Inflows in uranium mines of northern Saskatchewan: Risks and mitigation

R. Bashir
Golder Associates, Saskatoon, Saskatchewan and Department of Civil & Geological Engineering, University of Saskatchewan, Saskatoon, Saskatchewan

J. F. Hatley
Cameco Corporation, Saskatoon, Saskatchewan

ABSTRACT Northern Saskatchewan boasts some of the world's largest known high-grade uranium deposits. The successful mining of these deposits cannot be accomplished without overcoming technical challenges, however, which include high grade of the uranium ore, specialized mining methods to deal with groundwater at high pressures, and poor ground conditions. This paper discusses the various mechanisms of inflows at three of Cameco Corporation's sites in northern Saskatchewan. The risk of inflows is quantified in terms of unique challenges due to hydrogeological conditions, rock mass integrity, and uncertainty in geological conditions. Strategies in the case of an inflow are also briefly described.

KEYWORDS Uranium, Inflow, Athabasca Basin, Remediation, Hydrogeology, Rock mechanics, Cameco, McArthur River, Cigar Lake, Rabbit Lake

RÉSUMÉ Le Nord de la Saskatchewan peut se vanter d'avoir les plus gros gisements d'uranium à haute teneur au monde. Toutefois, il n'est pas possible d'exploiter ces gisements de manière efficace sans surmonter des défis techniques, lesquels comprennent la haute teneur du minerai d'uranium, les méthodes spécialisées d'extraction qui tiennent compte de l'eau souterraine à pression élevée et les mauvaises conditions du sol. Le présent article discute les divers mécanismes de venue d'eau à trois sites de la Cameco Corporation situés dans le Nord de la Saskatchewan. Le risque des venues d'eau est quantifié en termes des défis uniques dus aux conditions hydrogéologiques, à l'intégrité de la masse rocheuse et à l'incertitude des conditions géologiques. Des stratégies d'atténuation en cas de venue d'eau sont aussi brièvement décrites.

MOTS CLÉS uranium, venue d'eau, bassin de l'Athabasca, mesures correctives, hydrogéologie, mécanique des roches, Cameco, rivière McArthur, lac Cigar, lac Rabbit

INTRODUCTION

The Province of Saskatchewan is home to the world's largest producers of fertilizer (KCl) and uranium concentrate (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 the Athabasca Basin are of hydrothermal origin, located along ancient fractures at the unconformity between the Archean basement and the overlying sandstones of the Proterozoic (Figure 1). Saskatchewan also has an abundance of surface water and groundwater. Most of the potash and underground uranium mines have experienced significant unplanned water inflows at 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 groundwater regimes. This paper discusses the recent lessons learned by

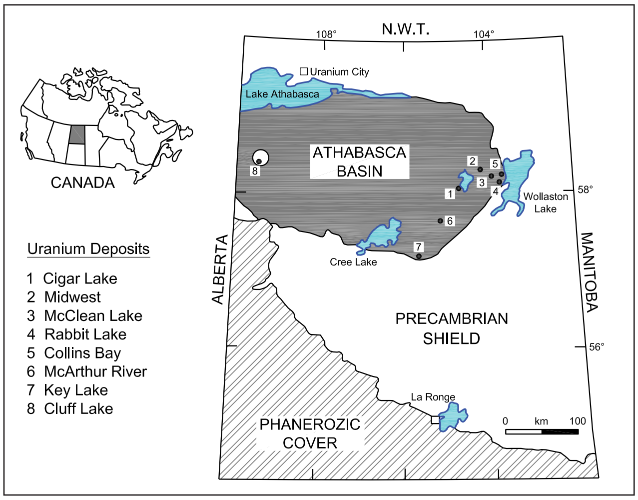

Figure 1. Uranium deposits of the Athabasca Basin, northern Saskatchewan, Canada.

the mining operations staff of the Cameco Corporation in the challenging Athabasca Basin, located in northern Saskatchewan.

TYPES OF MINE INFLOWS

Two of the primary challenges in mining the unconformity-type deposits in the Athabasca Basin are control of groundwater and 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 is present in significant quantities, with pressures up to 4–5 MPa, depending on depth. These areas can produce significant flows if intersected by mine development in the basement rock. 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, and 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 that have a substantial amount of water at 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 accurately quantified. In addition to the holes drilled underground for geotechnical characterization, there is also the potential to encounter surface holes during underground development. These surface exploration holes may or may not be grouted and, with their passage through approximately 400 m of water-saturated sandstone, can be a significant source of water. Moreover, in some instances, exploration holes drilled from the bottom of a lake 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 2003 McArthur River operation 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 cementitious 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 an 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. The 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.

Mining 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, initial mining studies determined that non-entry mining methods would be required to mine the deposit. The raisebore 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 (Bronkhorst, Edwards, Mainville, Murdock, & Yesnik, 2008).

Hydrogeological setting

The upper bedrock at McArthur River consists of 480–560 m of Athabasca Group sandstones, which unconformably overlie crystalline Archean and Aphebian basement rocks. The mineralization at the McArthur River operation is associated with a major thrust fault zone known as the P2 fault and the majority of the mineralization 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 hydrogeological investigations. Stratigraphically from highest to lowest, they are: overburden, sandstone, fanglomerate (and/or conglomerate) with a basal paleoweathered zone, unconformity, mineralized zone, and basement rocks (Figure 2).

The overburden is a relatively thin unit, averaging 10–15 m thick with a maximum thickness of approximately 50 m in the drumlin to the east of the mine (Figure 2). Its hydraulic conductivity is in the range of 0.1–1.0 m/day. The underlying Athabasca sandstone, which is approximately 550 m thick in the mine area, has been locally differentiated

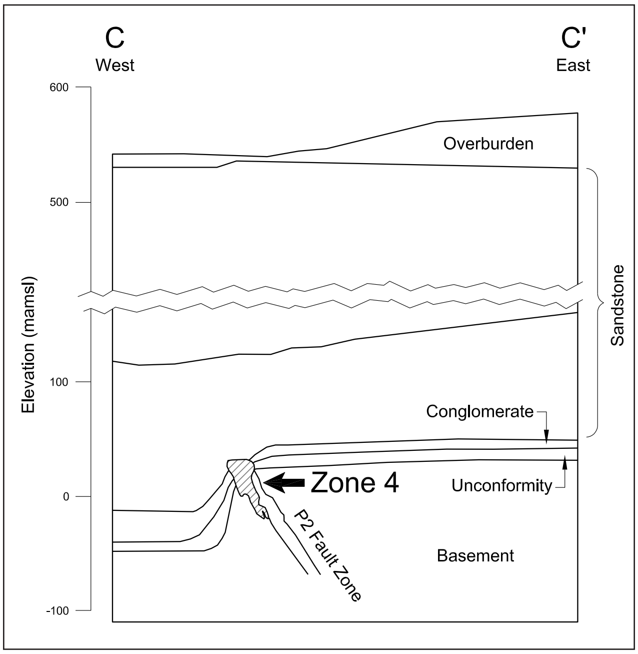

Figure 2. Cross section of stratigraphy and orebody at the McArthur River operation.

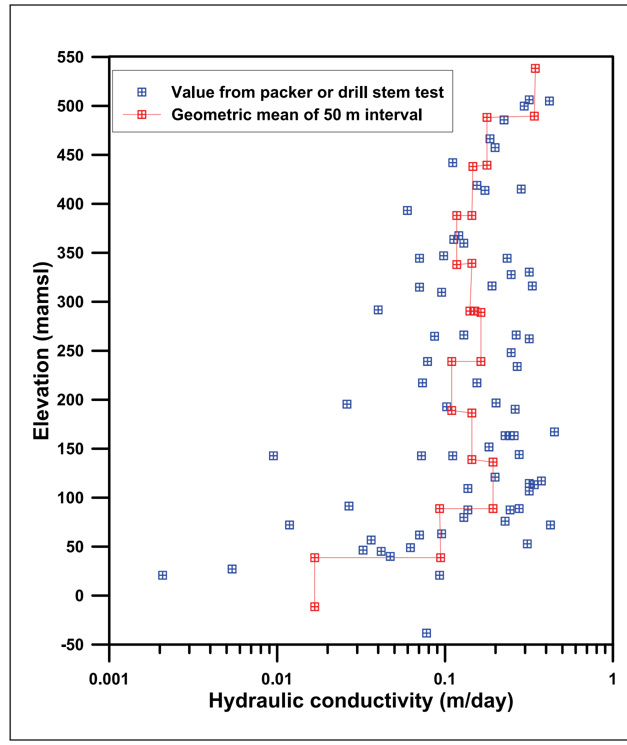

Figure 3. Hydraulic conductivity versus depth in sandstone at the McArthur River operation.

into four subunits: 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 desilicified). Fractures, however, induce a bulk hydraulic conductivity in the range of 0.02–0.5 m/day. Figure 3 shows the horizontal hydraulic conductivity (K_h), with the 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, where it appears to be approximately one order of magnitude less permeable than in the overlying sandstone. The fanglomerate and paleoweathered zone is a relatively thin unit with a thickness ranging from approximately 10 to 30 m. Its hydraulic conductivity values are in the range of 0.002–0.08 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 0.002–0.05 m/day.

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 1,050–1,100 m^3/h, including the existing background inflow of 225 m^3/h. The installed pumping capacity at the time of inflow ranged from 450 to 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 the addition of a number of low-head, 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 during this event.

The 7320 East Freeze drift on the 510 m sublevel was being excavated in proximity to the unconformity and the P2 fault in preparation for freeze drilling of zone 2. The drift was to be advanced below the unconformity and experience indicated that the 8–10 m 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 be a factor. Unfortunately, this assumption proved incorrect. Prior to the drift development, the McArthur River operation spent three months drilling 28 holes, each averaging approximately 35 m in depth, into the area and the greatest flow 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 of probe drilling ahead of development, and hundreds of probe and grout drilling covers at the McArthur River operation have proven successful. Unfortunately, this drilling does not always find all the water-bearing structures; however, it is the best method to predict what lies ahead.

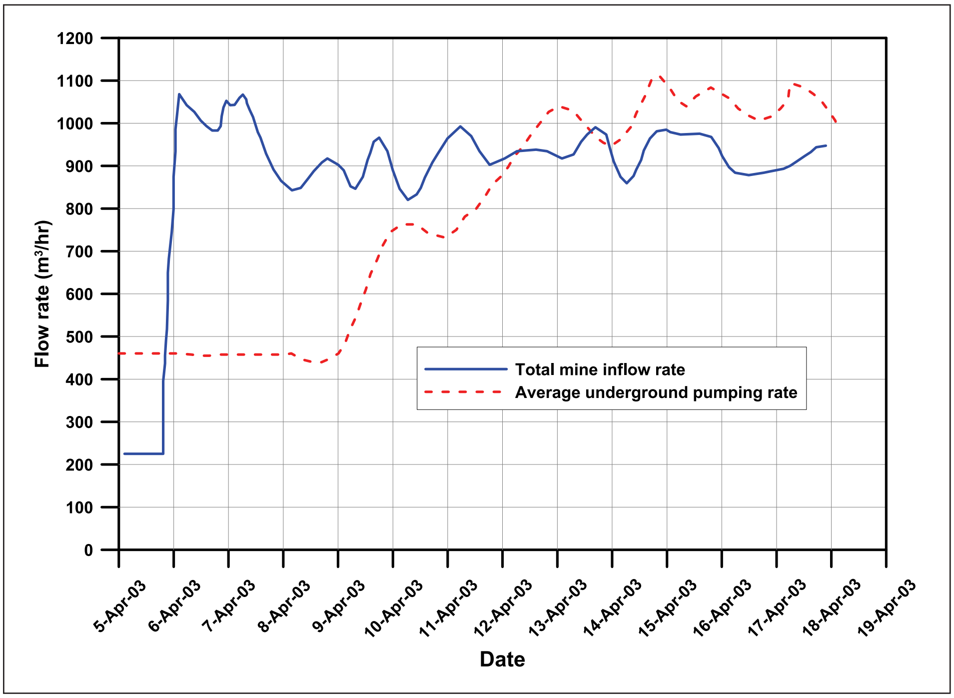

Figure 4. Total mine inflow rate and average underground pumping rate during the 2003 inflow at the McArthur River operation.

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, an Armtec arch was installed 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. Normal ground control support continued with 2.4 m long Dywidag rockbolts, 1.8 m long split set bolts and screening, 75 mm primary and 50 mm secondary shotcrete, and three layers of IBO Ankor Bolts spiling. This support system had resulted in no significant issues elsewhere in the mine, including drifts that were in proximity to the unconformity and the P2 fault. Cable bolting was not used 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 too 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 and stress conditions on the steel Armtec arch, causing differential loading and stress concentration at the 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 the back of the drift remained the same, the general hydraulic gradient remained constant for the drift, but locally in the new fracture set water pressure of 5 MPa was 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 expanded to relieve pressure to reach a more stable state. The continued movement in the back and walls would have resulted in a small amount of inflow with instantaneous release of high pressure. Initially, the flow would have been small, but it would have continued as more joints unraveled and key blocks fell 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 because the clay material would have washed out of the fractures in the rock above. As more material washed out, 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, 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 m strike length, and resulting in development of a maximum inflow rate.

Remediation

All inflows are unique and require careful planning to remediate. The special 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 (Figure 5).

A number of choices and decisions must be made by professionals, including senior mine staff and experienced consultants, at the design and the implementation stages of bulkheads. For McArthur River, structural bulkheads or monolithic bulkheads were suitable. Structural bulkheads are generally 2 m thick, notched into the surrounding rock, and contain inner rebar mats on the dry face to counter bending forces. Monolithic bulkheads are generally 8 m 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.

Due to the volume of water in the 2003 inflow at McArthur River, a culvert was used initially to direct the flow away so flanged piping could be installed (Figure 6). Two bulkhead walls were created, and 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 resealed 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 began over many months. This program involved detailed water pressure and geotechnical stability monitoring of the bulkhead and the surrounding tunnels. Approximately 1,000,000 kg of cement was used in cementitious grouting around the inflow cavity, into the immediate area at the unconformity, and into the sandstone above. This area, also referred to as zone 2 panel 5, was successfully redeveloped in 2009 using ground freezing, careful excavation, and applicable controls, as described in the Lessons Learned section of this paper.

CIGAR LAKE MINE

The Cigar Lake project is an underground uranium mine that is under

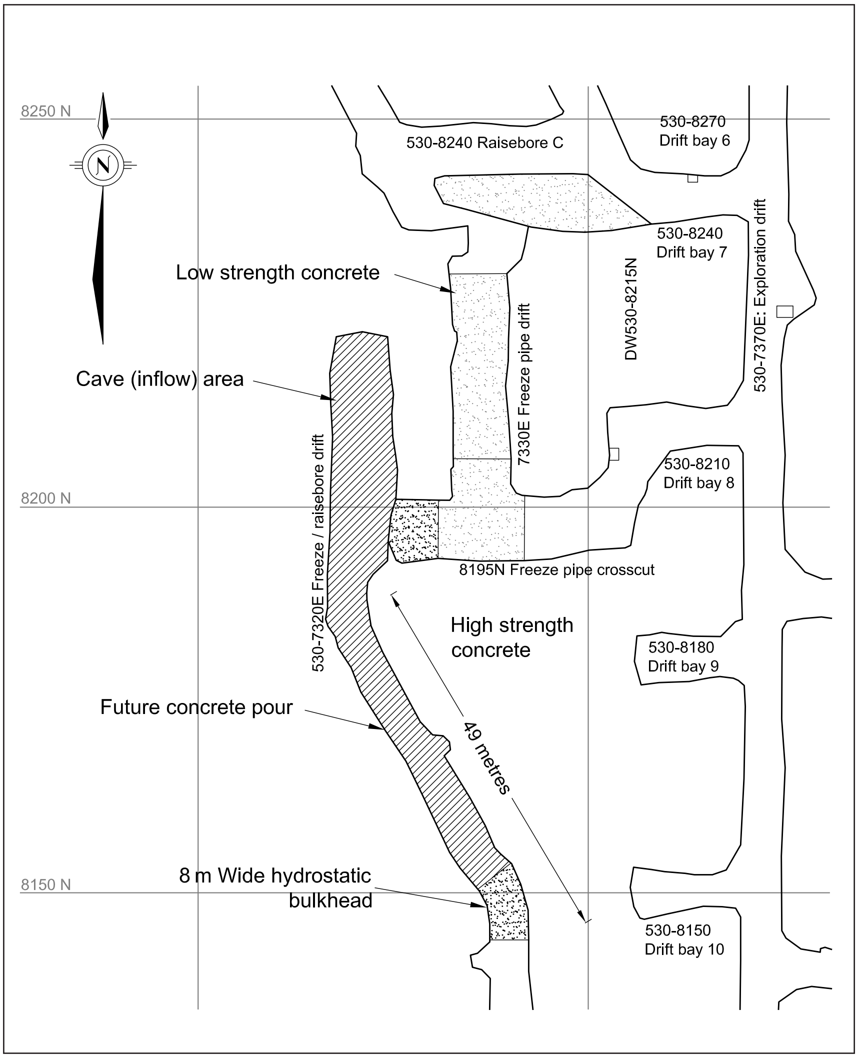

Figure 5. McArthur River 2003 inflow hydrostatic bulkhead and concrete fill locations.

Figure 6. Initial timber bulkhead wall, McArthur River.

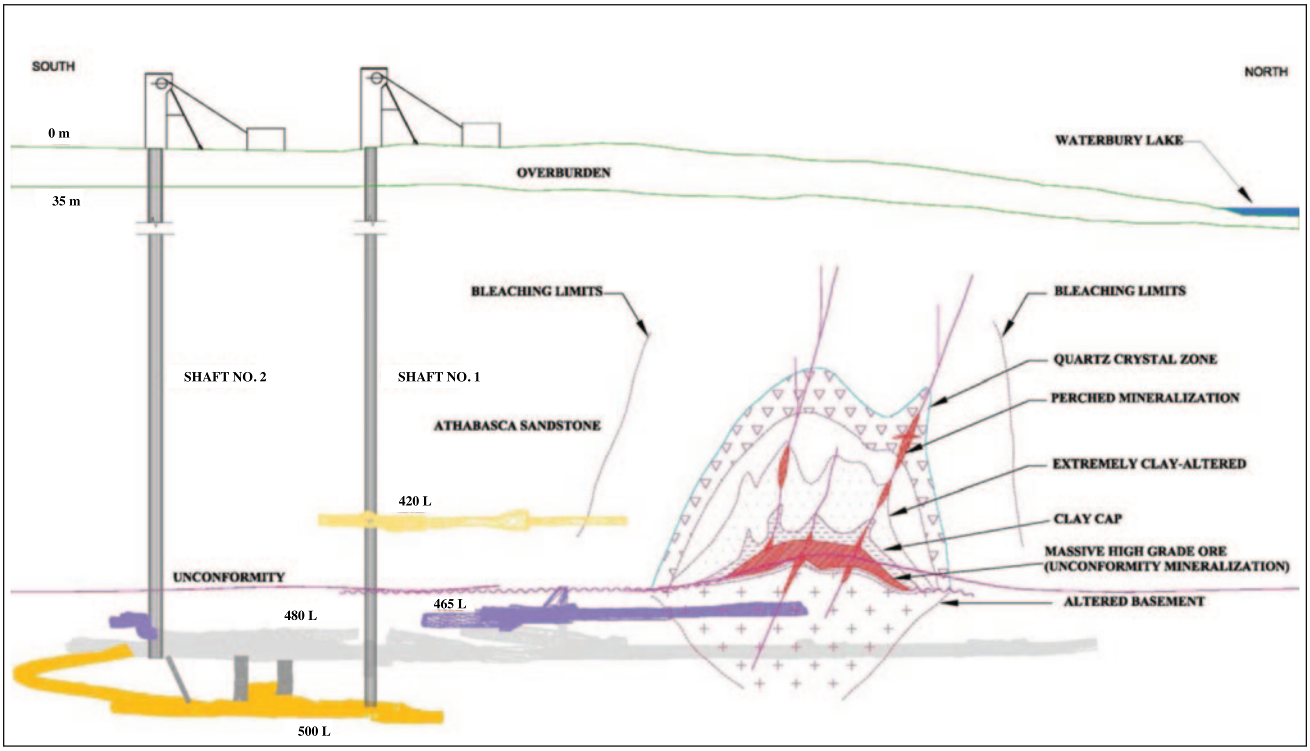

Figure 7. Schematic geological cross-section of the Cigar Lake orebody and existing underground development.

construction in northern Saskatchewan, 70 km southwest of McClean Lake and 660 km north of Saskatoon. The Cigar Lake orebody is situated approximately 430 m below the surface at the unconformity between metamorphic basement rocks and flat-lying sandstone. The deposit is characterized by a series of geochemical alteration haloes arranged geometrically 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 7 shows the schematic geological cross-section of the Cigar Lake orebody and existing underground development.

Some of the major technical factors influencing the selection of a mining method include ground stability, control of groundwater, radiation exposure, and ore handling and storage. The deposit is 20–100 m wide and approximately 2150 m long. It is crescent shaped in cross-section and averages 6 m thick, with a maximum thickness of 15 m. 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, 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 hydrogeological setting of the Cigar Lake deposit can be found in Winberg and Stevenson (1994). They theorized 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;
- an intermediate flow regime with water recharging in the upstream end, partly discharging into Waterbury Lake and partly feeding the lower sandstone; and
- 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 semi-regional groundwater flows primarily within the lower sandstone; final discharge for all regimes is Waterbury Lake, as suggested by tracking of water particles from the local model into the regional model. Groundwater flow at the depth of mineralization is horizontal, from south to north, with an average hydraulic gradient of approximately 1 %.

The sandstone in the Cigar Lake mine area is 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 for this increase is that it is a result of the fluids that moved through it and above the unconformity as part of the ore-forming process. Figure 8 shows the horizontal hydraulic conductivity (K_h) values derived from the packer tests in the three coreholes drilled as shaft pilot holes. Assuming the groundwater system is essentially hydrostatic (i.e., there

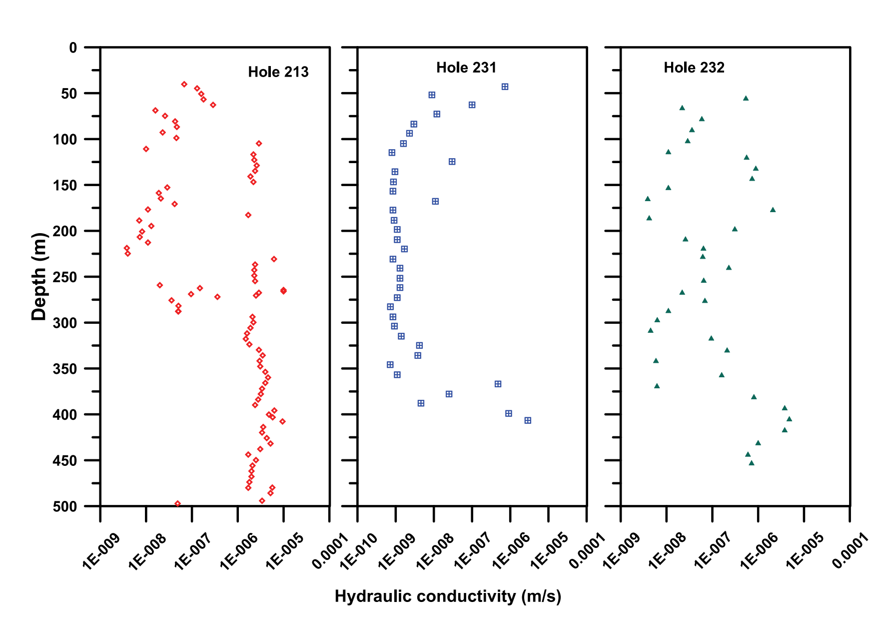

Figure 8. Distribution of horizontal hydraulic conductivity with depth, Cigar Lake.

are no significant differences in hydraulic heads with depth) and the phreatic surface above the site is at a depth of approximately 20–30 m, the pore pressure at the unconformity at a depth of 448 m is approximately 4.2 MPa.

Mine inflows of 2006 and 2008

The primary risk associated with inflows at the 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 water-bearing zones.

The test mining program at the Cigar Lake project demonstrated the effectiveness of artificial freezing to control water inflows. In the development area, to the south side of the orebody, however, 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 (Bishop, Goddard, Mainville, & Schwartz, 2007). It was in this unfrozen section of development, in the 465–944 drift on the 465 level, where a water inflow event occurred on October 26, 2006. The inflow event was hypothesized to be the result of the failure of a relatively thin (approximately 8 m) beam of weak, fractured rock exposed in the roof of the drift and loaded from above by relatively high hydrostatic pressure (approximately 4.2 MPa). It is further hypothesized that the initial seepage through the incipient roof failure resulted in erosion of the fracture infilling (clay and sand), resulting in further rock collapse. 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 much less catastrophic failure had occurred approximately 65 m away in the 465–743 crosscut north (XCN) on October 6, 1999. In that case, the inflow rate was minimal— approximately 40 m^3/h. The collapsed chimney reportedly went up approximately 7 m, which is probably approximately 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 drift east (DRE) is shown in Figure 9.

Remediation

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 designed 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 10. Four additional holes were drilled to the 500 m level of the mine and were installed with borehole pumps to be used for dewatering the mine during the second phase of remediation. This component of the remediation was a prerequisite for the dewatering strategy. This pumping system was to be used to assist with mine dewatering, and continue to be available for use for emergency dewatering during the remainder of construction and operations.

Following the installation of the concrete plug from the 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 needed 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 11 shows the results from pre- and post-grout drawdown tests. From this figure, it can be seen that the installation of the 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 for a period of time at predetermined intervals. The water level was held at these predetermined intervals to estimate the amount of inflow while 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 or the rate at which the water level was lowered) was determined 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 inflow to the mine significantly increased while the water level was 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 more than 20 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 was to be abandoned. Cameco installed a permanent bulkhead and filled the entire 420 m level with concrete backfill. The remediation for the August 2008 water inflow is shown in Figure 12.

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 13). The Harrison Peninsula, of which Eagle Point is part, is generally low lying with swampy areas. The ground slopes gently from a low ridge near the centre of the peninsula towards both Collins Bay to the northwest and Ivison Bay

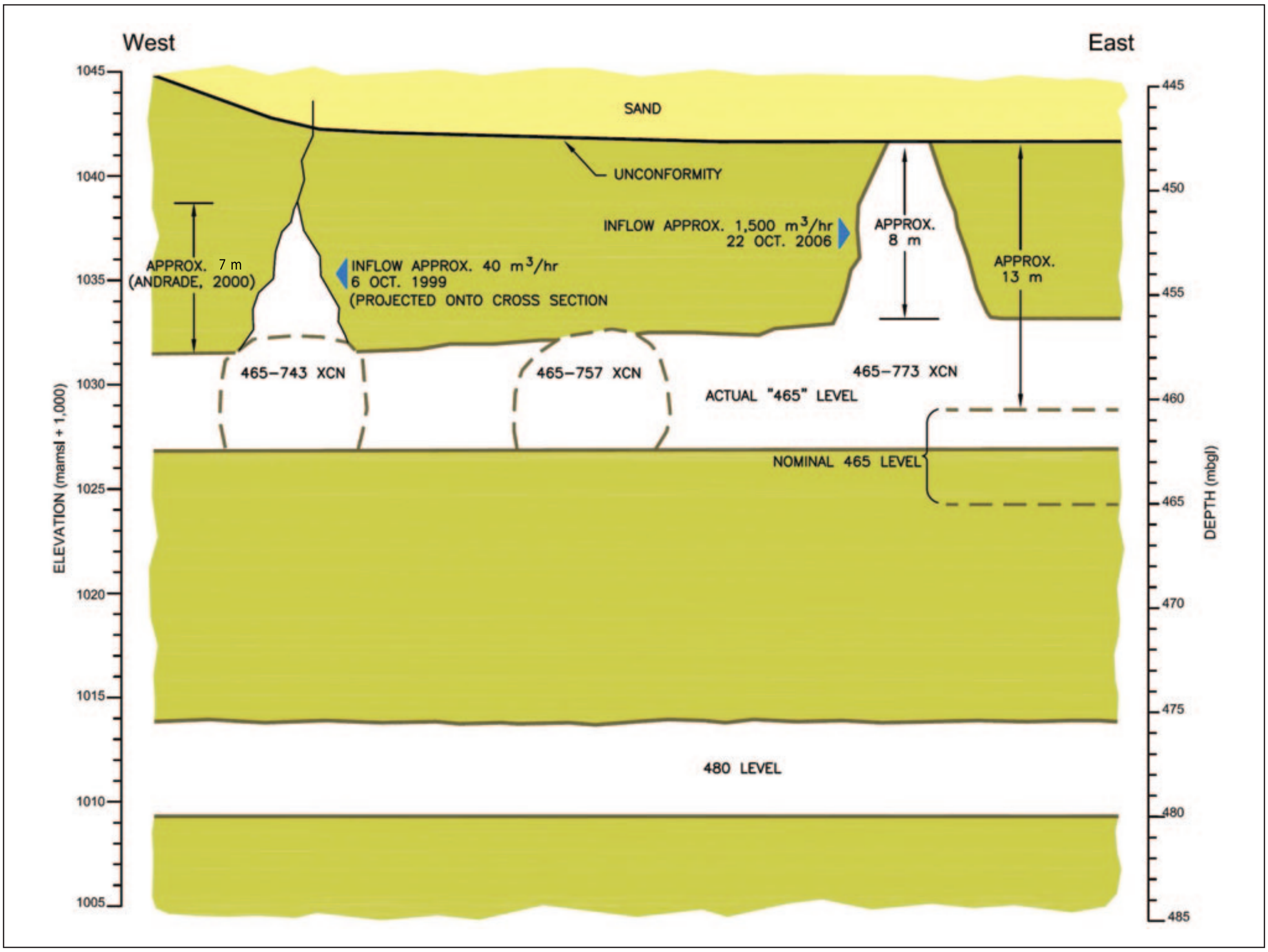

Figure 9. Cross-section of roof failures at 465–743 XCN and 465–944 DRE, Cigar Lake.

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 approximately 40 m. The total length of the deposit is approximately 5760 m, as measured along strike.

Hydrogeological setting

The formation can be divided into following hydrogeological units: 1) surficial sediments; 2) bedrock units; 3) structure; and 4) mineralization and alteration. Figure 14 shows the location of these units at the mine site. A brief description of each unit is provided in the following paragraphs.

The total thickness of the surficial sediments in the Eagle Point area is estimated to range between approximately 6 and 14 m. The average hydraulic conductivity of the total thickness of surficial sediments is estimated to range from 10^{-6} to 10^{-4} m/s. Surficial sediment can be further classified into the following units: soft lake sediments, upper deglacial sediments, lower till, and lower gravel.

The soft lake sediment is typically an organic silt layer only 1 m in thickness, although it thickens to approximately 3 m towards the centre of the lake. The hydraulic conductivity of this unit is fairly low and is estimated to range between 10^{-6} and 10^{-5} m/s. Upper deglacial sediments are possibly a reworked, sorted till and are typically 3–5 m in thickness. They are composed of glaciofluvial 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}–10^{-6} m/s. In areas where the silt and clay are absent, hydraulic conductivities could be as high as 10^{-4} m/s. The lower till unit ranges from 1 m in thickness to approximately 6–7 m 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 of approximately 10^{-7}–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}–10^{-4} m/s.

The bedrock sequence at the Eagle Point mine consists of a granitoid gneiss overlain by the Wollaston group paragneisses. 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, the 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 the alteration occurred. A reasonable range of hydraulic conductivity of the shallower Aphebian rocks is 10^{-8}–10^{-5} m/s; deeper rocks are in the range of 10^{-8}–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 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.

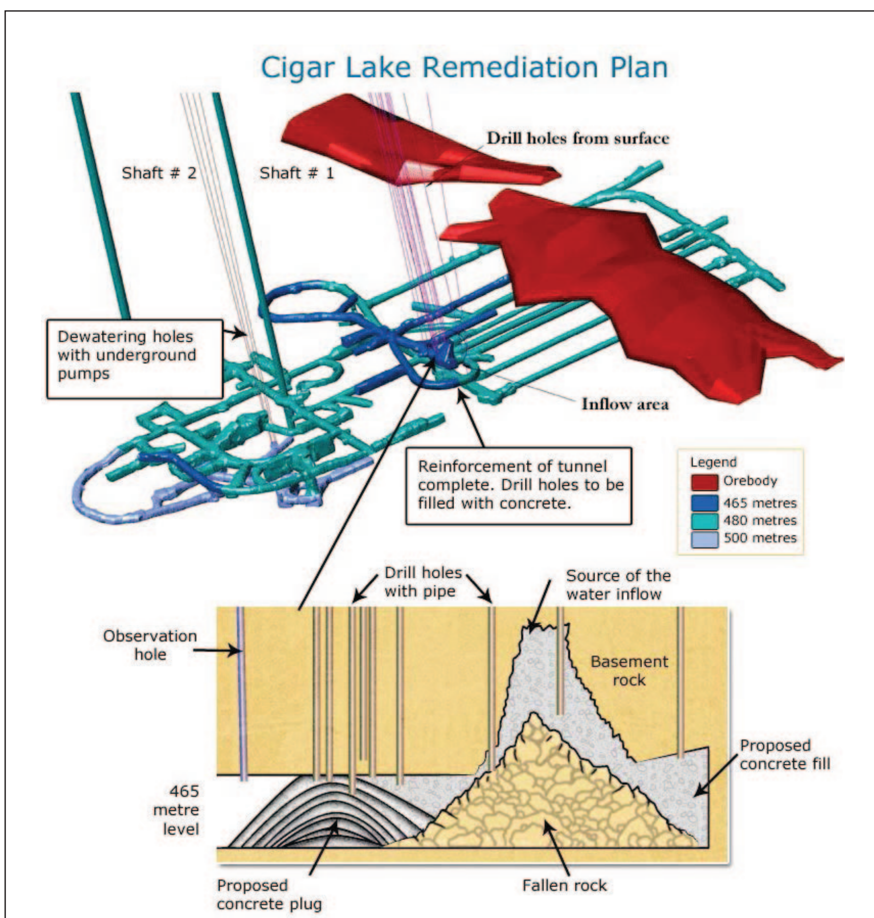

Figure 10. Schematic of the remediation activities, Cigar Lake.

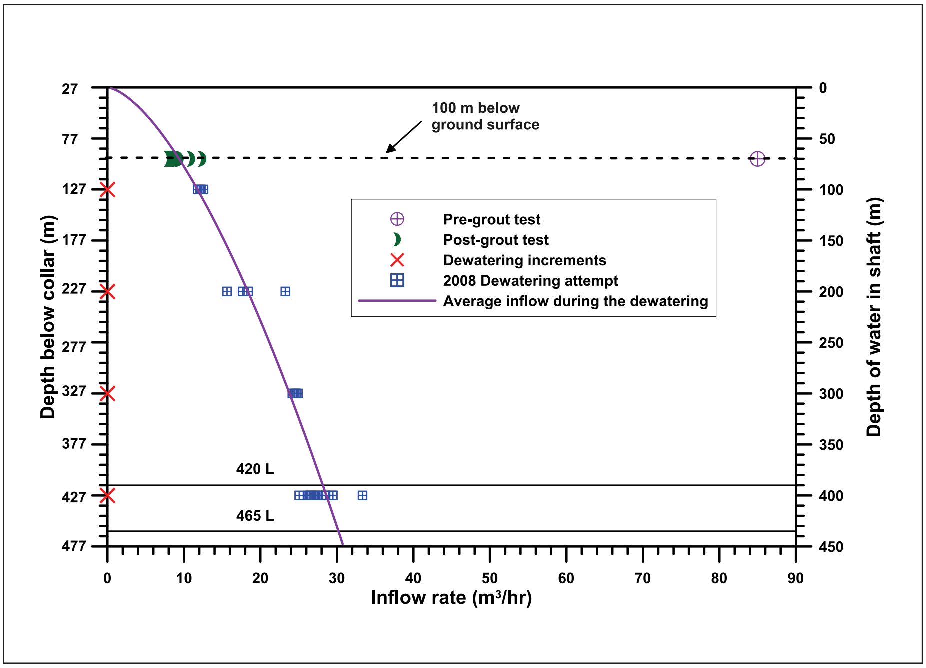

Figure 11. Inflow rate to the mine working under various stages of remediation, Cigar Lake.

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 the McArthur River and Cigar Lake mines. However, parts of the mining areas are under Wollaston Lake, so under a catastrophic failure of the crown pillar (the rock left between the uppermost levels and the bottom of Wollaston Lake), the inflows could vastly exceed any pumping system capacity. The mine dewatering system requires that the mine water be pumped to the mill 15 km away for treatment

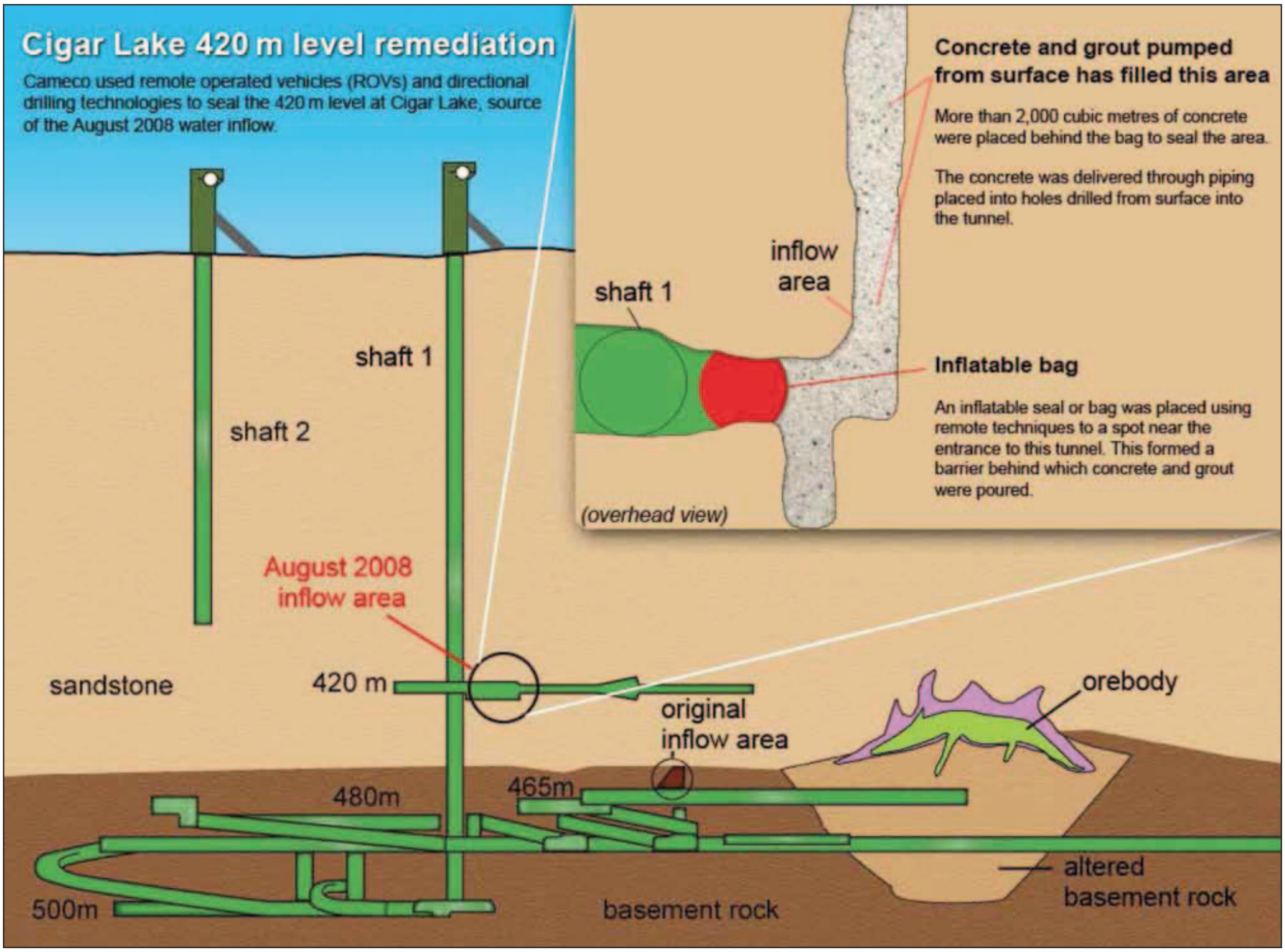

Figure 12. Remediation strategy for inflow remediation on 420 level, Cigar Lake.

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 an intersection with 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 15). Detailed risk assessments are done prior to mine development in these areas.

A direct connection to Collins Bay via a geological 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.

Mine inflow of 2007

An inflow began on November 26, 2007 at the Eagle Point mine. The 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. The 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 surface hole, drilled on winter lake ice and collared at lake bottom, had a structural connection from the hole serving as a conduit to the 90L overcut development at longhole stoping of 105–533 stope in the 03 zone. The hole was drilled as part of the surface diamond drilling program during the 1980s. In the early years, some of the holes drilled from the lake were grouted, whereas others were left ungrouted. The 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 the Darcy-Weisbach formula with a friction factor similar to that of a concrete pipe produced an inflow rate similar to the one measured at the weir. Similar calculations are done routinely in karst hydrogeology for conduit flows (White & White, 2005). A remotely operated vehicle (ROV) was used to search the lake bottom and was successful in positively identifying the hole responsible for the inflow.

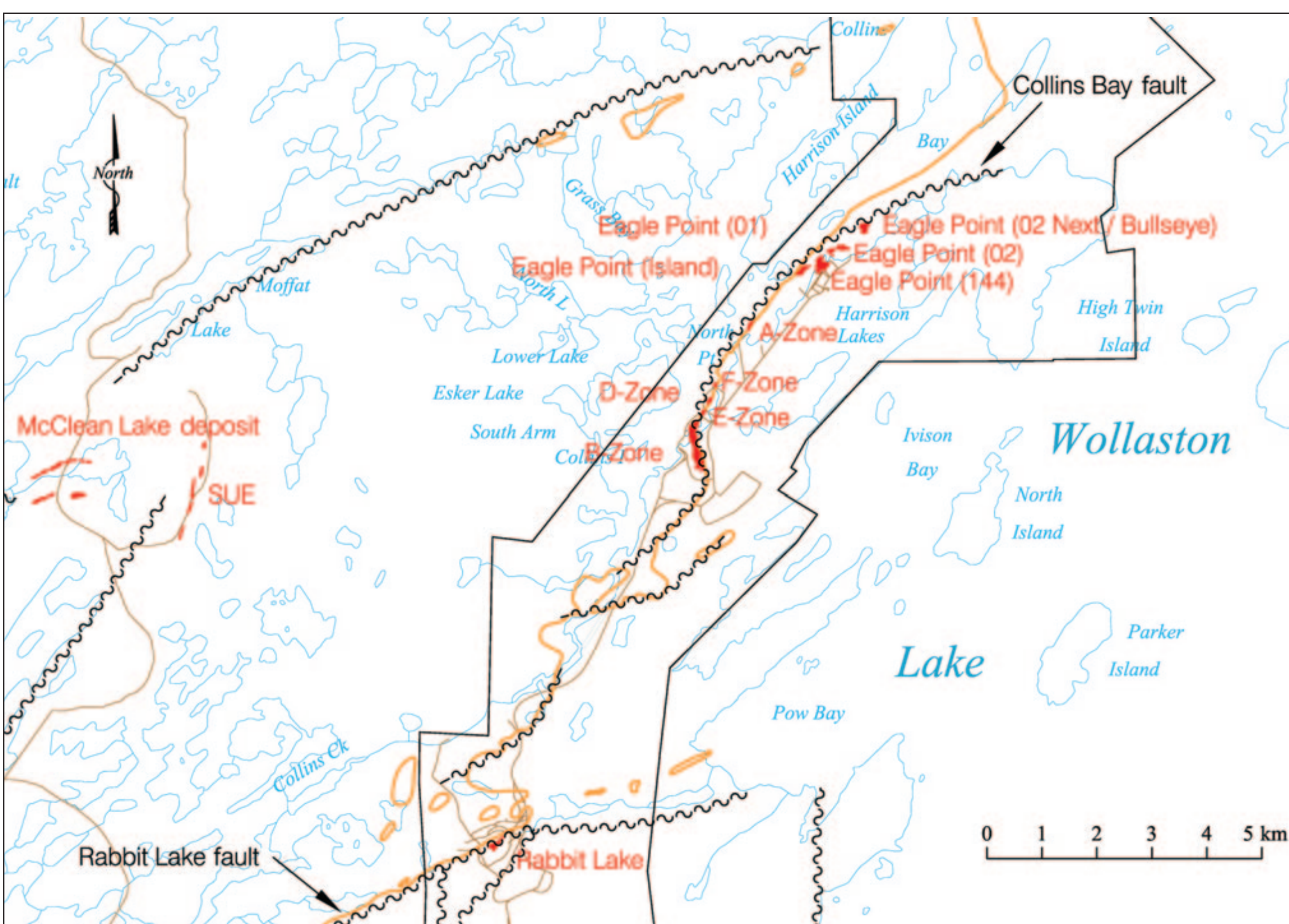

Figure 13. Rabbit Lake operation site map.

Remediation

Although the source of this inflow was 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 hole was plugged from on top of ice (in the lake) using casing and a packer, followed by grouting off the hole with cement against the down hole. 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 was investigated by the TapRooT® investigation method. In the case of the McArthur River 2003 and Cigar Lake 2006 inflows, the TapRooT® investigation was carried out by an independent third party. TapRooT® is a well-established investigative tool that systematically and objectively examines an incident/accident to determine the root causal factors. This is achieved by developing a flow chart of the events and identifying conditions that have a significant impact on the events, enabling the identification of causal factors. TapRooT® focuses closely on the performance of systems in place, including the systems that did not operate. The TapRooT® investigation concludes with development, evaluation, and implementation of corrective

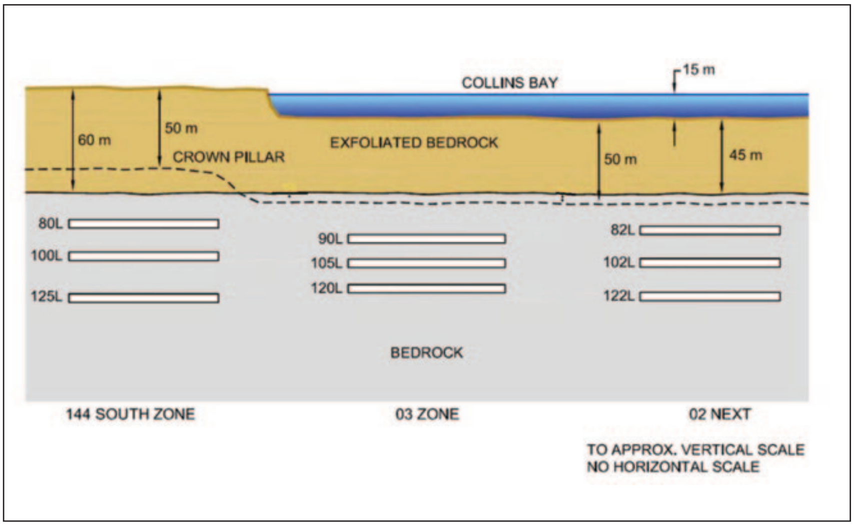

Figure 15. Existing and proposed development on upper parts of the Eagle Point mine.

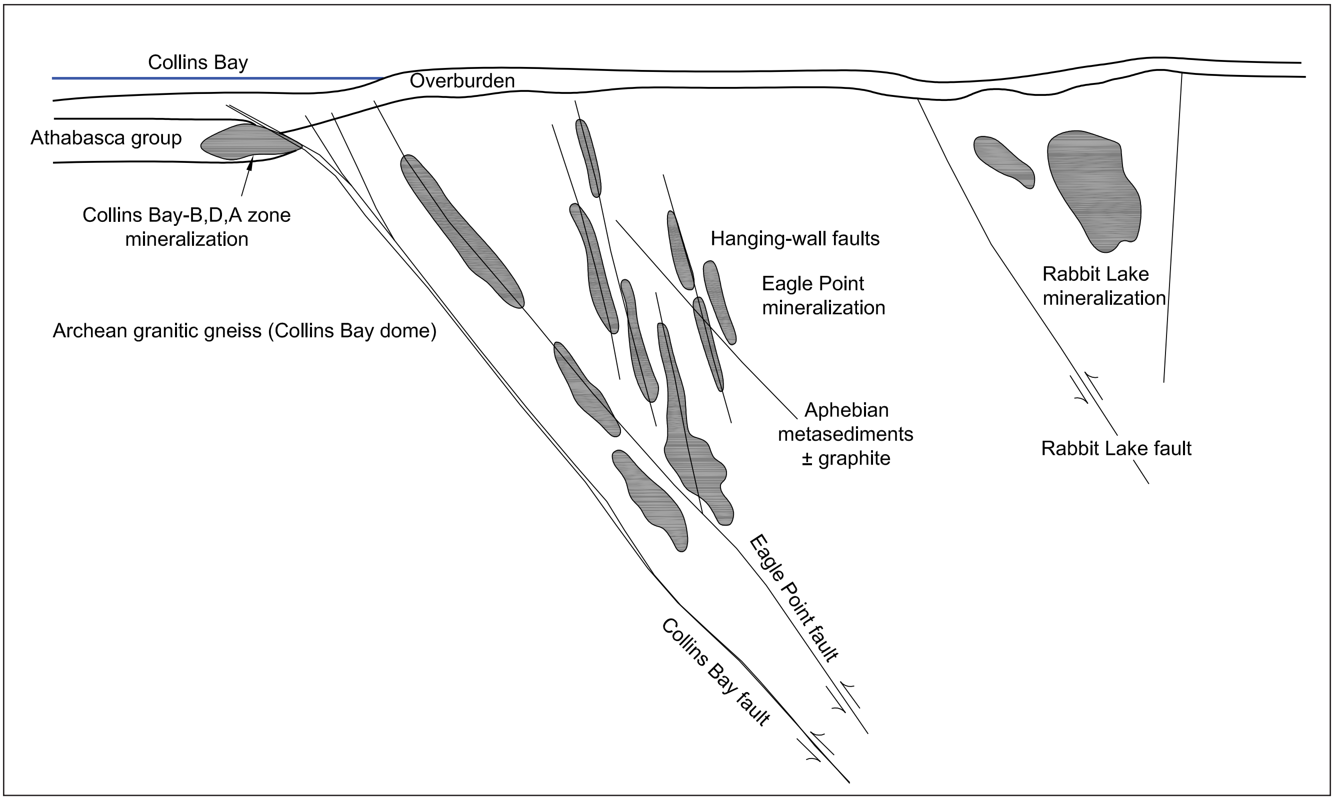

Figure 14. Cross-section of deposits at Rabbit Lake.

actions. The seven steps in a TapRooT® investigation are shown in Figure 16.

Table 1 summarizes the select causal factors identified in the formal TapRooT® investigations for all three inflows and provides 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.

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 proven to be an incorrect assumption because the development in the basement at Cigar Lake in 2006 and McArthur River in 2003 resulted in ground failures that caved upwards into the sandstone. Quantifying how much dewatering capacity is needed becomes a function of maximum inflow rate if a direct connection with the sandstone develops. New groundwater 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 the maximum encountered inflow, and the water storage available underground has also been quantified. Cameco has also developed a corporate standard for predevelopment 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 company wide and sites have been audited both by a corporate mine hydrogeologist and by external auditors. The corporate standard has an explicit requirement for each site to establish a maximum uncontrolled inflow rate and have a suitable dewatering capacity with contingency available before development is to proceed underground. The corporate standard also requires sites to carry out a formal risk assessment before developing in high-risk areas (e.g., close to an unconformity or in areas with surface diamond drill holes) and have mitigation plans in place.

The ground control standards and development standards when mining in proximity to an 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 a third party

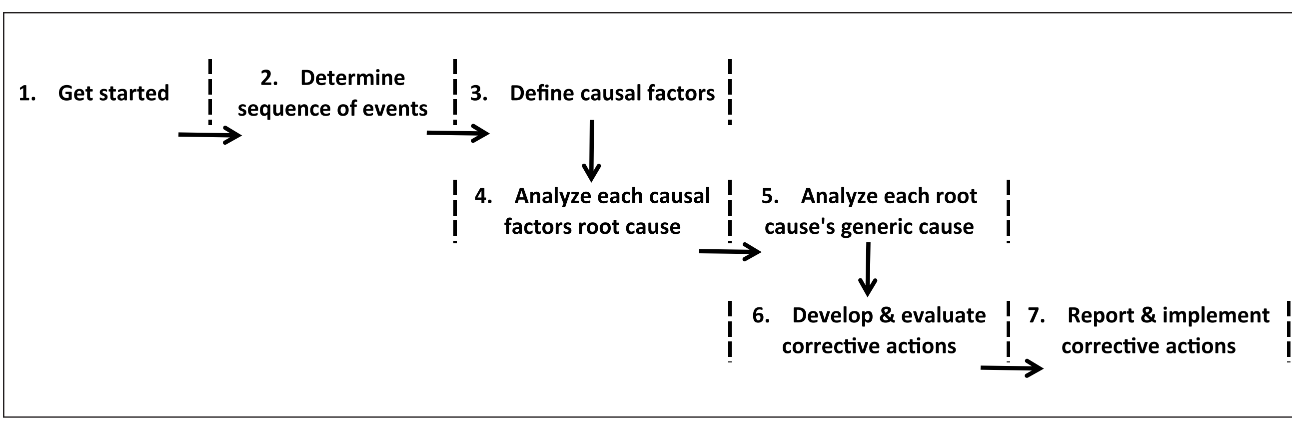

Figure 16. Steps in the TapRooT® investigation.

Table 1. Causal factors (CFs) 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 drillhole grout status
	Alternative excavation and support methods for high-risk development	Formal standards for mine development	Risk assessment omission (grout status of surface drillholes)
		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)		
Total causal factors	**6**	**11**	**4**

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 are conducted by corporate staff and external consultants. To get a better definition of the geological and geotechnical hazards of higher-risk development areas, procedures are in place for documenting results and recommendations of probe and grout campaigns.

As part of comprehensive geotechnical assessment and 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 and mitigating the risk from diamond drill holes.

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

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 unique characteristics associated with the uranium deposits of the Athabasca Basin;
- the events leading up to the 2003 McArthur River and 2006 Cigar Lake inflows and lessons learned;
- basic geology and the effects of water pressure;
- hazards and risks related to inflows; and
- ground control, support, and warning signs.

Training began early in 2008 for employees and contractors at McArthur River, Cigar Lake, and Rabbit Lake.

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Rashid Bashir, PhD, P.Eng. is a senior geotechnical engineer with Golder Associates in their Saskatoon office. He earned his doctorate from McMaster University in Hamilton, ON, with specialization in computational mechanics and contaminant hydrogeology. He also holds an appointment as a professional affiliate in the Department of Civil and Geological Engineering at the University of Saskatchewan.

James Hatley, MBA, P.Eng., PMP is the program director of the McArthur River extension with Cameco Corporation. He has worked in operations, consulting, and executing major projects in the mining industry for more than 20 years. He holds an MBA from Athabasca University specializing in the energy industry and is on the Board of Directors for the Saskatchewan Mining Association. **James_Hatley@cameco.com**

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