inflow potential did not represent valid design criteria. This has been proved to be an incorrect assumption as the development in the basement at Cigar Lake in 2006 and McArthur River in 2003 as ground failures caved upwards into the sandstone. Quantifying how much dewatering capacity is needed becomes function of maximum inflow rate if a direct connection with the sandstone develops. New ground water modeling was commissioned for both McArthur River and Cigar Lake to establish new maximum uncontrolled inflow rates. Studies were conducted to better understand the hydrogeology of the areas of inflow and inflow volumes have been back calculated to calibrate models and assess dewatering requirements. Contingency pumping has been made available, well beyond maximum encountered inflow, plus the water storage available underground has also been quantified. Cameco has also developed a corporate standard on pre-development mine inflow and water handling assessments The purpose of this corporate standard is to describe the minimum requirements for pumping, storage and treatment of mine water during shaft sinking or underground development and operation. The corporate standard has been implemented corporate wide and sites have been audited by a corporate mine hydrogeologist and external auditors. The corporate standard has an explicit requirement for each site to establish a maximum uncontrolled inflow rate, have suitable dewatering capacity with contingency available before development to proceed underground. The corporate standard also requires sites to carry out formal risk assessment before developing in high risk area (close to unconformity or areas with surface diamond drill holes) and have mitigation plans in place.

The ground control standards and development standards when mining in the proximity to the unconformity have been reviewed and improved. Assessments for high risk development include: evaluation of the risk, modeling of the area for ground support requirements and third party review of ground control methods and design parameters. In high risk areas, methods to mitigate water under pressure such as ground freezing, grout covers, or drainage are taken to reduce the potential risk of an inflow. Ground freezing and probe and grout covers are still used to create barriers to water inflow around production areas with the resulting freeze walls tied into dry basement rock. In addition, ground control audits from corporate staff and external consultants are conducted. To get a better definition of geology and geotechnical hazards of higher-risk development areas, procedures are in place for documenting results and recommendations to proceed from probe and grout campaigns.

As part of the comprehensive geotechnical assessment & monitoring, the current practice for high risk ground includes: controlled advance rates, time requirements for ground support installation, geotechnical mapping with each advance, ground control inspections for each advance, communication of unusual occurrences, surveying of every second advance and maintenance of closure stations with development.

Although potential inflows originating from diamond drill holes are commonly prevented by grouting, anecdotal evidence from industry suggests that the grouting integrity of surface exploration holes is always questionable. Potential causes for inadequately grouted holes include: human error or omission; interference from rock structures during grouting; or environmental degradation. Guidelines have been put in place for assessing the risk from diamond drill holes and methods for mitigating the risk.

In terms of human factors, roles and responsibility clarification has been a common identifier in most of the TapRooT® investigations. As part of a 2006 corrective actions, organizational structure has been analyzed in detail. Specific supervisory roles have been clearly defined to enhance communication and management of the mining contractor. Various additional technical roles such as technical superintendent, ground control engineer and corporate rock mechanics engineer positions have been added.

In order to improve risk awareness for underground employees and contractors, a water inflow awareness training program was developed. This program consists of a series of five modules that imparts skills and knowledge required to understand: the (1) unique characteristics associated with the uranium deposits of the Athabasca Basin; the (2) events leading up to the 2003 McArthur River and 2006 Cigar Lake inflows and lessons learned; (3) basic geology and the affects of water pressure; (4) hazards and risks related to inflows; and, (5) ground control, support and warning signs. Training commenced early in 2008 and by year-end a total of 680 employees and contractors at McArthur River, Cigar Lake and Rabbit Lake had been through the program and received the necessary qualifications.

REFERENCES

1. W.B.White and E.L.White "Ground water flux distribution between matrix, fractures, and conduits: constraints on modeling", Speleogenesis and Evolution of Karst Aquifers, Vol. 3, No. 2, 3 (2), 2005, 1-6.
2. B. W. Jamieson and S. E. Frost, "The McArthur River Project: High Grade Uranium Mining", Uranium Institute Annual Symposium 1997, 1-14.
3. Cameco (2007) Cameco Corporation Cigar Lake Project, Northern Saskatchewan, Canada NI 43-101 Technical Report (http://www.cameco.com/common/pdf/investors/financial_reporting/technical_report/Cameco_-_Cigar_Lake_Technical_Report_2010.pdf)
4. Winberg, A. and Stevenson, D.R.. Hydrogeological modelling. In: J.J. Cramer and J.A.T. Smellie (Eds.), Final Report of the AECL/SKB Cigar Lake Analog Study (pp. 104-142). AECL Tech. Rep. (AECL-10851) 1994, Pinawa, and SKB Tech. Rep. (TR 94-04), Stockholm.
5. Cameco (2008), Cameco corporation McArthur River Operation, Northern Saskatchewan , Canada NI 43-101 Technical Report (http://www.cameco.com/common/pdf/investors/financial_reporting/technical_report/McArthur_Technical_Report_2009.pdf)
6. Chamber of Mines of South Africa, Construction of Underground Plugs and Bulkhead Doors Using Grout Intrusion Concrete, Code of Practice, February 1983.

7. T. Yameogo and J. Hatley, "Overcoming the Geotechnical Challenges of Drift Development at the Unconformity in the Athabasca Basin", Uranium 2010, 3rd International Conference on Uranium.