Prefeasibility Study Report for the Wheeler River Uranium Project Saskatchewan, Canada

Report Prepared for
Denison Mines Corp.

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1 Summary

1.1 Introduction

The Wheeler River uranium project is an advanced exploration stage joint venture owned by Denison Mines Corp. (Denison) at 90% with the remaining 10% by JCU (Canada) Exploration Company Ltd. (JCU). The greenfield project site is located in northern Saskatchewan in the eastern Athabasca Basin approximately 35km north of Key Lake operation. The Wheeler River property is located in close proximity to important infrastructure, including provincial electrical transmission lines and an all-season provincial highway.

The Wheeler River property has been explored since the late 1970's. In 2004 Denison entered into an agreement with the Wheeler River Joint Venture (WRJV) partners to earn into a majority 60% interest and become operator of the joint venture. Since Denison became the operator, the project has achieved several milestones, including:

- Discovery and delineation of the Phoenix deposit in 2008-2014, including the completion of the May 2014 mineral resource estimate;
- Discovery and delineation of the Gryphon deposit in 2014-2017, including the completion of the January 2018 mineral resource estimate; and
- Completion of a Preliminary Economic Assessment in May 2016.

From an ownership standpoint Denison has recently entered into two separate agreements to increase its ownership interest in the project.

- In January 2017, Denison executed an agreement with the partners of the WRJV that is expected to increase Denison's ownership of the Wheeler River project up to approximately 66% by the end of 2018. Under this agreement, Denison is funding 50% of Cameco's ordinary share of joint venture expenses in 2017 and 2018 (based on Cameco's 30% interest at the time of the agreement). On January 31, 2018, Denison announced that it had increased its interest in the Wheeler River project, based on spending on the project during 2017, from 60% to 63.3% in accordance with this agreement.
- On September 4, 2018, Denison entered into an agreement with Cameco to increase its ownership in the Wheeler River project through the acquisition of 100% of Cameco's minority interest. The acquisition of increased Denison's ownership in the project to 90%.

In December 2017, the WRJV commissioned a Prefeasibility Study (PFS) for the project to be completed in accordance with the Canadian Securities Administrator's National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101), the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) standards and best practices, the AACE International Cost Estimate Classification system as well as other standards. Denison has authorized a select group of qualified and experienced engineering firms (SRK Consulting Inc., Stantec Consulting Ltd, Hatch Ltd, RPA Inc., and Woodard and Curran Inc.) to lead in the preparation of the NI 43-101 PFS. Certain other specialized firms have also contributed to the PFS including Arcadis Canada Inc., Engcomp Engineering and Computing Professionals Inc., Newmans Geotechnique Inc., North Rock Mining Solutions Inc., Paterson and Cooke Canada Inc. and Clifton and Associates amoung others. The objective of the PFS is to assess the technical and economic viability of uranium production at Wheeler River and to provide input into the strategic development of the property. This technical report aims to provide a full description of the work completed on all aspects of the project in accordance with the above standards.

1.2 Technical Summary

1.2.1 Property Description, Location, and Access

The property is located along the eastern edge of the Athabasca Basin in northern Saskatchewan, Canada, approximately 35 km north-northeast of the Key Lake mill and 35 km southwest of the McArthur River uranium mine.

Access to the property is by road or air from Saskatoon. The property is well located with respect to all-weather roads and the provincial power grid. Vehicle access to the property is by the provincial highway system to the Key Lake mill, then by the ore haul road between the Key Lake and McArthur River operations to the eastern part of the property. The Fox Lake access road also runs between Key Lake and McArthur River and provides access to most of the northwestern side of the property. Gravel and sand roads and drill trails provide access by either four-wheel-drive or all-terrain-vehicle to the rest of the property.

1.2.2 Land Tenure

The property consists of 19 mineral claims totalling 11,720 ha with an aggregate annual requirement of CAD$293,000 in either work or cash to maintain title to the mineral claims. Based on previous work submitted and approved by the province of Saskatchewan, title is secure until 2035.

Any uranium produced from the Wheeler River property is subject to uranium mining royalties in Saskatchewan, in accordance with Part III of the Crown Mineral Royalty Regulations. There is a 10% Net Profits Interest (NPI) associated with the property held by the WRJV in approximate proportion to the ownership interests of each WRJV participant. There are no other back-in rights or third party royalties applicable to this property.

There are no known environmental liabilities associated with the property, and there are no other known significant factors and risks that may affect access, title, or the right or ability to perform work on the property. All necessary permits for surface exploration on the property are in place and current. Additional permits and licenses will be required (refer to section 20) prior to commencement of development and production activities.

1.2.3 Geology and Mineralization

The Wheeler River property is located near the southeastern margin of the Athabasca Basin in the southwest part of the Churchill Structural Province of the Canadian Shield. The Athabasca Basin is a broad, closed, and elliptically shaped cratonic basin with an area of 425 km (east-west) by 225 km (north-south). The bedrock geology of the Athabasca basin area consists of Archean and Paleoproterozoic gneisses unconformably overlain by up to 1,500 m of flat-lying unmetamorphosed sandstones and conglomerates of the mid-Proterozoic Athabasca Group.

The Wheeler River property is located near the transition zone between two prominent litho-structural domains within the Precambrian basement, namely the Mudjatik Domain to the west and the Wollaston Domain to the east. The Mudjatik Domain is characterized by elliptical domes of Archean granitoid orthogenesis separated by keels of metavolcanic and metasedimentary rocks, whereas the Wollaston Domain is characterized by tight to isoclinal, northeasterly trending, doubly plunging folds developed in Paleoproterozoic metasedimentary rocks of the Wollaston Supergroup, which overlie Archean granitoid orthogenesis identical to those of the Mudjatik Domain. The area is cut by a major northeast-striking fault system of Hudsonian Age. The faults occur predominantly in the basement rocks but often extend up into the Athabasca Group due to several periods of post-depositional movement.

Local geology is comprised of relatively undeformed late Paleoproterozoic to Mesoproterozoic Athabasca Group strata comprised of Manitou Falls Formation sandstones and conglomerates which unconformably overlie the crystalline basement and have a considerable thickness from 170 metres over the quartzite ridge to at least 560 m on the western side of the property. Basement rocks beneath the Phoenix and Gryphon deposits are part of the Wollaston Domain and are comprised of metasedimentary and granitoid gneisses. The metasedimentary rocks include graphitic and non-graphitic pelitic and semipelitic gneisses, meta-quartzite, and rare calc-silicate rocks. Pegmatitic segregations and intrusions are common in all units with garnet, cordierite, and sillimanite occurring in the pelitic strata, indicating an upper amphibolite grade of metamorphism. Graphitic pelite and quartzite units appear to play important roles in the genesis of Athabasca Basin unconformity-type deposits. Thus, the presence of extensive subcrop of both units (18 km of quartzite and 152 line-km of conductors, assumed to be graphitic pelite) greatly enhances the geological potential of the Wheeler River property. The Wheeler River property is partially covered by lakes and muskeg, which overlie a complex succession of glacial deposits up to 130 m in thickness. These include eskers and outwash sand plains, well-developed drumlins, till plains, and glaciofluvial plain deposits. The orientation of the drumlins reflects southwesterly ice flow.

The Phoenix uranium deposit was discovered in 2008 and can be classified as an unconformity-related deposit of the unconformity-hosted variety. The deposit straddles the sub-Athabasca unconformity approximately 400 m below surface and comprises three zones (A, B, and C) which cover a strike length of 1.1 km. The deposit consists of an exceptionally high-grade core surrounded by a lower grade shell. The deposit is interpreted to be structurally controlled by the WS shear, a prominent basement thrust fault which occurs in the footwall of a graphitic-pelite and the hangingwall of a garnetiferous pelite and quartzite unit. Mineralization within the Phoenix deposit lenses is dominated by massive to semi-massive uraninite associated with an alteration assemblage comprising hematite, dravitic tourmaline, illite, and chlorite. Secondary uranium minerals (including uranophane) and sulphides are trace in quantity.

The Gryphon uranium deposit was discovered in 2014 and can be classified as an unconformity-related deposit of the basement-hosted variety. The deposit occurs within southeasterly dipping crystalline basement rocks of the Wollaston Supergroup below the regional sub-Athabasca Basin unconformity. The deposit is located from 520 m to 850 m below surface, has an overall strike length of 610 m and dip length of 390 m, and varies in thickness between 2 m and 70 m, depending on the number of mineralized lenses present. The mineralized lenses are controlled by reverse fault structures, which are largely conformable to the basement stratigraphy and dominant foliation. The A, B, and C series of lenses are comprised of stacked, parallel lenses which plunge to the northeast along a fault zone (G-Fault) which occurs between hangingwall graphite-rich pelitic gneisses and a more competent pegmatite-dominated footwall. A ubiquitous zone of silicification (Quartz-Pegmatite Assemblage) straddles the G-Fault and the A, B, and C series of lenses occur in the hangingwall of, within, and in the footwall of the Quartz-Pegmatite Assemblage respectively. The D series lenses occur within the pegmatite-dominated footwall along a secondary fault zone (Basal Fault) or within extensional relay faults which link to the G-Fault. The E series lenses occur along the G-Fault, up-dip and along strike to the northeast of the A and B series lenses, within the upper basement or at the sub-Athabasca unconformity. Mineralization within the Gryphon deposit lenses is dominated by massive, semi-massive, or fracture-hosted uraninite associated with an alteration assemblage comprising hematite, dravitic tourmaline, illite, chlorite, and kaolinite. Secondary uranium minerals (including uranophane and carnotite) and sulphides are trace in quantity.

1.2.4 Sampling, Analysis and Data Verification and Mineral Resources

The updated mineral resource estimate for the Gryphon deposit was prepared for Denison by RPA in accordance with CIM Definitions (2014). The effective date of the updated Gryphon mineral resource estimate is January 30, 2018. The mineral resource estimate for the Phoenix deposit with an effective date of May 28, 2014 remains current, as no further resource drilling has been completed on this deposit. The Phoenix cut-off grade of 0.8% U_3O_8 is based on internal conceptual studies by Denison and a price of USD$50/lb U_3O_8, while the cut-off grade of 0.2% U_3O_8 for Gryphon is based on RPA estimates using assumptions based on historical and known mining costs from mines operating in the Athabasca Basin, incremental operating costs for low-grade material, and a price of USD$55/lb U_3O_8.

As provided in Table 1-1 below, the total indicated mineral resources for both the Phoenix and Gryphon deposits are estimated at 1,809,000 tonnes at an average grade of 3.3% U_3O_8 containing 132.1 million pounds of U_3O_8. Total inferred mineral resources are estimated at 82,000 tonnes at an average grade of 1.7% U_3O_8 containing 3.0 million pounds of U_3O_8. Mineral resources that are not classified as mineral reserves do not have demonstrated economic viability.

Table 1-1. RPA Mineral Resource Estimate – Wheeler River Project – January 30, 2018

Deposit	Category	Tonnes	Grade (% U_3O_8)	Million lbs U_3O_8 (100% Basis)	Million lbs U_3O_8 (Denison 63.3%)
Gryphon	Indicated	1,643,000	1.7	61.9	39.2
Phoenix	Indicated	166,000	19.1	70.2	44.4
Total Indicated		**1,809,000**	**3.3**	**132.1**	**83.6**
Gryphon	Inferred	73,000	1.2	1.9	1.2
Phoenix	Inferred	9,000	5.8	1.1	0.7
Total Inferred		**82,000**	**1.7**	**3.0**	**1.9**

Notes:
1. CIM definitions (2014) were followed for classification of mineral resources.
2. Mineral resources for the Gryphon deposit are estimated at an incremental cut-off grade of 0.2% U3O8 using a long-term uranium price of USD$50/lb and a USD$/CAD$ exchange rate of 0.75. The cut-off grade is based on incremental operating costs for low-grade material.
3. Mineral resources for the Phoenix deposit are reported above a cut-off grade of 0.8% U3O8. Mineral resources for the Phoenix deposit were last estimated in 2014 to reflect the expansion of the high-grade zone. As no new drilling has been completed at Phoenix since that time, the mineral resource estimates for the Phoenix deposit remain current.
4. Bulk density is derived from grade using a formula based on 196 measurements from Phoenix and 279 measurements from Gryphon.
5. Numbers may not add due to rounding.
6. Mineral resources are inclusive of mineral reserves.
7. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Phoenix Deposit Estimation Methodology

The mineral resource estimate at Phoenix is based on data collected from several surface diamond drilling campaigns from 2008 to 2014.

Geology, structure, and the size and shape of the mineralized zones have been interpreted using data from 243 diamond drill holes, which resulted in three-dimensional wireframe models that represent 0.05% U_3O_8 grade envelopes. The mineralization model consists of a higher-grade zone within an envelope of lower grade material, resulting in two main estimation domains - higher grade and lower grade. Additionally, a new domain representing a small zone of structurally controlled basement mineralization was added at the north end of the deposit.

Based on 196 dry bulk density determinations, Denison developed a formula relating bulk density to uranium grade which was used to assign a density value to each assay. Bulk density values were used to weight grades during the resource estimation process and to convert volume to tonnage.

Uranium grade times density (GxD) values and density (D) values were interpolated into blocks in each domain using an inverse distance squared (ID^2) algorithm. Hard domain boundaries were employed such that drill hole grades from any given domain could not influence block grades in any other domain. Very high-grade composites were not capped but grades greater than a designated threshold level for each domain were subject to restricted search ellipse dimensions in order to reduce their influence. Block grade was derived from the interpolated GxD value divided by the interpolated D value for each block. Block tonnage was based on volume times the interpolated D value.

The mineral resource estimate for the Phoenix deposit was classified as indicated and inferred based on drill hole spacing and apparent continuity of mineralization. The block models were validated by comparison of domain wireframe volumes with block volumes, visual comparison of composite grades with block grades, comparison of block grades with composite grades used to interpolate grades, and comparison with estimation by a different method.

Gryphon Deposit Estimation Methodology

The updated mineral resource estimate for the Gryphon deposit is based on a total of 210 drill holes. The three-dimensional mineralized wireframes were created by Denison utilizing Gemcom software following detailed interpretation of the deposit geology and structure, and then audited for completeness and accuracy by RPA using Vulcan software. The wireframes were defined using a threshold of 0.05% U_3O_8 and minimum thickness of two metres. One higher grade domain was defined within the A1 lenses and three higher grade domains were defined in the D1 lenses based on a threshold of 4.0% U_3O_8.

Based on 279 dry bulk density determinations, a polynomial formula was determined relating bulk density to uranium grade, which was used to assign a density value to each assay. Bulk density values were used to weight grades during the resource estimation process and to convert volume to tonnage. GxD values and D values were interpolated into blocks measuring five metres by one metre by two metres using an ID^2 algorithm since variograms were not considered appropriate to derive kriging parameters. Hard domain boundaries were employed at the wireframe edges, so that blocks within a given wireframe were only informed by grade data from that wireframe. For the A1 high-grade domain, assays were capped at 30% U_3O_8 with a search restriction applied to composite grades over 20%, and for the D1 high-grade domains, assays were capped at 20% U_3O_8 with no search restriction. For the A1-A4, B3-B7, C4-C5, and D2-D4 low-grade domains, assays were capped at 10% U_3O_8. For the C1 low-grade domain, assays were capped at 20% U_3O_8 with a search restriction applied to composite grades over 10%. For the B1, B2, E1, and E2 low-grade domains, assays were capped at 15% U_3O_8 with search restrictions applied to composite grades over 10% U_3O_8 for the B1 domain and 5.0% U_3O_8 for the E2 domain. For the D1 low-grade domain, assays were capped at 5% U_3O_8. Block grade was derived from the interpolated GxD value divided by the interpolated D value for each block. Block tonnage was based on volume times the interpolated D value.

The mineral resource estimate for the Gryphon deposit was classified according to the drill hole spacing and the apparent continuity of mineralization, as either indicated mineral resources (generally, drill hole spacing of 25 m x 25 m) or inferred mineral resources (generally, drill hole spacing of 50 m x 50 m). The block models were validated by comparison of domain wireframe volumes with block volumes, visual comparison of composite grades with block grades, comparison of block grades with composite grades used to interpolate grades, and comparison with estimation by a different method.

1.2.5 Mineral Reserves within PFS Design Plan

Phoenix

The mineral reserve for Phoenix is estimated at 59.7 million pounds of U_3O_8 at an average grade of 19.1% over 141,000 tonnes as summarized in Table 1-2. The mineral reserve was prepared by Woodard & Curran Inc. based on the mineral resources prepared by RPA and the use of In situ Recovery (ISR) mining method. The ISR process has been designed to a level appropriate for a PFS. The mineral reserve estimate stated herein is consistent with CIM definitions and is suitable for public reporting. As such, the mineral reserves can only be based on measured and indicated mineral resources and cannot include any inferred mineral resources. The Phoenix mineral resource does not include any measured resource material. Indicated resources are converted directly to probable reserves.

Table 1-2. Mineral Reserve Estimate - Phoenix

Category	Million lbs U_3O_8	Grade	Tonnes
Proven	0	0	0
Probable	59.7	19.1%	141,000
TOTAL	**59.7**	**19.1%**	**141,000**

Notes:
1. CIM definitions (2014) were followed for classification of mineral reserves.
2. Mineral reserves are stated at a processing plant feed reference point.
3. Mineral reserves for the Phoenix deposit are reported at the mineral resource cut-off grade of 0.8% U_3O_8. The mineral reserves are based on the block model generated for the May 28, 2014 mineral resource estimate. A mining recovery factor of 85% has been applied to the mineral resource above the cut-off grade.

Gryphon

The Gryphon mine production plan is based on using longhole mining methods to recover the ore located between approximately -30 and -280 metres above sea level (MASL). The mineral reserve for Gryphon is estimated at 49.7 million pounds U_3O_8 (1.2M tonnes grading at 1.8% U_3O_8) as summarized in Table 1-3. The mineral reserve has been estimated by Stantec based on the resource block model prepared by RPA.

The mine design and mineral reserve estimate have been completed to a level appropriate for a PFS. The mineral reserve estimate stated herein is consistent with CIM definitions, based on indicated mineral resources, and does not include any inferred mineral resources. The Gryphon block model did not include any measured resource material. Indicated resources are converted directly to probable reserves. The inferred resources contained within the mine design are classified as waste.

Table 1-3. Mineral Reserve Estimate - Gryphon

Category	Tonnes	Grade (% U_3O_8)	Million lbs U_3O_8
Proven	0	0.0	0
Probable	1,257,000	1.8	49.7
TOTAL	**1,257,000**	**1.8**	**49.7**

Notes:
1. CIM definitions (2014) were followed for classification of mineral reserves.
2. Mineral reserves are stated at a processing plant feed reference point.
3. Mineral reserves for the Gryphon deposit are estimated at a cut-off grade of 0.58% U_3O_8 using a long-term uranium price of USD$40/lb and a USD$/CAD$ exchange rate of 0.80. The mineral reserves are based on an operating cost of $574/tonne, milling recovery of 97%, and 7.25% fee for Saskatchewan royalties. Mineral reserves include for diluting material and mining losses.

1.2.6 Hydrogeology

Hydrogeological conditions at the Wheeler River deposits were assessed during drilling programs in 2015 through 2018. Data from the hydraulic testing, pressure transducer systems, water levels surveys, water chemistry, and lab testing of core samples were combined with geological modelling and structural interpretation to build an understanding of the hydrogeological system.

The natural groundwater elevation at Phoenix is shallow, within a few meters of surface elevation. The deposit, sitting at the unconformity at a nominal 420 m depth, is below the natural ground water elevation and is therefore subject to the full hydrostatic head of the water in the overlying sandstone. The sandstone formation surrounding the deposit and in the hangingwall is highly fractured and unconsolidated, resulting in highly permeable ground conditions. Rock quality in the basement rock below the deposit and the associated paleo weathering profile indicates fair to good rock mass conditions with the presence of fault structures. These units are much more competent with localized fractures and structures. As a whole, the basement units have lower permeability (restricting water flow) with localized structures or fractures bearing higher permeability. The deposit itself is a highly variable mass with localized zones of competent and lower permeable conditions, intermixed with areas of unconsolidated zones with much higher permeability.

The hydrogeological system surrounding the Phoenix deposit has been assessed as it pertains to the proposed ISR mining option. The mine design, with the encapsulating freeze dome and underlying basement formations, provides a controlled groundwater system that will greatly simplify control of ISR fluids due to the hydraulic containment. Testing to date indicates that fluid flow through the mineralized portion of the deposit is expected to be viable at rates required for designed production.

The Gryphon deposit has an extensive data set with data covering the overlying Athabasca formations, the regional unconformity, and the basement complex. This data, captured during drill programs in 2015-2018, was used to model potential inflows to the basement hosted deposit and underground workings, with results comparing very closely to similar mines in the Athabasca Basin.

Similar to Phoenix, the natural groundwater elevation is at or close to surface and is hydraulically connected to the unconformity located at approximately 465 m below surface. The sandstone above the unconformity features a smaller alteration signature with generally competent sandstone with lower permeability features. With the exception of the shafts, no mine development occurs in the sandstone, thus no geotechnical assessments were completed in this unit. The excavation method and liner for both the production and ventilation shafts were designed considering the results of the shaft test hole program completed in 2017.

Potential for inflow of water from the overlying unconformity and Athabasca formations was assessed both numerically and benchmarked against other mines in the Athabasca Basin. Based on the low hydraulic conductivity of the unconformity in all tests to date at Gryphon and the lack of identified hydraulic connection between the unconformity and the mining zones, the risk of inflow in this geological zone is considered to be lower than at other Athabasca basin operations. Inflow control and risk mitigation from the overlying sedimentary units is considered to be feasible through mine design (avoidance of thin crown pillar) and mining practice (assessment and mitigation through probe and grout) in potential areas of suspected steep angle structure, etc.

1.2.7 Mine Geotechnical

For the Gryphon deposit, the PFS mine plan proposes two underground mining methods, longitudinal and transverse long hole stoping with cemented rock backfill (CRF). A 15 m level spacing is proposed with longitudinal stopes averaging 5.9 m wide, 17 m along strike. Stope dimensions were analyzed and defined using the empirical open stope design methodology known as Mathews-Potvin or the Stability Graph Method (Hutchinson & Diederichs 1996). Application of the method indicates the deposit is amenable to the planned longitudinal and transverse long hole stoping (refer to section 16).

Rock mass quality throughout the Gryphon deposit typically ranges from predominantly “FAIR” to “GOOD” using established rock mass classification terminology (Rock Mass Rating (RMR) and Q-systems). Within the Basement units the intact rock strength can vary between R0 (very weak) to R5 (very strong). ‘Typical’ fresh basement rock is classified as R3 (strong rock, 50-100 MPa). In the mine plan a standard ground support pattern of bolts and screen has been designed to control rock movement during operations. However, during operations there are likely to be localized areas that may be subject to mining-induced deterioration of ground conditions which may require additional ground support.

Recent hydrogeological assessment completed by SRK indicates relatively low hydraulic conductivity rock mass conditions within the basement units at Gryphon. For geotechnical engineering purposes the rock masses have been considered wet but dewatered (i.e. not subject to significant water pressure or flows). In the event that permeable geological structures are encountered during mine development, localized water control grouting campaigns may be required. Rock discontinuities generally appear amenable to cementitious grout injection.

The upper portion of the mining horizon is located ~25 m below the unconformity and below the paleo weathering profile. It is recommended to complete this and other high-risk development under probe and grout cover programs to reduce the risk of uncontrolled water inflow.

1.2.8 Phoenix Mining

In situ recovery (ISR), also known as solution mining, involves leaving the ore in the ground and extracting the minerals from the ore by dissolution using a leaching solution (lixiviant) and recovering the uranium bearing solution. Once recovered, the solution is transported to a mineral processing facility, where the uranium is recovered in much the same way as in any other uranium mill (World Nuclear Association, October 2017, “In Situ Leach Mining of Uranium”). Consequently with ISR mining there is minimal surface disturbance and no tailings or waste rock generated.

Denison plans to mine the Phoenix deposit using the ISR extraction method with a low pH lixiviant. This same method is successfully used in global ISR operations in areas such as Kazakhstan and

Australia. Uranium ISR uses the native groundwater in the orebody, which is fortified with a low pH solution and in most cases an oxidant. Pumping the solution through the ore zone and allowing the solution to contact the ore requires sufficient permeability of the ground. As noted in Section 1.2.6, the Phoenix deposit does exhibit the required permeability conditions and is expected to be amenable for ISR mining.

Physically ISR mining is conducted through drill holes from surface to the orebody, known as wells. Wellfields are the groups of wells, installed and completed in the mineralized zones that are designed to effectively target delineated mineralization and reach the desired production goals. The mineralized zones are the geological sandstone units where the leaching solutions are injected and recovered via wells in an ISR wellfield. At present, the drilling of individual wells will be carried out utilizing either air rotary or mud rotary methods. The wellfield at Phoenix has been designed using a standard hexagonal pattern with 10m spacing between wells. Table 1-4 includes the total meters and drillholes required for the life of mine using this design and Figure 1-1 shows an isometric view of the drillholes from surface.

Table 1-4. Cumulative Wells and Meterage for the Phoenix Orebodies

Well Type	Number of Wells	Meterage
Recovery Wells	94	40,420.00
Injection Wells	199	85,570.00
Monitoring Wells	17	7,310.00
Total	**310**	**133,300.00**

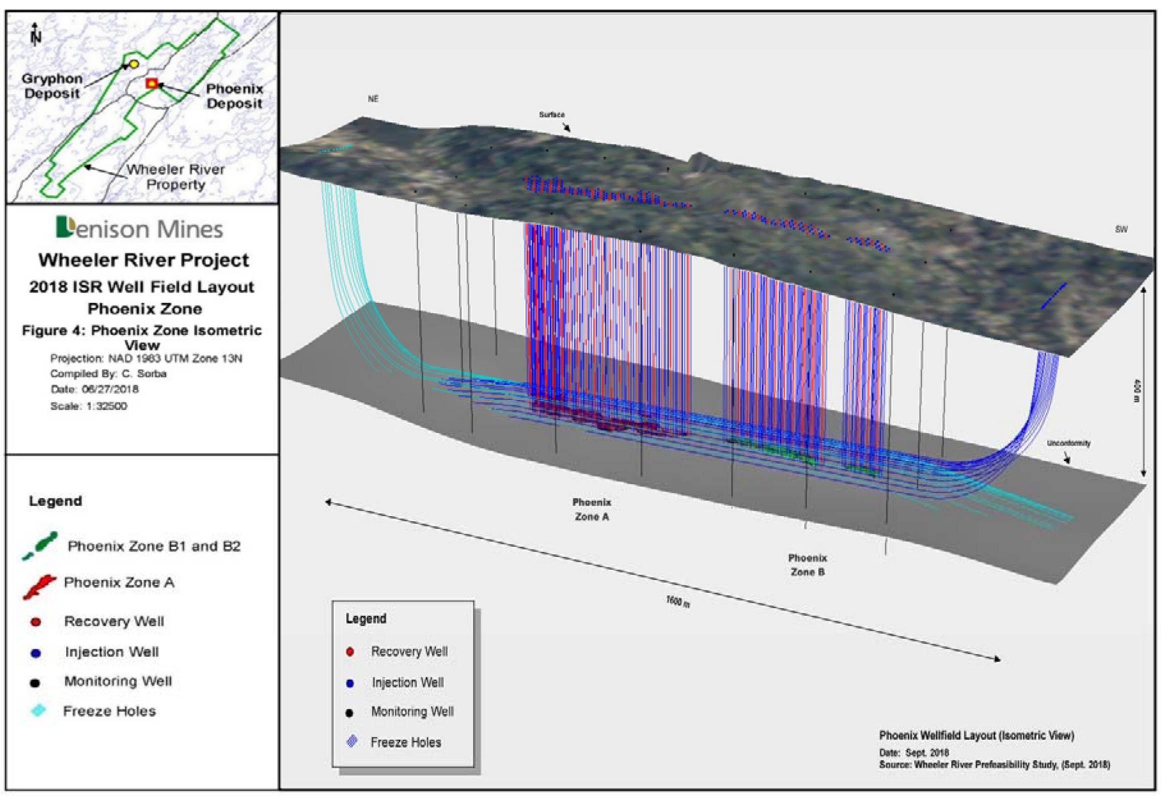


Figure 1-1. Isometric View of Phoenix Wellfield Layout

The uranium ISR process proposed in this PFS will involve the dissolution of the water-soluble uranium compound from the mineralized host sandstones at low pH ranges using acidic solutions. The acidic solution will dissolve and mobilize the uranium, allowing the dissolved uranium to be pumped to the surface within the mining solution.

Containment of the solution is a requirement in ISR operations to ensure recovery of the uranium and to minimize regional groundwater infiltration into the ore zone and associated dilution of the mining solution. In typical ISR operations, this is normally achieved through natural clay or other impermeable geological layers. At Phoenix, the basement rock below the orebody achieves this purpose but the sandstone formation which hosts and surrounds the ore zone is not impermeable and is hydraulically connected to the regional groundwater system throughout the Athabasca basin. As a result, in order to maintain containment, the entire orebody will be isolated by use of an artificial freeze wall that will cover all sides and above the orebody to create an impermeable dome to surround the deposit. This dome will be keyed into the impermeable basement rocks on all sides. The freeze wall will be established by drilling a series of cased holes from surface and across the orebody, and keyed into the basement rock. Circulation of a low temperature brine solution in the holes will remove heat from the ground, freezing the natural groundwater, and establishing an impermeable frozen wall encapsulating the deposit (Figure 1-2).

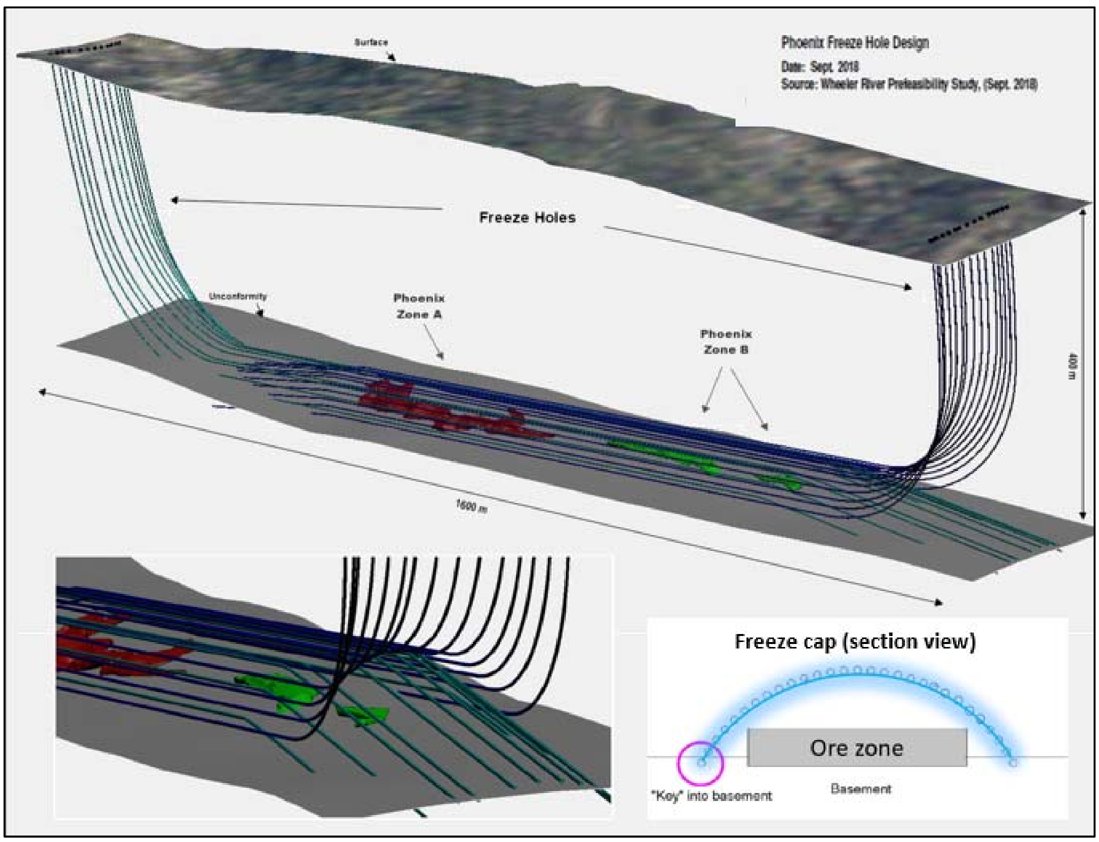


Figure 1-2. Phoenix Freeze Cap Design

After the low pH solution has passed through the deposit, dissolving uranium, it will be pumped to a surface processing plant for uranium recovery. Due to the low impurity levels in the ore and the high uranium concentration of the solution expected to be recovered (due to the high grade nature of Phoenix mineralization), processing of the uranium bearing solution is not expected to require certain typical recovery circuits (i.e. ion exchange and solvent extraction) and may be directly precipitated out of solution. The precipitates are then dried and packaged for sale. The barren solution is then re-fortified with reagents and recycled through the process, resulting in a closed loop system with no expected discharge to the environment. Figure 1-3 shows the ISR mining process designed for Phoenix.

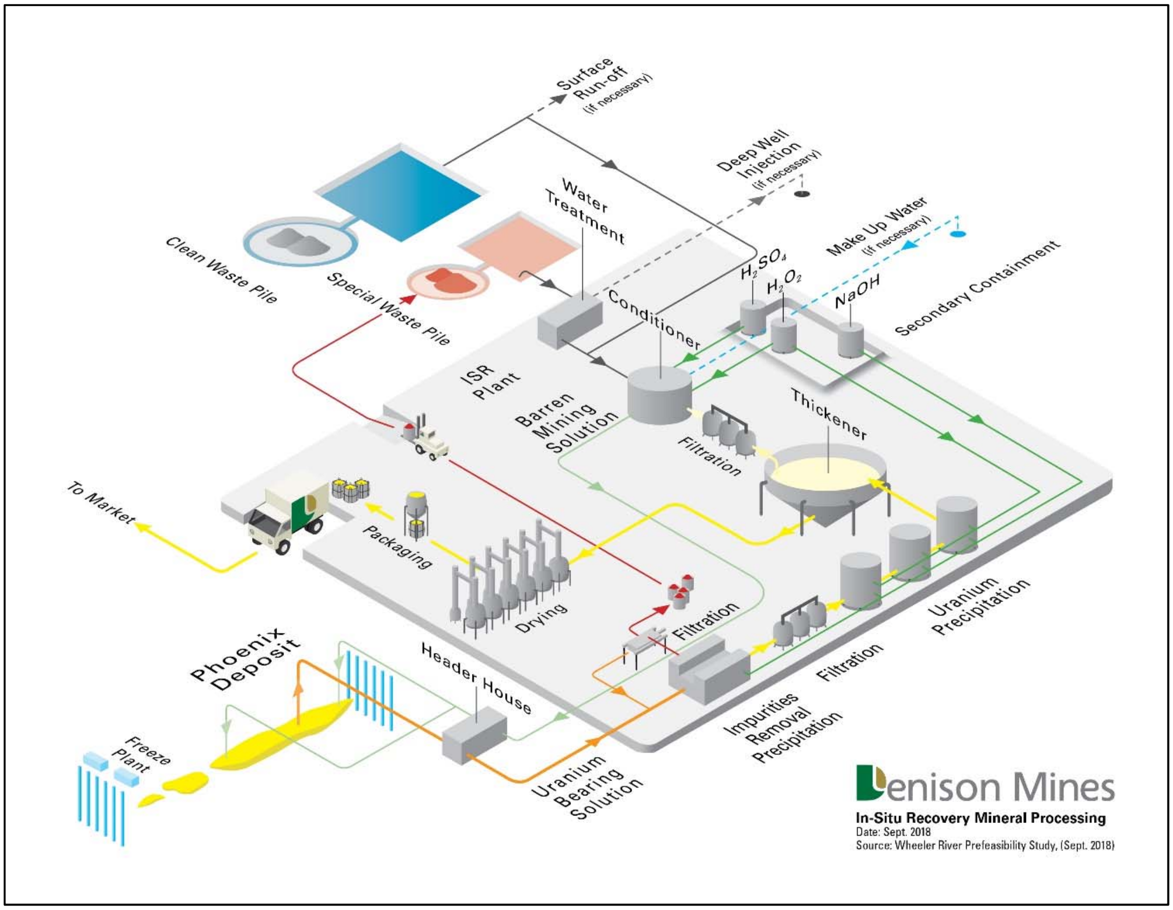


Figure 1-3. Phoenix ISR Design

The authors have estimated the mine life based on head grade, estimated resource, flow rates, and closure requirements for the Phoenix deposit. Production will occur consecutively and simultaneously across the Phoenix deposit over a period of approximately 11 years (including a one-year ramp up and one-year ramp down period). Restoration and reclamation will be implemented following production and will continue for approximately 5 years beyond the production period. Accordingly, the overall mine life of the Phoenix deposit is approximately 18 years from initiation of construction activities to completion of restoration and decommissioning/reclamation.

Development and Production Schedule

A Gantt style schedule was prepared to demonstrate the wellfield drilling, processing plant construction, and other site surface construction of the Phoenix project (Figure 1-4). Construction is expected to take approximately 2.5 years with the critical path to production being establishing the freeze wall to encapsulate the deposit and construction of the processing plant. The production period will be approximately 11 years. Total annual production is estimated at 6M U_3O_8 lbs per year with Figure 1-5 illustrating life of mine production.

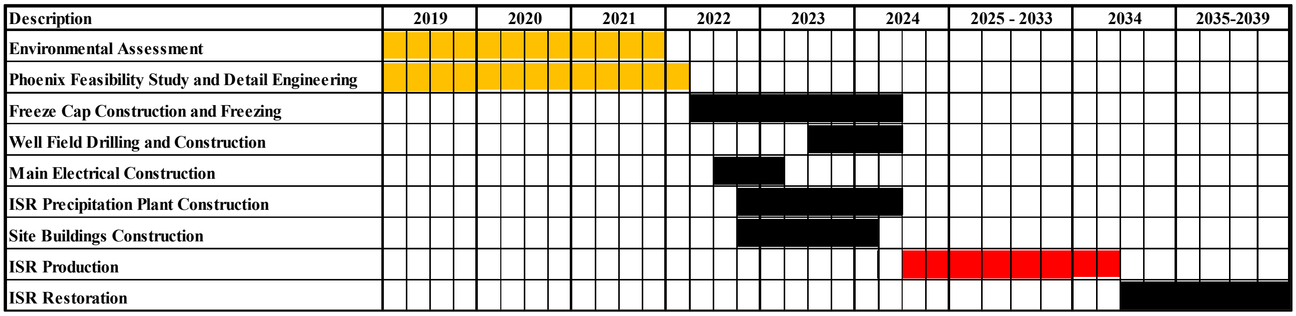


Figure 1-4. Phoenix Construction Schedule

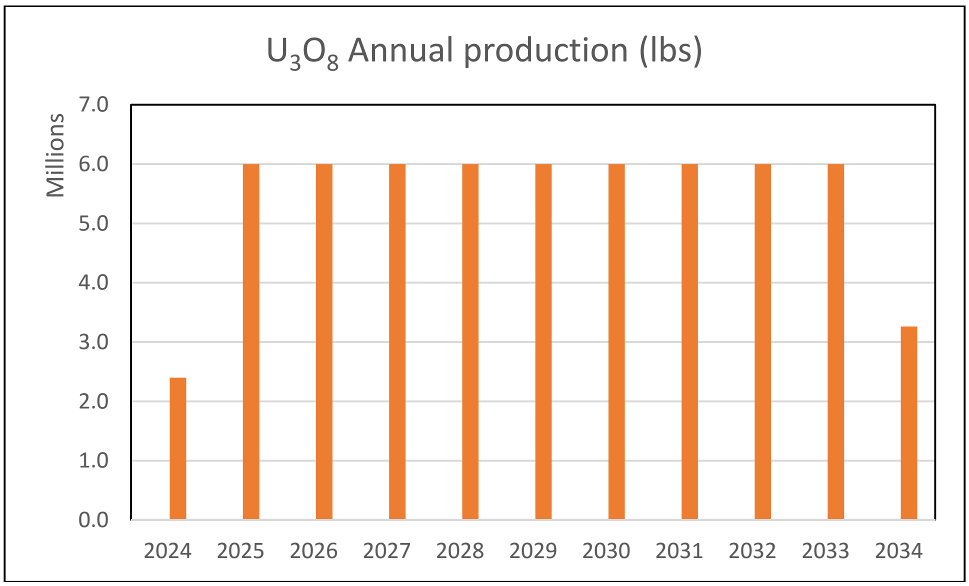


Figure 1-5. Phoenix Production Profile

1.2.9 Gryphon Mining

The PFS mine plan allows for Gryphon to be accessed via two shafts from surface, the production shaft (full-service, 5 m diameter, 550 m deep) and the ventilation shaft (4.5 m diameter, 500 m deep), to support underground development and production. Heated fresh air will be delivered via the production shaft, with return air exhausted up the ventilation shaft. An emergency hoist/conveyance will be installed in the ventilation shaft. Figure 1-6 shows an isometric view of the Gryphon mine.

The Gryphon resource extends from -2 MASL elevation to the -306 MASL elevation. A minimum 25 m permanent pillar will be left below the unconformity, with the first underground longhole mining level located at the 567 Level or -32 MASL elevation.

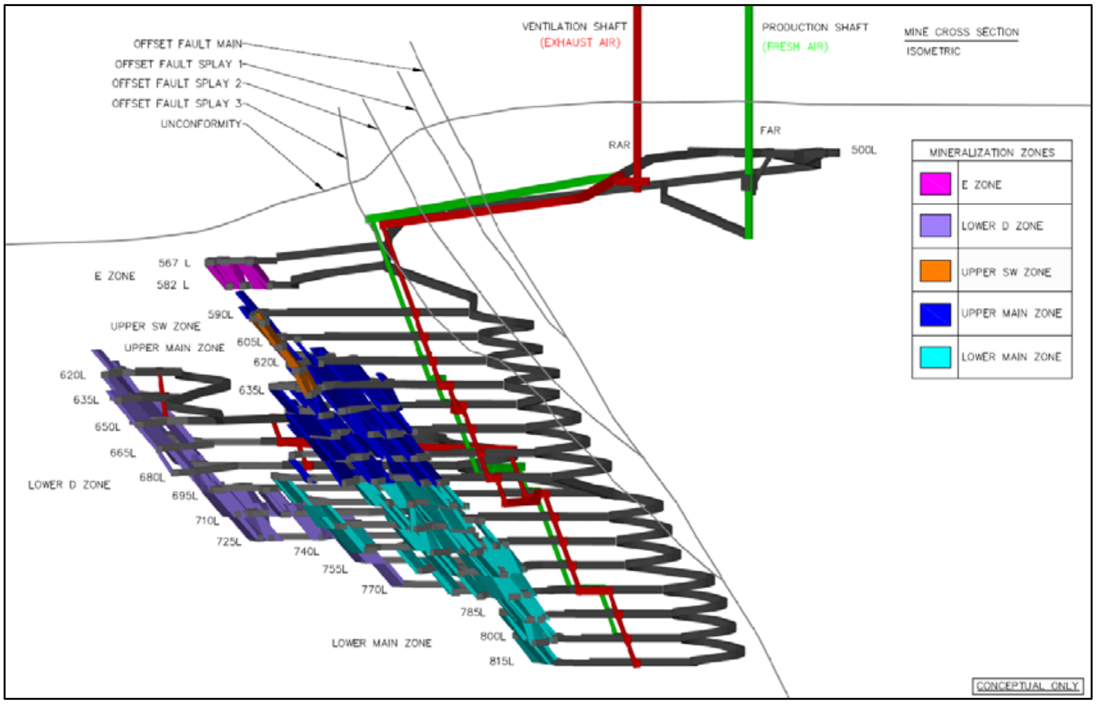


Figure 1-6. Isometric View – Gryphon 3D Mine Model

Access from the production shaft to the mine workings will be via a single ramp (4.5 m wide x 5.0 m high at a typical gradient of -15 percent) to be developed from the 500 Level (Shaft Station) to the 815 Level. The main haulage ramp will be located on the hangingwall side of the deposit and will be used to provide access for personnel and materials from the shaft to the mine workings, for movement of mining equipment from level to level within the mine, and for ore/waste haulage to the rockbreaker station near the shaft.

Each mining sublevel (15 m vertical intervals) is connected to an internal fresh air raise and an internal exhaust raise. The fresh air raise will serve as a second means of exit from the sublevels. The Gryphon deposit plunges to the northeast and the access ramp is designed to follow the plunge. Short sections of ventilation transfer drifts are included in the design to allow the ventilation raise systems to follow the plunge.

Ore will be truck hauled to a rockbreaker/grizzly station on the 500 Level near the production shaft and hoisted to surface. The underground mine is expected to produce approximately 605 tonnes per day of ore and an average of 330 tonnes per day of waste rock during the steady state operating period.

Underground production will be from the longhole stoping mining method, primarily longitudinal retreat. Longitudinal retreat involves accessing the resource from a central access point on each sublevel, developing ore sills (overcut and undercut drifts) along strike to the extents of each zone, and mining stopes from the extents back to the initial access. Mined stopes will be backfilled using a combination of rockfill, cemented rockfill, and hydraulic fill. The hydraulic fill will be directed to the empty stopes by means of boreholes and pipelines. Waste rock and cemented rockfill will be directed to the stopes via underground haulage trucks and LHDs.

The mine has been divided into five mining blocks, E Zone, Lower D, Upper and Lower Main, and Upper SW. Each mining block will be mined from the bottom up.

Stope overcut and undercut drifts will be driven at an average of 5.9 m wide x 4.0 m high and will include 100% shotcrete coverage and 150 mm of ballast on the floor to reduce the potential for radiation exposure.

The mine ventilation system will consist of two ventilation openings to surface. The production shaft will supply fresh air and the ventilation shaft will exhaust the air. Fresh air will split on the 500 Level down the haulage ramp and across to the 500 Level fresh air raise (FAR). Fresh air will transfer down a series of FARs and will be pulled off on the levels and into the active working areas. Rigid ducting will be installed close to the face and will pull the exhaust air from the face back to the return air raises (RAR), which are connected to the ventilation shaft on 500 Level. The ventilation shaft will also serve as a second egress from the mine.

The main mine dewatering system will consist of a clean water pumping system, using decanting sumps on the 500 Level to settle solids. The decanted water will be pumped to surface via piping in the ventilation shaft. A series of boreholes and sumps will stage the water to the 500 Level decanting station.

In the case of a major inflow of water, an emergency sump/pump station will be established on the 582 Level. Water staged from the underground workings will be directed to the unconsolidated waste-filled stopes above the 582 Level sump. The water will be collected at the bottom of the stopes (using the fill to filter out some of the suspended material) and pumped directly to the main pump station on 500 Level, bypassing the decanting sumps. Bulkheads will be constructed in the ramp at strategic locations to reduce the risk of the inflows overwhelming the dewatering system.

Development and Production Schedule

A Gantt style schedule was prepared to demonstrate the surface construction, shaft sinking, and the construction, development, and production phases of the Gryphon project (Figure 1-7). There will be slightly more than six years of pre-production period from the time the shaft construction starts in Q3 of 2023 until first production begins in Q1 2029. The production period will be approximately six years. Average annual production is ~600 tonnes per day equating to ~6M lbs per year and is driven to match expected mill capacity.

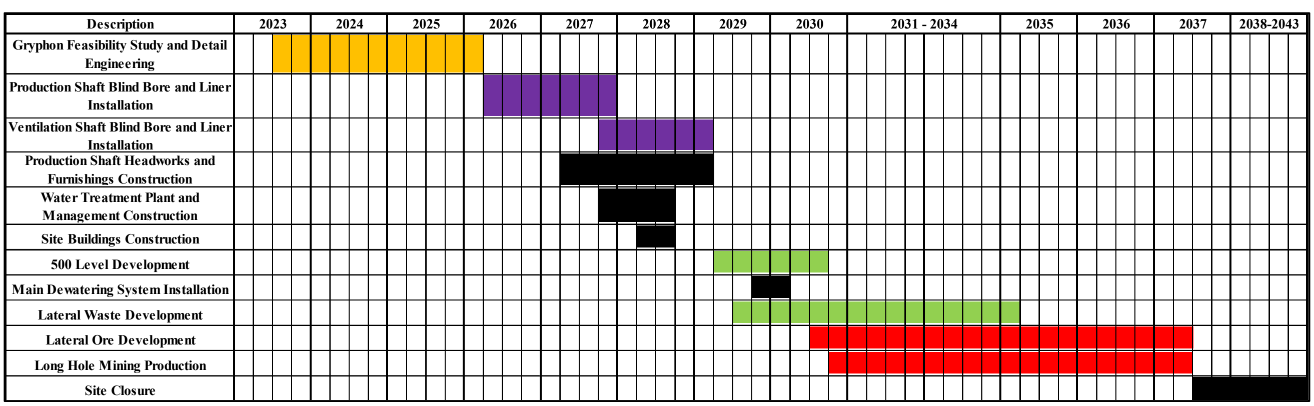


Figure 1-7. Underground Production and Development Schedule

Figure 1-8 shows the Gryphon mine summary production schedule. Estimated life of mine (LOM) production totals 1.26 Mt of mill feed at an average grade of 1.79% U_3O_8 containing 49.7 Mlbs of U_3O_8.

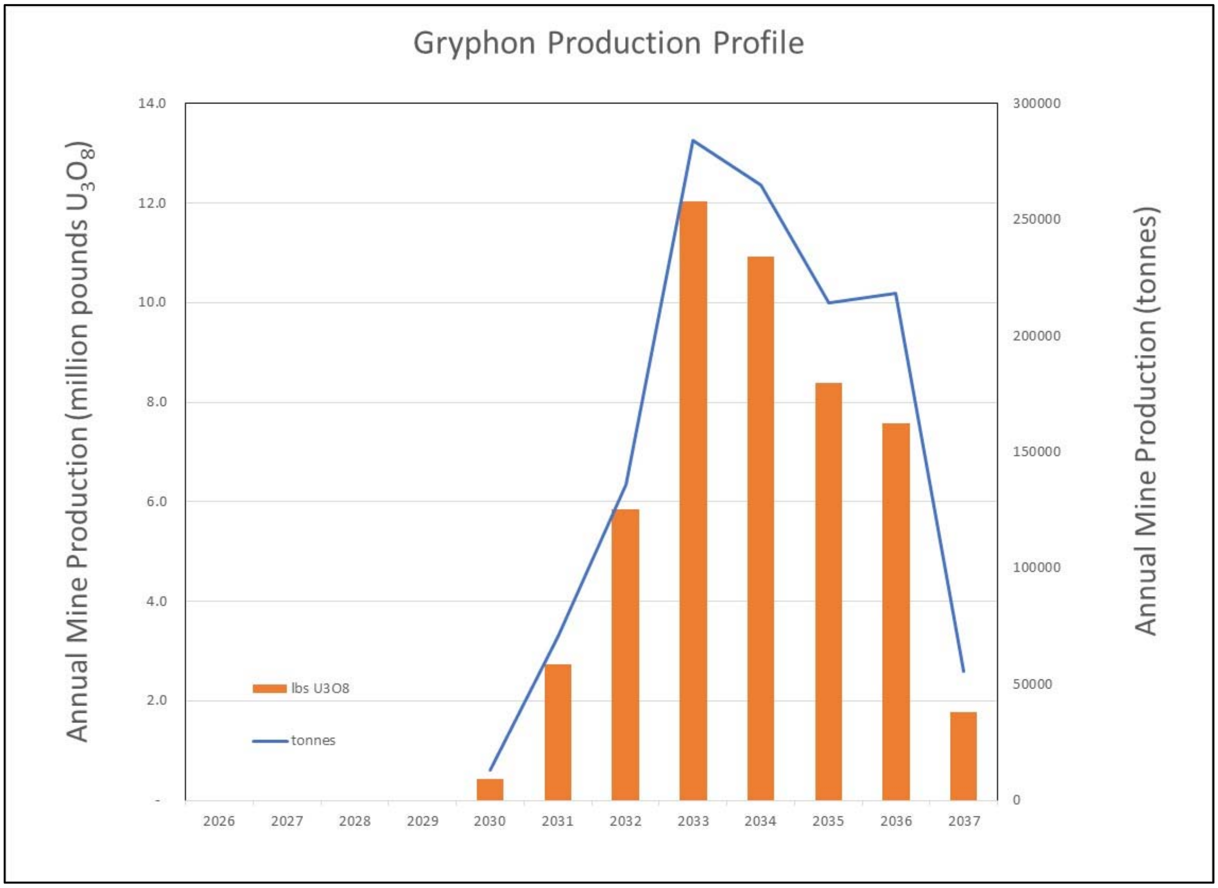


Figure 1-8. Gryphon Production Profile

Underground Mine Development

Mine development will be completed using traditional drill and blast mining methods. The development of the haulage ramp is expected to be on the critical path for the mine development. As development progresses to the production levels, additional headings will become available. The estimated LOM lateral development requirements are summarized in Table 1-5.

Table 1-5. LOM Lateral Development Requirements

Item	Quantity (m)
Capital	**9,658**
Ramp	3,576
Access/Infrastructure	6,082
Operating	**407**
Access	407
Ore Body Development (Silling)	**9,544**
Ore	3,829
Marginal	1,184
Waste	4,531
Total Lateral	**19,609**

Table 1-6 shows the estimated LOM vertical development requirements planned for Gryphon.

Table 1-6. LOM Vertical Development Requirements

Item	Quantity (m)
Vertical Development	
Production Shaft (5.0 m dia. finished)	550
Ventilation Shaft (4.5 m dia. finished)	500
Internal Ventilation Raise (4.0 m dia.)	317
Internal Ventilation Raise (3.0 m dia.)	285
Total Vertical	**1,652**

Waste Rock Broken and Backfill Requirements

Table 1-7 shows estimated LOM quantities of development waste rock broken and rock required for backfilling. Underground waste rock is considered potentially acid generating. For the Gryphon orebody, early waste development rock will be hoisted to surface as required and stored in a suitably designed containment area. A total of 320,000 tonnes of hoisted waste rock has been identified over years 1 through 4, inclusive, during the LOM (excludes waste produced during shaft sinking).

Table 1-7. Waste Rock Broken, Backfill, and Ballast Quantities

Item	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Waste Material Produced										
Marginal Ore	0	0	0	0	7,763	11,749	10,692	11,161	0	0
Special Waste	0	0	11,156	18,668	27,714	47,903	48,612	40,577	23,697	2,893
Other Waste	0	57,579	153,896	136,394	83,344	62,899	55,390	22,700	0	0
Consumed Underground										
Backfill	0	0	7,016	43,826	89,676	107,994	101,365	65,099	19,366	2,197
Ballast	0	5,606	16,739	15,936	13,240	14,558	13,330	9,338	4,331	696
Hoisted (waste only)	**0**	**51,972**	**141,297**	**95,300**	**15,905**	**0**	**0**	**0**	**0**	**0**

1.2.10 Phoenix Mineral Processing

The uranium bearing solution from the Phoenix wellfield will be directed to a self-contained processing facility located adjacent to the wellfield. The processing plant will house most of the process equipment in a 46,500 square foot pre-fabricated metal building. The proposed processing plant for the Phoenix ISR process will have four major circuits: impurities removal, yellowcake precipitation, dewatering/drying, and packaging. The processing plant will also have filtration systems, bulk chemical storage, process solution storage tanks, and a control room.

Uranium bearing solution from the wellfields will be pumped to the processing plant for beneficiation as described below:

- Impurities removal – Uranium liberated from underground in the Phoenix deposit will be routed to an iron/radium removal circuit, where the pH of the solution will be adjusted to allow the precipitation of iron hydroxide and other metals. Once the iron hydroxide has precipitated out of the solution, the solution will be routed to the primary yellowcake precipitation circuit.
- Yellowcake precipitation – The solution will be pH adjusted to optimal levels for uranium precipitation with sodium hydroxide, then yellowcake product will be precipitated with

hydrogen peroxide, using sodium hydroxide to maintain optimal pH. Following uranium precipitation into yellowcake slurry, the barren mining solution will be reconstituted to the proper acid level prior to being pumped back to the wellfield for reinjection.

- Yellowcake dewatering/drying and packaging – The precipitated yellowcake slurry will be transferred to a filter press, where excess liquid will be removed. Following a fresh water wash step that will further clean the yellowcake product, the resulting yellowcake will be transferred to the dryer, which will further reduce the moisture content, yielding the final dried, free-flowing product. Refined yellowcake will be packaged in 55-gallon drums.

For the PFS, a process flowsheet and mass balance of the process has been completed, as well as a general arrangement of the plant itself, including all major mechanical equipment (Figure 1-9).

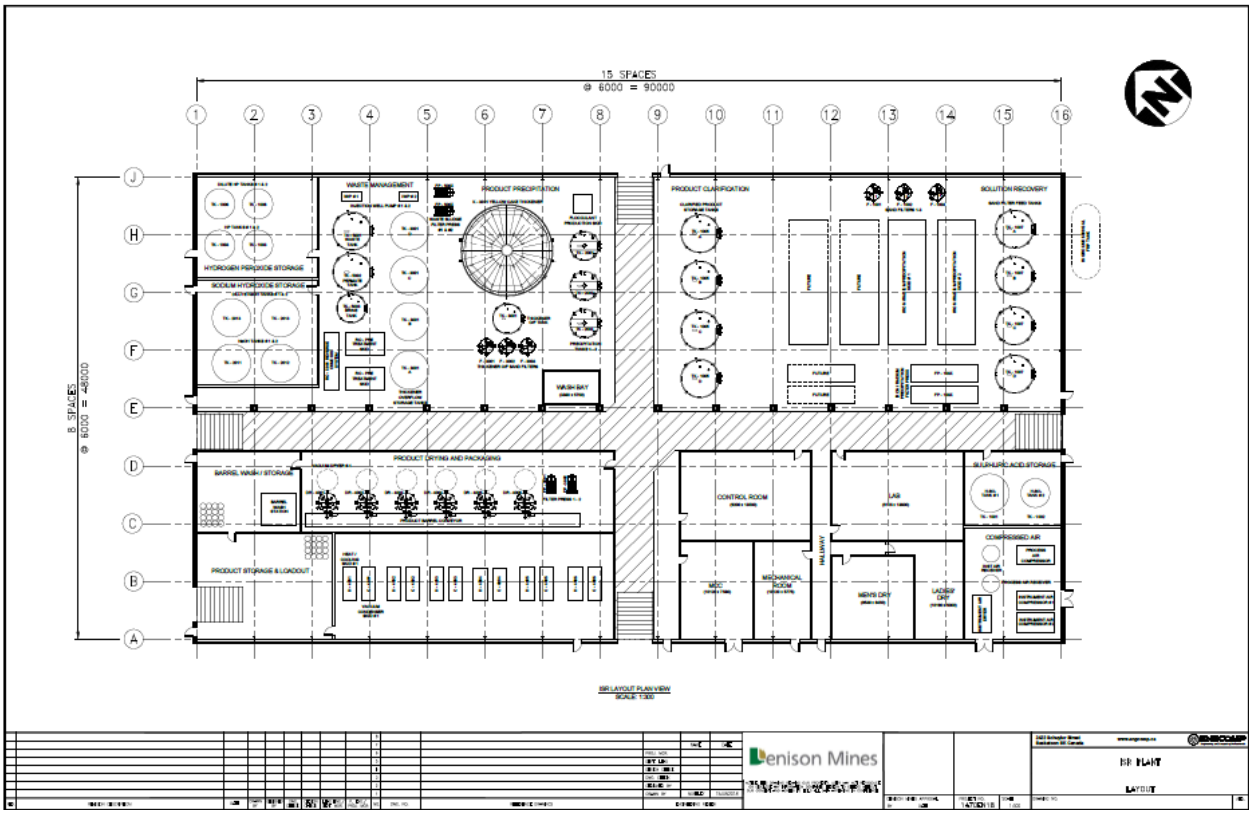


Figure 1-9. General Arrangement of the ISR Plant

The different types of chemical reagents will be stored, used, and managed to ensure worker and environmental safety in accordance with standards developed by regulatory agencies and vendors. Taken together, it is expected to achieve 98.5% recovery of metal delivered to the plant.

1.2.11 Gryphon Mineral Processing

The PFS plan assumes that Gryphon ore will be transported to the McClean Lake mill for processing. The mill is currently processing material from the Cigar Lake mine; however, it has additional licenced processing capacity to a total annual production of up to 24,000,000 lbs U_3O_8.

The mine plan for Wheeler River aligns well with known available capacity at the McClean Lake mill. Proposed Gryphon deposit production scenarios do not exceed McClean Lake's currently licenced capacity of 24 Mlbs/a U_3O_8 production, given certain assumptions regarding future production from the Cigar Lake mine. Gryphon ore is expected to be milled in parallel to Cigar Lake Phase 2 production. Cigar Lake Phase 2 production, which is not fully defined, is assumed to be 15 M lbs/a U_3O_8, allowing for Gryphon ore processing at 9 M lbs/a U_3O_8.

The results of the metallurgical test work program completed for the PFS indicate that the Gryphon deposit is amenable to recovery utilizing the existing McClean Lake mill flowsheet. Moreover, the deposit is amenable for processing under similar conditions to those currently used in the McClean Lake mill. Overall process recovery based on metallurgical test work conducted to date has been estimated at 98.4% (co-processed with Cigar Lake) for Gryphon ore. Uranium recoveries of 98.2% have been applied in the financial modelling for Gryphon.

Processing the Gryphon deposit will require certain modifications to the McClean Lake mill. These modifications include expansion of the leaching circuit, the addition of a filtration system to complement the Counter Current Decantation (CCD) circuit capacity, the installation of an additional tailings thickener, and expansion of the acid plant. Various other upgrades will also be required throughout the mill to permit production at the full 24 M lbs / a U_3O_8 licenced capacity, as described in Section 17.

1.2.12 Surface Infrastructure

Main land access to the sites is from Saskatchewan Highway 914, the existing haul road between the McArthur River mine and the Key Lake processing plant.

Site preparation earthworks will first be undertaken at the Phoenix site including areas for the wellfield and freeze hole drilling, precipitation plant and designated camp and services areas.

A 1,600 m long airstrip is planned to be positioned in a natural, relatively flat valley to the northeast of the Phoenix deposit.

Figure 1-10 is a plan view of the Wheeler River project, showing the Gryphon and Phoenix deposits relative to the existing Wheeler River camp and Provincial Highway 914. The Gryphon deposit is roughly 3 km NW of the Phoenix deposit.

With the exception of the airstrip, all common facilities and services will be provided at the Phoenix site, as it will be developed first. Gryphon personnel will be housed at the camp facilities on the Phoenix site.

Production from the Gryphon site will be trucked to the existing McClean Lake mill to the northeast, via existing Provincial Highway 914, including 51 km of new road between the McArthur River mine and the Cigar Lake mine.

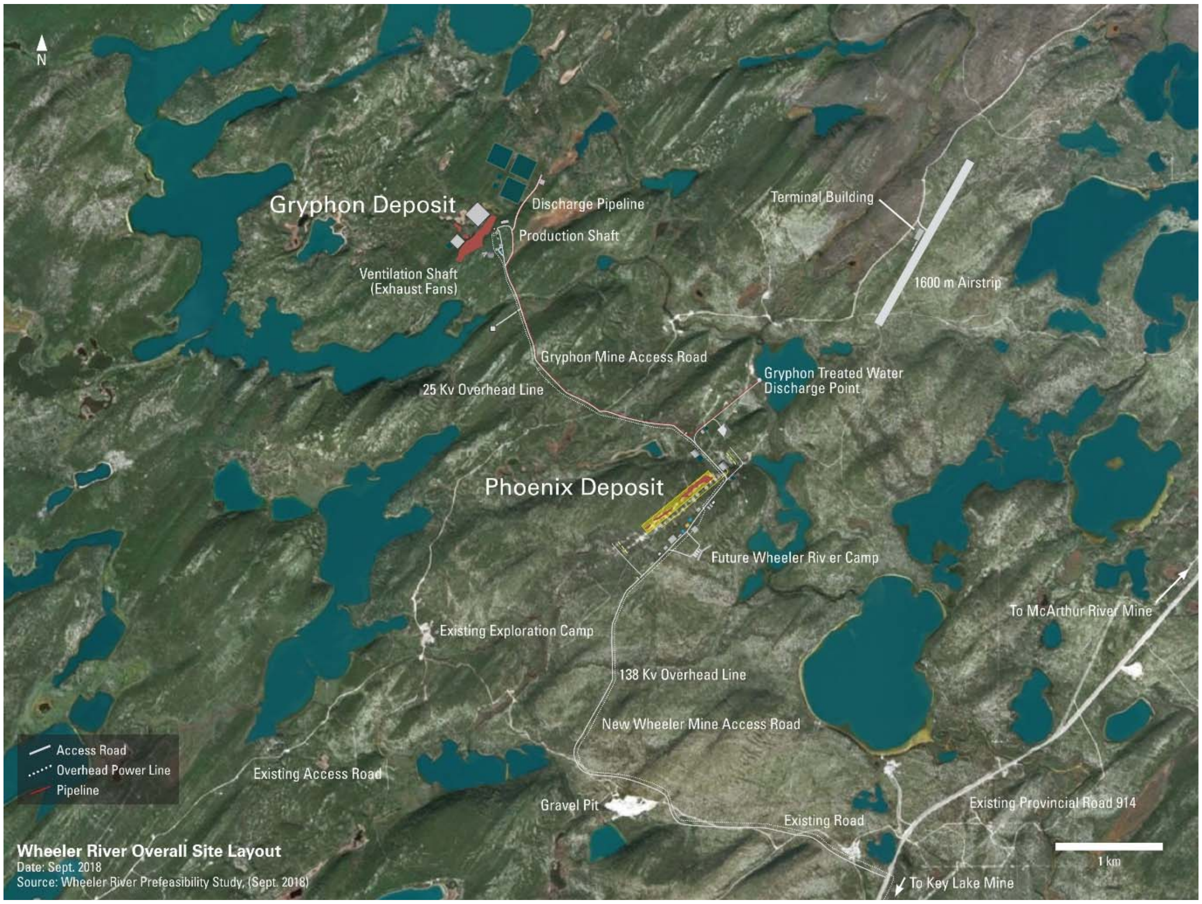


Figure 1-10. Wheeler River Project Site Showing Phoenix and Gryphon Deposits

Figure 1-11 is a conceptual layout of the plan view of the Phoenix operation's surface facilities, showing the relative scale and nominal footprint size of major infrastructure items, including:

- Area allocation over the defined deposit for an in situ leaching wellfield option (90 m x 800 m);
- ISR processing plant (90 m x 48 m);
- Operations centre (61 m x 41 m), including men's and women's dry facilities, 3-bay maintenance shop, welding bay, warehouse, emergency response vehicle storage, mine rescue and emergency response office, laboratory, nurse's station, training room, offices (administration, maintenance, and supply chain), meeting rooms, lunch room, and radiation monitoring room;
- 150-person camp with kitchen and laundry facilities;
- Personal-vehicle parking;
- Main electrical substation (50 m x 50 m);
- North and south gatehouses;
- Outdoor and covered storage (15 m x 30 m);
- Wash bay and scanning facility;
- 30 m long, 80 tonne weigh scale;
- Potable water treatment facility;
- Fuel storage and dispensing facility (gas and diesel);

- Fire water tank and pumphouse;
- Two bullet propane tank farm;
- Sewage treatment facility;
- Incinerator;
- Backfill plant with storage facility;
- Outdoor fenced hazardous storage area (30 m x 30 m);
- Fenced landfill area (90 m x 90 m);
- Water discharge station;
- Special waste storage (46 m x 46 m, 3,200 cubic meter capacity); and
- Clean waste rock storage (60 m x 60 m, 7,100 cubic meter capacity).

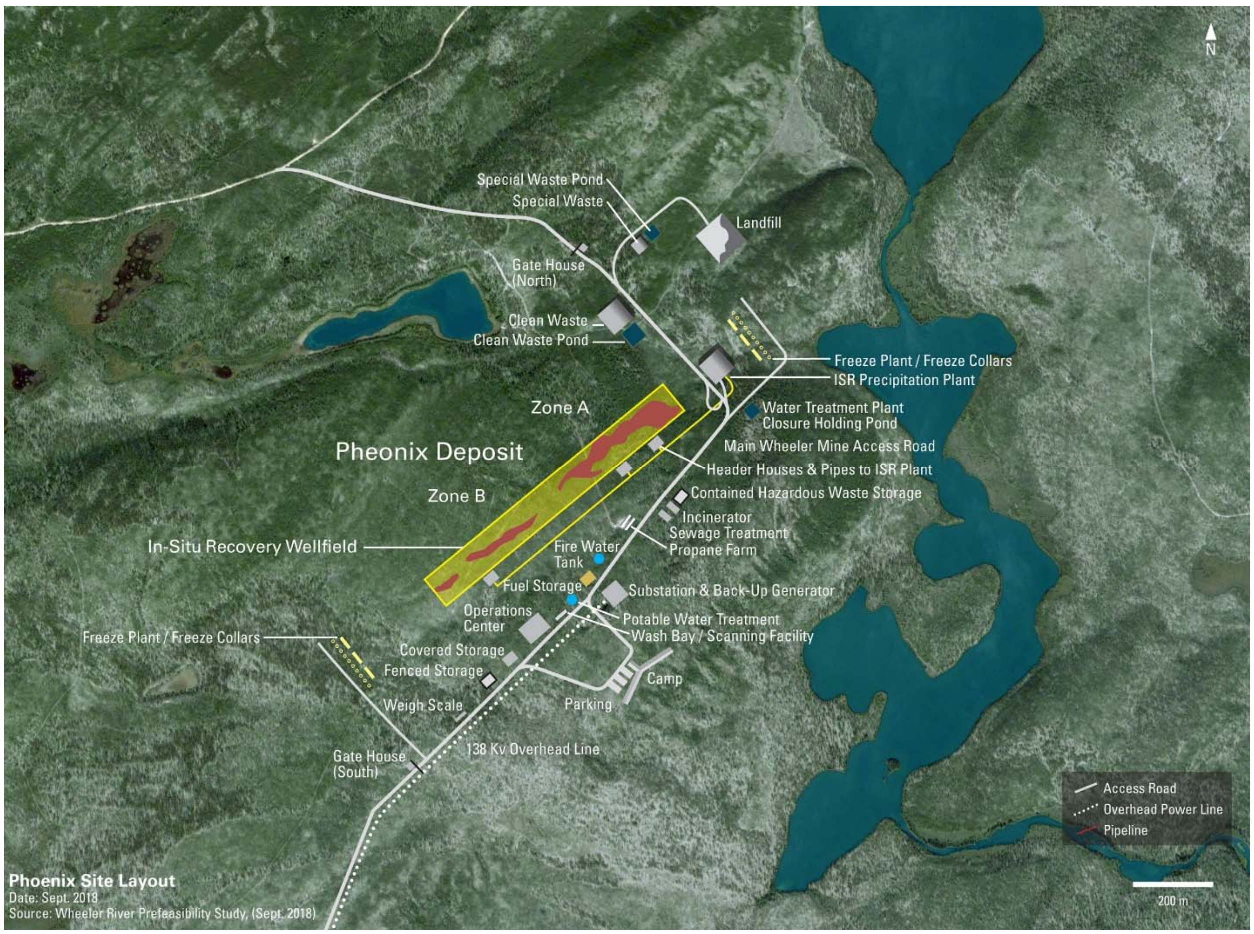


Figure 1-11. Phoenix Site Conceptual Layout

Figure 1-12 is a conceptual layout of the plan view of the Gryphon site surface facilities, showing the relative scale and nominal footprint size of major infrastructure items, including:

- Headframe and collar house for the 5.5 m diameter production shaft;
- Hoist house and production/service hoist for production shaft;
- Hoist house and hoist for auxiliary cage in production shaft;
- Headframe, hoist house, and hoist for auxiliary cage in ventilation shaft (secondary egress);

- Fresh air ventilation fans and propane fired air heaters with ventilation plenum at headframe;
- Surface ore storage (55 m x 55 m, 3,000 cubic meter capacity);
- Clean waste rock storage (104 m x 104 m, 45,000 cubic meter capacity);
- ARD/ML waste storage (180 m x 180 m, 210,000 cubic meter capacity);
- Main south gatehouse;
- Operations centre (20 m x 20 m), including space for mine rescue equipment and facilities, and a number of small offices;
- Backfill plant (20 m x 20 m) and 60 m diameter backfill aggregate pile;
- Electrical room module (20 m x 6 m);
- Explosives magazine (50 m x 50 m), with a designated security gate on its access road;
- Outdoor fenced storage (15 m x 30 m);
- Fuel storage and dispensing facility (gas and diesel);
- Fire water tank and pumphouse;
- Three bullet propane tank farm (close proximity to headframe);
- Water treatment plant (40 m x 40 m);
- Two water treatment plant holding ponds (each at 48 m x 48 m, 7,500 cubic meter capacity); and
- Three water management ponds (each at 200 m x 200 m, 270,000 cubic meter capacity).

Figure 1-12. Gryphon Site Conceptual Layout

1.2.13 Environmental and Permitting and Community Consultations

There are no recognized environmental fatal flaws associated with this project.

The Phoenix project has the potential to be one of the most environmentally friendly uranium mining projects in the world.

- The ISR approach produces no tailings products.
- The closed loop system of the processing plant eliminates any major sources of water to be discharged to the environment. Due to evaporation and moisture content of the yellowcake product, the processing plant may require small volumes of make-up water.
- Minimal volumes of surface run-off will be captured, treated, and used as make-up water in the processing plant or re-injected underground.
- Low to near zero carbon emissions due to the lack of heavy equipment and provision of power from the provincial power grid.
- Small volumes of waste products from the iron precipitation circuits will be temporarily stored on surface and disposed of in the underground stopes at Gryphon.

At Gryphon, the most significant environmental concern associated with the project will be the management of treated mine effluent. Investigations into environmentally acceptable discharge locations has identified suitable sites nearby that will minimize any impacts from treated effluent discharge. Other waste products, such as potentially acid generating waste rock or low-grade waste products, will be used underground as backfill on a priority bases where possible. Otherwise, such materials will be stored in approved facilities designed for safe closure and decommissioning. Future studies will evaluate the potential for 100% underground storage to elimination the need for surface facilities.

Denison believes all potential environmental impacts can be successfully mitigated through the implementation of industry best practices.

The project will require completion of Federal and Provincial environmental assessments. It is estimated the assessments will require approximately 24 to 36 months to complete following the submission of a detailed project description.

Denison recognizes the importance of early stakeholder engagement and has been developing relationships with key stakeholders since 2016. A detailed stakeholder engagement program was developed with highlights to date including:

- Establishment and financial support for training and educational programs for residents of stakeholder communities;
- Establishment of employment opportunities for residents of stakeholder communities;
- Procurement of goods and services from northern based suppliers to support continued exploration activities;
- Engagement of stakeholder communities to provide input in aspects of the project design including access road routing, treated water discharge location, selection of mining method for Phoenix; and
- Generally supportive responses from communities encouraging the development of the project and requests for negotiation of formal support agreements.

1.2.14 Capital and Operating Costs

Capital and operating cost estimates were developed to support the PFS of the Gryphon and Phoenix deposits. The estimates address the initial capital, sustaining capital and operating costs required to engineering, procure, construct, commissioning, start-up and operate the mines, ISR precipitation plant and related infrastructure at the Wheeler River site and upgrades at the McClean Lake mill. Estimates were completed to AACE class four level with an accuracy of -15% to -30% on the low side and +20% to +50% on the high side.

The Wheeler River project total capital cost is estimated at approximately $1.13 billion, comprised of $322.5 million of initial pre-production capital for the Phoenix operation and $623 million of initial pre-production capital for the Gryphon operation as outlined in Table 1-8.

Table 1-8. Capital Cost Summary

Wheeler River Capital Cost (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Phoenix	$ 322,539	$ 103,411	$ 425,950
Gryphon	$ 623,120	$ 82,743	$ 705,862
Sub Total	**$ 945,659**	**$ 186,154**	**$ 1,131,813**

The capital costs for the ISR mining of the Phoenix deposit are categorized in Table 1-9.

Table 1-9. Phoenix Capital Cost Summary

Phoenix Capital Cost Details (1,000's)			
Direct Capital Costs	**Initial**	**Sustaining**	**Total**
Wellfield	$ 63,674	$ 35,402	$ 99,076
ISR Precipitation Plant	$ 50,935	$ 4,606	$ 55,541
Water Treatment Plant	$ 1,268	$ 18,676	$ 19,944
Surface Facilities	$ 22,325	$ 49	$ 22,374
Utilities	$ 6,538	$ 803	$ 7,341
Electrical	$ 18,834	$ -	$ 18,834
Civil & Earthworks	$ 44,309	$ 1,331	$ 45,640
Offsite Infrastructure	$ 7,950	$ -	$ 7,950
Decommissioning	$ -	$ 27,454	$ 27,454
Total Direct Costs	**$ 215,834**	**$ 88,321**	**$ 304,155**
Indirect Costs	$ 28,288	$ 5,669	$ 33,957
Owner's Costs	$ 14,227		$ 14,227
Contingency Costs	$ 64,190	$ 9,421	$ 73,611
Total Costs	**$ 322,539**	**$ 103,411**	**$ 425,950**

The capital costs for the underground mining of the Gryphon deposit are shown in Table 1-10.

Table 1-10. Gryphon Capital Cost Summary

Gryphon Capital Cost Details (1,000's)			
Direct Capital Costs	**Initial**	**Sustaining**	**Total**
Shafts	$ 131,522	$ -	$ 131,522
Surface Facilities	$ 46,932	$ 6,074	$ 53,006

Underground	$ 49,518	$ 68,842	$ 118,360
Utilities	$ 3,946	$ 263	$ 4,209
Electrical	$ 3,613	$ -	$ 3,613
Civil & Earthworks	$ 11,791	$ 483	$ 12,274
McClean Mill Upgrades	$ 49,920	$ -	$ 49,920
Offsite Infrastructure	$ 32,392	$ -	$ 32,392
Decommissioning	$ -	$ 1,575	$ 1,575
Total Direct Costs	**$ 329,634**	**$ 77,236**	**$ 406,871**
Indirect Costs	$ 142,015	$ 5,112	$ 147,127
Other (Owner's) Costs	$ 28,143		$ 28,143
Contingency Costs	$ 123,328	$ 394	$ 123,722
Total Costs	**$ 623,120**	**$ 82,743**	**$ 705,863**

Operating costs are estimated for the 14-year mine production period from July 1, 2024 through to March 31, 2037. Phoenix mine production is scheduled from July 1, 2024 to June 30, 2034 and Gryphon mine production is scheduled from September 1, 2030 to March 31, 2037. Table 1-11 presents a summary of the Wheeler River prefeasibility level operating cost estimates and total estimated sales.

Table 1-11. Wheeler River Operating Cost Summary

Cost Area	Phoenix		Gryphon		Total Cost
	$000's	$/lb U_3O_8	$000's	$/lb U_3O_8	$000's
Mining	$ 44,020	$ 0.75	$ 266,202	$ 5.45	$ 310,222
Milling	$ 115,577	$ 1.97	$ 412,621	$ 8.45	$ 528,198
Transport to Convertor	$ 12,341	$ 0.21	$ 10,252	$ 0.21	$ 22,593
Site Support / Administration	$ 82,264	$ 1.40	$ 53,346	$ 1.09	$ 135,610
Total	**$ 254,202**	**$ 4.33**	**$ 742,421**	**$ 15.21**	**$ 996,623**
Total USD		**$ 3.33**		**$ 11.70**	
U_30_8 Sales - lbs in 000's	**58,767**		**48,817**		

1.2.15 Indicative Economic Results

The WRJV, which owns the Wheeler River project, is a joint venture and is not itself a taxable entity. Instead each joint venture partner reports its share of the joint venture operations in its own tax return. As each JV partner has a unique tax profile, the Wheeler River project has been evaluated using two different cash flow model approaches:

- A pre-tax discounted cash flow model which shows the economics of the project on a 100% basis and excludes tax specific items related to Canadian income taxes and Saskatchewan profit-based royalties, each of which will vary depending on each joint venture participants unique circumstances; and
- A post-tax discounted cash flow model, specific to Denison (Section 22.6), which shows the economics of the project based on Denison's ownership interest in the project and includes tax specific items related to Canadian income taxes and Saskatchewan profit-based royalties and other non-tax related items which are unique and applicable to Denison's economic interest in the project

Inputs into both pre-tax and post-tax models include:

- Discount rate of 8%;
- Estimated metallurgical process uranium recoveries of 98.5% and 98.2% for Phoenix and Gryphon mill feeds, respectively;
- Project capital and operating costs as provided in Section 21;
- Project schedule as outlined in Figure 1-13;
- Mine production as outlined in Section 16 and depicted in Figure 1-14;

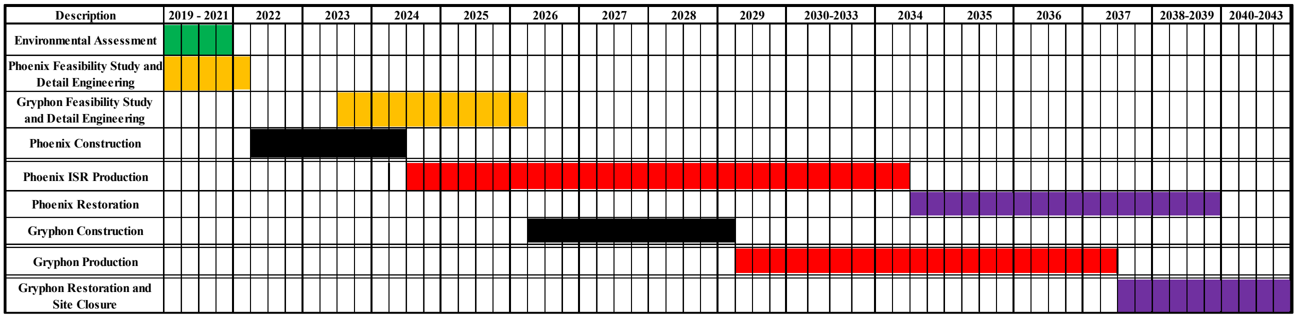


Figure 1-13. Wheeler River Project Schedule

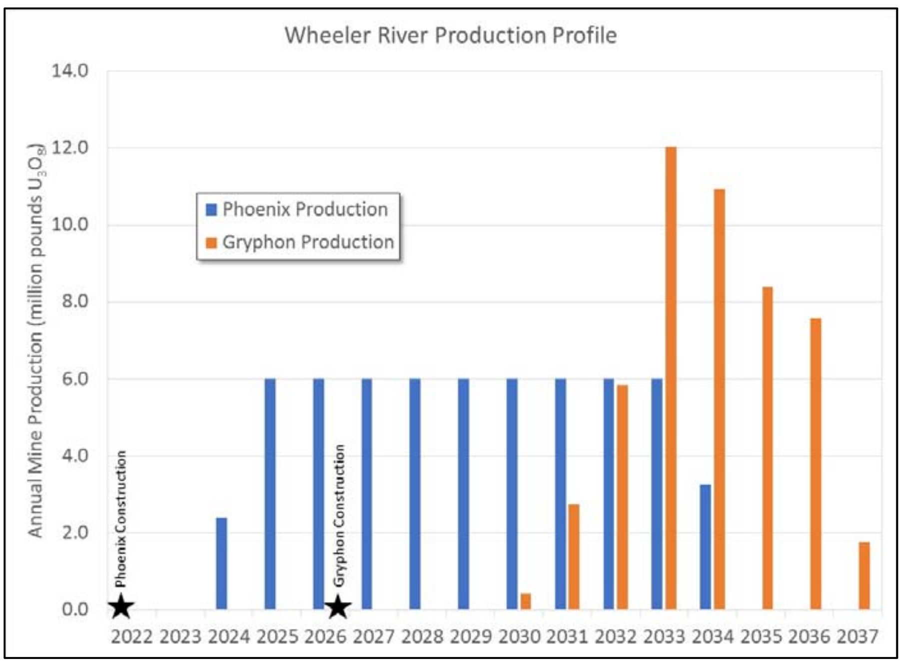


Figure 1-14. Wheeler River Production Schedule

- Base case uranium pricing scenario as follows: a) Phoenix – based on UxC's Q3-2018 Uranium Market Report Composite Midpoint spot price projection, in constant (uninflated) 2018 dollars, ranging from USD$29.48 to USD$45.14 per pound U_3O_8 during the Phoenix mine production period, translated to CAD using an exchange rate of 1.30 CAD/USD; and b) Gryphon – based on a fixed price of USD$50.00 per pound during the Gryphon mine production period, translated to CAD using an exchange rate of 1.30 CAD/USD;
- Saskatchewan revenue-based royalties and surcharges applicable to uranium revenue, as follows: a) a basic royalty of 5.0% of uranium revenue; b) a resource credit of 0.75% of uranium revenue (which partially offsets the basic royalty); and c) a resource surcharge of

3.0% of the value of uranium revenue. For the purposes of these calculations, revenue has been computed as gross uranium revenue less transportation costs to the convertor; and
- No inflation or escalation of revenue or costs have been incorporated.

The Wheeler River project pre-tax indicative base case economic results are illustrated in Table 1-12:

Table 1-12. Pre-tax Economic Results

Pre-Tax Results	NPV	IRR	Payback
Base Case (UxC spot price)	$1,308 million	38.7%	~ 24 Months
High Case ($65 / lb)	$2,587 million	67.4%	~ 11 Months

(1) NPV and IRR are calculated to the start of pre-production activities for the Phoenix operation in 2021;
(2) Payback period is stated as number of months to pay back from start of uranium production

A post-tax economic assessment includes similar inputs as the pre-tax assessment with the following modifications:

- Denison's share of project development costs is included in the project's capital costs along with their impact on Denison's estimated tax pools;
- The impact of the Saskatchewan Profit Royalty as estimated for Denison is included;
- Denison's expected provincial and federal income taxes payable are included; and
- Denison's recovery of toll milling fees paid to the MLJV (22.5% owned by Denison) by the WRJV for the toll milling of Gryphon ores.

The Wheeler River project post-tax indicative economic results as detailed in section 22 are as follows in Table 1-13.

Table 1-13. Post-tax Economic Results

Post-Tax Results	NPV	IRR	Denison Ownership Interest
Base Case (UxC spot price)	$506.4 million	31.7%	63.3%
Base Case (UxC spot price)	$755 million	32.7%	90%
High Case ($65 / lb)	$1,006.2 million	53.8%	63.3%
High Case ($65 / lb)	$1,483.8 million	55.7%	90%

(1) NPV and IRR are calculated to the start of pre-production activities for the Phoenix operation in 2021;
(2) Payback period is stated as number of months to pay back from start of uranium production

More detailed comparison tables for pre-tax and post-tax results are shown in Tables 1-14 (Base Case) and 1-15 (High Case). Denison's ownership interest was 63.3% as of the effective date of the report (September 24, 2018) but was increased to 90% as a result of the transaction referenced in section 1.1

Table 1-14. Base Case Cash Flow Model – Pre-Tax vs Post Tax Comparison

Item Description	Base Case Pre-Tax Summary	Base Case Post Tax Summary	Base Case Post Tax Summary
CAD$ millions			(Pro-Forma)
Project Percentage	**100.0%**	**63.30%**	**90.00%**
Gross Uranium Revenue	**6,142.6**	**3,888.3**	**5,528.3**
Toll Milling Fees	Excl.	4.8	1.3
Operating Costs	(996.6)	(630.9)	(897.0)
Operating Costs – Toll Milling Credits	Excl.	8.2	11.7
Saskatchewan Revenue Royalties, Surcharges	(443.7)	(280.9)	(399.3)
Operating Cash Flow	**4,702.3**	**2,989.5**	**4,245.0**
Capital Costs	(1,131.8)	(716.4)	(1,018.6)
Capital Costs – Project Development	Excl.	(13.5)	(19.2)
Contribution before Taxes	**3,570.5**	**2,259.6**	**3,207.2**
Saskatchewan Profit Royalties	Excl.	(341.0)	(421.9)
Canadian Federal / Provincial Income Taxes	Excl.	(497.2)	(685.5)
Net Contribution	**3,570.5**	**1,421.4**	**2,099.8**
NPV (8%) at fiscal 2021	1,308.3	506.4	755.9
IRR	38.7%	31.7%	32.7%

Net contribution represents the undiscounted cash flow impact applicable to the Wheeler project.

Table 1-15. High Case Cash Flow Model – Pre-tax vs Post Tax Comparison

Item Description	High Case Pre-Tax Summary	High Case Post Tax Summary	High Case Post Tax Summary
CAD$ millions			(Pro-Forma)
Project Percentage	**100.0%**	**63.30%**	**90.00%**
Gross Uranium Revenue	**9,090.9**	**5,754.5**	**8,181.8**
Toll Milling Fees	Excl.	4.8	1.3
Operating Costs	(996.6)	(630.9)	(897.0)
Operating Costs – Toll Milling Credits	Excl.	8.2	11.7
Saskatchewan Revenue Royalties, Surcharges	(657.5)	(416.2)	(591.7)
Operating Cash Flow	**7,436.8**	**4,720.4**	**6,706.1**
Capital Costs	(1,131.8)	(716.4)	(1,018.6)
Capital Costs – Project Development	Excl.	(13.5)	(19.2)
Contribution before Taxes	**6,305.0**	**3,990.5**	**5,668.3**
Saskatchewan Profit Royalties	Excl.	(617.5)	(776.8)
Canadian Federal / Provincial Income Taxes	Excl.	(889.9)	(1,254.2)
Net Contribution	**6,305.0**	**2,483.1**	**3,637.3**
NPV (8%) at fiscal 2021	2,587.7	1,006.2	1,483.8
IRR	67.4%	53.8%	55.7%

Net contribution represents the undiscounted cash flow impact applicable to the Wheeler project. Net Present Value (NPV) and Internal Rate of Return (IRR) are calculated to the start of pre-production construction activities in 2021.

1.2.16 Risks and Opportunities

During the completion of the PFS, several risk and opportunity assessment sessions were held with appropriate Qualified Persons, consulting engineers, and Denison personnel. All elements of the project were subjected to this structured approach.

Ranking action plans (opportunities) and mitigations plans (risks) were identified and developed to address the factor in future work. The action and mitigation plans, and associated budgets will be incorporated into the next steps in project development.

Project Opportunities

1. Increase in total production at Phoenix: ISR metallurgical testing achieved over 90% resource extraction with testing halted prematurely while lixiviant solution concentrations remained above 5 g/l. With economic cut-off grades for ISR operations typically far below this level, there may be potential for additional resource extraction above the 85% assumed in the PFS. In addition, the current mineral resource estimate at Phoenix is based on a 0.8% U_3O_8 cut-off grade based on previously assumed conventional mining practices. Using an ISR extraction method is significantly less costly and as a result, a lower cut-off grade may be appropriate. Furthermore, there are other areas at Phoenix (Zone C, Zone D, mineralized zones between Zone A and B) which are known to contain mineralization but have not been drilled or quantified to a mineral resource level of confidence due to the negligible impact expected to have on an underground mining operation. However, this mineralization may prove to be attractive under an ISR extraction technique.

2. Phoenix annual production increase: The current production plan is based on the assumption that 10 recovery wells will be producing 10 g/L solution for an overall production level of 6 M lbs/yr. Mineral processing test work has demonstrated uranium bearing solution average grades of 12 g/L and grades as high as 27 g/L. Additionally, there is a total of 94 recovery wells planned for the Phoenix Zone A and Zone B mineral resources. Based on the above information, should the solution grades be higher than the assumptions used in this study, or if additional recovery wells are operated, there is potential for production levels to increase above 6 M lbs/yr.

3. Wellfield drilling: The project design assumed contractor drilling of the wellfields using a reverse circulation drill rig for all aspects of the well drilling and installation. In the future, evaluation of owner supplied equipment and/or utilization of multiple drill rigs for installation could significantly reduce costs and schedule for wellfield construction.

4. Gryphon ore sorting: During the metallurgical testing, it was determined that radiometric sorting of the ore may be possible. This may significantly reduce the quantity of ore to be transported to the mill for processing and could result in material reductions in transportation costs, milling costs, tailings storage requirements, etc.

5. Processing Gryphon ores at Wheeler River: Due to the mining sequence, the Phoenix ISR plant will be constructed and operating well before Gryphon is developed. The current plan for processing of Gryphon ores is to ship them to McClean Lake, which requires significant capital and operating costs, including construction of a 50 km extension of Highway 914, McClean Lake mill upgrades, toll milling and tailings storage fees, mill operating costs, etc. Future work will evaluate the merit of building additional front end processing circuits to the ISR processing plant (i.e. grinding and leach circuit) to process Gryphon ores on site and potentially reduce capital and operating costs.

6. Rare earth metals: Both the Phoenix and Gryphon deposits contain levels of rare earth elements. The PFS focused on the production of uranium and did not consider recovery of other valuable elements. However, from metallurgical test work, it is known that rare earth metals are recovered in the leaching process. Future work will evaluate the recovery of other metals from mining solution, which could have a material impact to revenues from by-products.

7. Gryphon sill development: Current ground support designs in the ore sills require bolts and screen. Later in the development design stage, shotcrete was included in the design for radiological exposure reduction. In future project evaluations, optimization of ground support may allow for the elimination of ground support duplication, potentially saving significant time and money during sill development phases.

8. Phoenix freeze wall spacing: Currently, the schedule allows for 5 months of freeze hole drilling followed by 14 months to establish the freeze wall around the deposit. There is potential to complete these activities simultaneously and thereby reduce total duration of freeze activities. With additional time requirements, there is potential to reduce capital costs by increasing freeze hole spacing to 6 or 7 m (reducing the total meters of drilling) and/or purchasing smaller freeze plants (requires longer time to freeze).

Project Risks

1. Regulatory approvals: The design assumes that Federal and Provincial project approvals will be granted in certain time periods and without major impact to the project. No assurance can be provided that such approvals will be received or that they will be received in the time period assumed.

2. Gryphon toll milling agreements: Processing of Gryphon ores would require a toll milling agreement between the WRJV and the MLJV. No assurance can be provided that such an agreement will be completed or that the terms of the agreement, toll milling capacities, and /or associated fees would be at the levels assumed within this PFS.

3. Gryphon toll milling: Impact of future Cigar Lake grades on process design. Data was requested from Denison on future production grades from Cigar Lake, however Hatch was informed that this data was not available. Lower grades (higher tonnage) may increase the type and/or size of equipment required to process both feeds. However, if lower grades do occur, additional costs may not solely rest with Denison, as this would need to be reviewed according to the terms of the existing Cigar Lake toll milling JV agreement. The quality of future Cigar Lake feed grades could have a material impact on the results of Hatch's analysis.

4. Gryphon Toll milling: Test work has not yet been completed on comingled samples (Cigar Lake ore and Wheeler River ore). There is potential for impact on recovery (i.e. if a metallurgical interaction were to be observed in comingled leaching, resulting in lower recovery), capital costs (i.e. if an increased residence time were to be required to maintain recovery in a comingled circuit, and in turn a larger circuit is required), and operating costs (i.e. if higher acid addition were to be required in a comingled circuit), among others.

5. Inability for mining solution to move through Phoenix deposit at rates required: The PFS assumed the operation of ten recovery wells (out of 94 total recovery wells) and solution grades of 10 g/L to produce 6 M lbs/yr. These assumptions were based on test work completed for the PFS, including from a column test of representative ore samples. At these

quantities, total solution to be recovery would be 500 L/min out of the wellfield. In order to produce this quantity of solution, the orebody must possess a certain level of permeability throughout the deposit. While hydrogeological testing to date indicates that the required permeability is within the range of field test results, there is potential for the overall permeability to be less than current field test results. Should this risk occur, operation of additional wellfields, drilling of wellfields with tighter spacing or recovery of higher-grade solutions may be required in order to meet annual production targets.

6. Gryphon toll milling: The current McClean Lake operating license from the Canadian Nuclear Safety Commission (CNSC) is valid to 2027. The current approval to operate from the Saskatchewan Ministry of the Environment is valid until 2023. There is a risk that McClean Lake may not receive or may be delayed in receiving future licenses, permits, and approvals, which would impact the ability to process ore from the Gryphon operation at McClean Lake.

7. Gryphon toll milling: The McClean Lake mill is an operating facility. Completing the required modifications to process Gryphon ore will require detailed execution planning and construction planning. Construction may need to occur over multiple years in order to utilize planned mill shutdowns to complete tie-ins and other critical tasks. As a result, there is a potential risk to project schedule and cost if execution planning is not carefully completed and managed.

8. Gryphon blind boring: The main production shaft requires a high degree of accuracy in terms of verticality to ensure conveyances are able to travel through the shaft. Application of traditional blind boring technology may not be able to produce a vertical shaft within specifications. In order to ensure verticality, a pilot hole or other measures may be required to guide the blind boring to the end target.

9. Underestimation of capital costs: Wheeler River is exposed to the same risks of capital cost increases as any major mining project. While best practices have been utilized to reduce this risk and deliver accurate cost estimates, actual results may differ from estimates.

1.2.17 Conclusions and Recommendations

The results of the PFS indicate that the Wheeler River project is expected to produce positive economic results under the base case assumptions considered. The results should be considered reliable to guide further decision making by Denison on future next steps in the development of the project which may be a definitive feasibility study.

Review of each area of focus of the PFS has created the following series of principal recommendations for further work programs to improve existing designs, mitigate risks, and unlock project opportunities. More detailed explanations are provided in Section 26.

Hydrogeological:

- Further Phoenix hydrogeological testing should consider targeted pumping tests in the orebody and sandstones;
- Additional lab work to test for chemical erosion of fractures within the ore should be carried out, if feasible; and
- Feasibility hydrogeological testing at Gryphon should target structural features (E-W sub vertical features) that have not been thoroughly tested by current drilling due to drill

orientation bias. Depending on results additional tests including a series of cross hole pumping tests may be appropriate.

Geotechnical:

- Gryphon geotechnical data collection should continue, and focus should be applied to data collection systems including a review of core logging and data collection procedures and use of RMR76 or RMR89 and Q;
- As part of a Gryphon feasibility study, additional geotechnical drilling is recommended. Holes that target the highest grade and widest lenses of economic ore are preferred;
- Additional geotechnical laboratory testing is recommended to confirm intact properties focusing on the altered basement in the hangingwall and ore zone of economic mineralization; and
- Development of a series of geotechnical models including alteration, RQD and RMR to support more detailed underground mine design.

Phoenix Mine Design:

- Evaluate delineation drilling of additional mineralized zones;
- Evaluate procurement of drilling equipment to avoid use of contractor supplier equipment and associate rental costs during wellfield installations;
- Optimize geometry and well spacing of the Phoenix underground freeze curtain;
- Update mineral resource estimate for lower cut off grades and additional resources; and
- Potential pilot test of freeze hole drilling to establish methodology, cost and schedule

Gryphon Mine Design:

- Investigate opportunities to recover additional resource material by using more selective mining methods;
- Review potential to design additional mining fronts early in the mine life to reduce production ramp up schedule;
- Evaluate the opportunity to eliminate services in the ventilation shaft;
- Further review the potential to optimize the delivery of shotcrete to the underground operations via slickline or borehole;
- Investigate options to install dewatering capacity earlier in the mine life to reduce risk of inflows during the initial off-shaft development period;
- Evaluate opportunities to deepen shaft to reduce time to first production; and
- Evaluate opportunities to reduce ore silling development costs.

Gryphon Mineral Processing:

- To further validate the performance of processing Wheeler River ores at the McClean Lake mill, it is recommended that further test work for all process circuits be conducted including co-mingled testing using both Gryphon and Cigar Lake ores;
- Detailed review and study of the existing calciner to confirm capacity of 24 M lbs/ yr U_3O_8;
- A detailed execution plan and construction schedule should be completed for the project to manage identified risk to project schedule;

- Consider potential for processing Gryphon ores on site by construction of front end circuits in the Phoenix ISR plant and avoid capital and operating costs associated with toll milling;
- Minor element deportment and water balance assessment; and
- Evaluate radiometric ore sorting to minimize throughput requirements in the mill.

Phoenix Mineral Processing:

- Evaluate implications of uranium solution concentrations exceeding 10 g/L on plant capacity;
- Conduct confirmatory laboratory pilot plant test work;
- Evaluate opportunities to recover rare earth elements; and

Environmental and Regulatory:

- Continuation of the collection of baseline environmental data;
- Continued engagement with the existing stakeholders and integrate additional traditional knowledge and community input as the project advances; and
- Initiate the Environmental Assessment process by submission of a Project Description.

Table 1-16 summarizes costs to complete the recommendations. These costs have been included in the capital and/or operating costs of the project, as presented in this report. Denison may choose to advance some or all of these items in its future work plan.

Table 1-16. Wheeler River Future Work Plan

Future Work Plan	
Environmental Assessment and other studies	$ 5,000,000
Phoenix Feasibility Study and other studies	$ 5,275,000
Gryphon Feasibility Study and other studies	$ 11,275,000
Wheeler River Grand Total	**$ 21,550,000**

2 Introduction

The Wheeler River project is an advanced exploration stage joint venture owned 90% by Denison and 10% by JCU. Denison is the operator of the joint venture.

Denison is a uranium exploration and development company with interests focused in the Athabasca Basin region of northern Saskatchewan, Canada. In addition to its 63.3% owned Wheeler River project, which hosts the high-grade Phoenix and Gryphon uranium deposits, Denison's exploration portfolio consists of numerous projects covering 320,000 ha in the Athabasca Basin region, including approximately 296,000 ha in the infrastructure rich eastern portion of the Athabasca Basin. Denison's interests in Saskatchewan also include a 22.5% ownership interest in the McClean Lake joint venture, which includes several uranium deposits and the McClean Lake uranium mill, which is currently processing ore from the Cigar Lake mine under a toll milling agreement, plus a 25.17% interest in the Midwest and Midwest A deposits, and a 65.45% interest in the J Zone deposit and Huskie discovery on the Waterbury Lake property. Each of Midwest, Midwest A, J Zone, and Huskie is located within 20 km of the McClean Lake mill.

Denison is also engaged in mine decommissioning and environmental services through its Denison Environmental Services division and is the manager of Uranium Participation Corp., a publicly traded company which invests in uranium oxide and uranium hexafluoride.

The Wheeler River property has been explored since the late 1970s. In late 2004, Denison entered into an agreement with the joint venture partners to earn into a majority 60% interest and become operator of the joint venture. In May 2007, Denison met the earn-in requirements and shortly thereafter in 2008, the Phoenix deposit was discovered.

Drilling at the property from 2008 to 2014 further delineated the Phoenix uranium deposit, which occurs at the intersection of the Athabasca sandstone basal unconformity, a regional fault zone, and graphite-bearing pelitic gneiss basement rocks. The Phoenix deposit consists of two separate lenses, known as Zones A and B, located approximately 400 m below surface within a one-kilometer-long, northeast-trending mineralized corridor. A maiden resource estimate was completed for Phoenix in November 2010 by SRK Consulting (Canada) Inc. (SRK) and in December 2010, Golder Associates Ltd. (Golder) prepared an internal report for Denison on the Phoenix deposit titled "Wheeler River Project – Concept Study" (Golder, 2010). The concept study was used to provide guidance to the exploration teams for exploration strategy, as well as to initiate basic geotechnical, hydrogeological, and environmental data collection programs. As drilling defined further mineralization, subsequent resource estimates were made on the Phoenix deposit in December 2012 and June 2014 by Roscoe Postle Associates Inc. (RPA).

Exploration drilling in early 2014 along the K-North trend resulted in the discovery of a new zone of mineralization, at what would become the Gryphon deposit, which is located approximately three kilometers northwest of the Phoenix deposit. A maiden resource estimate was completed for the Gryphon deposit in November 2015 by RPA and an updated technical report was issued for the Wheeler River project in accordance with the requirements of National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

In September 2015, Denison commissioned SRK and other consultants to prepare a NI 43-101 Preliminary Economic Assessment (PEA) for the project, including both the Phoenix and Gryphon deposits, based on the exploration drilling completed on the property through to the end of the summer 2015 exploration program.

In September 2017, Denison commissioned RPA to prepare an updated mineral resource estimate for the Gryphon deposit in accordance with NI 43-101, based on the additional exploration drilling completed on the property during 2016 and 2017. A technical report with an updated mineral resource estimate was issued in March 2018.

In January 2017, Denison executed an agreement with the partners of the WRJV that will result in an increase in Denison's ownership of the Wheeler River project, to up to approximately 66% by the end of 2018. Under this agreement, Denison is funding 50% of Cameco's ordinary share (30%) of joint venture expenses in 2017 and 2018. On January 31, 2018, Denison announced that it had increased its interest in the Wheeler River project, based on spending on the project during 2017, from 60% to 63.3% in accordance with this agreement.

In December 2017, Denison commissioned a NI 43-101 PFS for the project, including both the Phoenix and Gryphon deposits, based on the mineral resources estimates completed at Phoenix in June 2014 and at Gryphon in March 2018. The purpose of the study is to assess the technical and economic potential for recovery and sale of a uranium product.

2.1 Basis of Technical Report

This technical report (including the portions thereof reproduced from the 2018 PEA Technical Report with an Updated Mineral Resource Estimate for the Wheeler River Property, Northern Saskatchewan, Canada) is based on the following sources of information:

- March 2018, SRK Consulting technical report, "Technical Report with an Updated Mineral Resource Estimate for the Wheeler River Property, Northern Saskatchewan, Canada ";
- Technical and cost information provided by Denison from the Wheeler River and McClean Lake Operations;
- Technical and cost information provided by Woodard & Curran in the areas of in situ recovery (ISR), mineral processing, and wellfield design;
- March 2016, Cameco Corporation technical report, "Cigar Lake Operation Northern Saskatchewan, Canada National Instrument 43-101 Technical Report";
- Technical and cost information provided by Artisan Ltd. in the area of drilling;
- March 2016, SRK Consulting technical report, "Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada";
- Discussions with Denison technical and management personnel; and
- Additional information from public domain sources.

Significant contributions to this report were made by the following consulting firms:

- SRK, commissioned by Denison, is responsible for the evaluation of environmental and regulatory aspects included in section 20, hydrogeological aspects for both gryphon and phoenix included in section 16.2, and the financial modelling including in section 22 and the summary of these sections in the introduction and summary, and the interpretations, conclusions, and recommendations related to these sections.

- RPA, commissioned by Denison, responsible for report Sections 4 to 12, and 14, the summary of these sections in the introduction and summary, and the interpretations, conclusions, and recommendations related to these sections.
- Hatch, commissioned by Denison, responsible for design and costing of mineral processing of gryphon ores at McClean Lake including report Sections 13.2 and 17.2.2-17.2.5 the summary of these sections in the introduction and summary, and the interpretations, conclusions, and recommendations related to these sections.
- North Rock Mining Solutions Inc., commissioned by Denison, is responsible for geotechnical assessment of Gryphon underground mining included in report Section 16.3 summary, and the interpretations, conclusions, and recommendations related to this section.
- Engcomp – Engineering and Computing Professionals, commissioned by Denison, is responsible for Wheeler site surface facilities design and costing included in Section 18 and the summary of these sections in the introduction and summary, and the interpretations, conclusions, and recommendations related to these sections.
- Stantec, commissioned by Denison for Gryphon Mine Design and costing including report Section 16.5 and the summary of these sections in the introduction and summary, and the interpretations, conclusions, and recommendations related to these sections.
- Newmans Geotechnique Inc., commissioned by Denison for design and costing of the Phoenix freeze wall as per Section 16.4.3 and associated summaries contained in the introduction, conclusion and recommendations.
- Woodard and Curran, commissioned by Denison, is responsible for the design of the ISR wellfield and processing plant as outlined in Sections 16.4 and 17.1 associated summaries contained in the introduction, conclusion and recommendations.

This PFS technical report is based on mineral resource statements for the Phoenix and Gryphon deposits prepared by RPA as of March 2018. The mineral resource statements were prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1 and are suitable for public disclosure.

The term "mineral resources within PFS design plan" is used in this technical report to represent portions of the Phoenix and Gryphon uranium mineral resources that have had mining parameters applied to them, including cut-off criteria, external dilution, and mining losses. Mineral resources within PFS design plans are included in the economic analysis as uranium mill feed.

Unless otherwise stated, this technical report is based on Canadian currency and metric units of measure.

2.2 Qualified Persons

This technical report is authored by SRK, with Mark Liskowich, P.Geo. acting as the principal author for SRK and William E. Roscoe, Ph.D., P.Eng. of RPA acting as the principal author for the sections compiled from the March 2018 technical report.

The following Qualified Persons have contributed to those sections of this technical report related to their areas of expertise. By virtue of their education, membership to a recognized professional association, and relevant work experience, they are all independent QPs as this term is defined by NI 43-101.

- Mark Liskowich, P.Geo. – SRK (environmental, permitting, and social impact).

- Michael Royle, P.Geo. – SRK (hydrogeology).
- Michael Selby, P.Eng. – SRK (economic analysis).
- Douglas Graves, P.Eng. – Woodard & Curran Inc. (ISR and mineral processing).
- Mark Hatton, P.Eng. – Stantec (Gryphon mine design, mining costs, infrastructure).
- Mark B. Mathisen, C.P.G. – RPA (geology and mineral resource estimation).
- William McCombe, P.Eng. – Hatch (Gryphon metallurgical testing and mineral processing).
- Gordon Graham, P.Eng - Engcomp (Wheeler site surface facilities design.
- Geoff Wilkie, P.Eng. – Engcomp (Cost estimates for Wheeler site surface facilities including ISR facilities).
- Greg Newman, P.Eng. – Newmans Geotechnique Inc. (ground freezing)
- William E. Roscoe, Ph.D., P.Eng. – RPA (geology and mineral resource estimation).
- Roland Tosney, P.Eng. – North Rock Mining Solutions Inc. (mine geotechnical).

Specific areas of responsibility for each QP are listed in the QP certificates attached at the end of this technical report.

2.3 Qualifications of Consultants

A brief description of the key consultants is summarized below.

SRK comprises more than 1,400 professionals, offering expertise in a wide range of resource engineering disciplines. The independence of SRK is ensured by the fact that it holds no equity in any project it investigates and that its ownership rests solely with its staff. These facts permit SRK to provide its clients with conflict-free and objective recommendations. SRK has a proven track record in undertaking independent assessments of mineral resources and mineral reserves, project evaluations and audits, technical reports, and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. Through its work with many major international mining companies, SRK has established a reputation for providing valuable consultancy services to the global mining industry.

The Stantec community unites approximately 22,000 employees, working in over 400 locations across 6 continents. With local presence and experience in global locations, Stantec helps its clients address issues and advance their projects in harmony with the surrounding community and landscape. Working as one integrated team, Stantec combines its services across a global network, but locates the project team close to the client, so even the most complex projects run simply and smoothly. The best project team has been assembled to suit the needs of this project, with the most qualified resources and relevant services being provided at each stage, from environmental and permitting through to detailed engineering, construction, and maintenance.

Hatch is an employee-owned, multidisciplinary professional services firm that delivers a comprehensive array of technical and strategic services, including consulting, information technology, engineering, process development, and project and construction management to the mining, metallurgical, energy, and infrastructure sectors. In business for over 6 decades, Hatch has completed projects in over 150 countries.

Woodard & Curran personnel have been providing engineering and environmental services for the mining industry for over three decades and have been performing mine and mineral processing waste cleanup for over 25 years. Our mining expertise includes multidisciplinary site assessments, mine facility and reclamation design preparation, bid document preparation, cost estimating, construction oversight, project management, design-build, and post-construction monitoring and maintenance of mine reclamation designs. The Woodard & Curran team has developed multiple ISR process plant designs for the uranium industry, as well as multiple mineral resource estimates for uranium projects.

2.4 Site Visit

Dr. Roscoe visited the property on June 16, 2014 in connection with the Phoenix deposit mineral resource estimate and held discussions with technical personnel in RPA's Toronto office on May 4, 2014. Mark Mathisen (RPA) visited the property on March 23 to 25, 2015, during the winter drill program in connection with the initial Gryphon mineral resource estimate and again from September 21 to 22, 2017 during the summer drill program in relation to the most recent updated Gryphon resource estimate discussed herein. RPA visited several drill sites and reviewed all core handling, logging, sampling, and storage procedures. RPA examined core from several drill holes and compared observations with assay results and descriptive log records made by Denison geologists. As part of the review, RPA verified the occurrences of mineralization visually and by way of a hand-held scintillometer.

Michael Royle, Principal Consultant (Hydrogeology) of (SRK) visited the Wheeler River site June 21 to 23, 2016 to inspect drill core and meet with Denison geological staff to discuss ongoing drilling and hydrogeological testing work. During the site visit, SRK and Denison staff also reviewed the structural core logging and mapping to date, with emphasis on how structure could interact with the site groundwater system.

North Rock Solutions' Roland Tosney, P.Eng., and Lane Maxemiuk, B.Eng., visited the Gryphon site on September 26, 2017. The visit provided an opportunity to observe the project area, discuss regional and local geology with the exploration team, inspect select intervals of drill core to assess geotechnical conditions, and undertake quality assurance on the logged geotechnical parameters.

Michael Selby, P.Eng., Principal Consultant (Mining) of SRK visited the Wheeler River project site on September 26, 2017 to review the conditions and status of the site in preparation of the prefeasibility study.

A site visit to the McClean Lake mill was carried out by William McCombe, P.Eng., Senior Metallurgist from Hatch, between June 5 to 6, 2018.

2.5 Declarations

SRK's opinion contained herein and effective September 24, 2018 is based on information collected by SRK throughout the course of their investigation. The information in turn reflects various technical and economic conditions at the time of writing this report. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK are not insiders, associates, or affiliates of Denison, and neither SRK nor any affiliate has acted as advisor to Denison, its subsidiaries, or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

3 Reliance on Other Experts

The metallurgical test work summarized in Section 13 has been completed by Saskatchewan Research Council (SRC) and R and D Enterprises Inc. (RDE). This test work was relied on by Woodard & Curran for the Phoenix deposit process design and by Hatch for the Gryphon deposit process design.

Directional drilling technical expertise used by Newmans Geotechnique Inc. in generating the Phoenix deposit freeze cap design, summarized in Section 16.4.2, has been provided by Artisan Consulting Services Ltd.

Woodard and Curran and Stantec has relied on Denison and Arcadis Canada Inc. for the radiological screening assessments summarized in Section 16.6 for the Gryphon deposit underground mining and the Phoenix ISR mining and processing.

In preparing Sections 13 and 17, as well as respective portions of Sections 1, 21, 25, and 26, Hatch has relied on Denison for guidance on the terms and conditions of potential joint venture commercial agreements relating to Wheeler River operating costs at McClean Lake. For those same sections within this report, Hatch has also relied on Section 16.3 (including Table 6-2) of Cameco's Cigar Lake NI 43-101 Technical Report, dated March 29, 2016 as the basis for future Cigar Lake feed grades to the McClean Lake mill.

In preparation of Section 17.2.9 with respect to the McClean tailings management facility, SRK has relied upon Denison's knowledge of the status of their 22.5% owned McClean Lake property.

Stantec has relied on the blind boring expertise of Frontier-Kemper Constructors Inc. for the Gryphon deposit production and ventilation shafts excavation method design, including the shaft liner design, as described in Sections 18.6 and 18.7.

SRK has relied on the opinion of Mac McDonald, Chief Financial Officer of Denison, regarding aspects of taxation and royalties. This reliance extends to the information summarized in Section 22.

SRK has relied upon Denison to provide the uranium market overview, including all aspects in Section 19.

4 Property Description and Location

4.1 Property Location

The Wheeler River property, comprising the Phoenix and Gryphon uranium deposits, is located in the eastern Athabasca Basin, approximately 600 km north of Saskatoon, 260 km north of La Ronge, and 110 km southwest of Points North Landing, in northern Saskatchewan (Figure 4-1). The centre of the property is located approximately 35 km northeast of the Key Lake mill and 35 km southwest of the McArthur River mine along provincial highway 914. The property straddles the boundaries of NTS map sheets 74H-5, 6, 11, and 12. The UTM coordinates of the approximate centre of the property are 475,000E and 6,370,000N (NAD83, Zone 13N).

The Gryphon deposit is located approximately 3 km northwest of the Phoenix deposit. The Phoenix deposit was discovered in 2008 and the Gryphon deposit was discovered in 2014. Prior to this report, the estimated mineral resources contained in the Phoenix deposit was last updated in June of 2014, and the estimated mineral resources contained in the Gryphon deposit was last updated in March 2018. The Phoenix deposit is located at the unconformity between the Athabasca Basin and basement rocks, approximately 400 m below surface, whereas the Gryphon deposit is located predominantly in the basement rocks below the unconformity surface.

4.2 Land Tenure

The property, comprising 19 contiguous claims totalling 11,720 ha with an annual requirement of CAD$293,000 in either work or cash to maintain title to the mineral claims, is held as a joint venture among Denison (90%) and JCU (10%). Based on previous work submitted and approved by the Province of Saskatchewan, title is secure until 2035. The claims are shown in Figure 4-2 and listed in Table 4-1. Denison has been the operator of the property since November 10, 2004.

4.3 Mineral Rights

In Canada, natural resources fall under provincial jurisdiction. In the province of Saskatchewan, the management of mineral resources and the granting of exploration and mining rights for mineral substances and their use are regulated by the Crown Minerals Act and the Mineral Tenure Registry Regulations, 2012, which are administered by the Saskatchewan Ministry of the Economy. Mineral rights are owned by the Crown and are distinct from surface rights.

In Saskatchewan, a mineral claim does not grant the holder the right to mine minerals. A mineral claim (Crown disposition) grants the right or privilege to explore or prospect for any Crown mineral or any other right to or interest in any Crown mineral or any Crown mineral lands. A Saskatchewan mineral claim (Crown disposition) in good standing can be converted to a lease (Crown Lease) upon application. Leases have a term of 10 years and are renewable. A lease gives the holder with the exclusive right to explore for, mine, work, recover, procure, remove, carry away, and dispose of any Crown minerals within the lease lands which are nonetheless owned by the province. Surface facilities and mine workings are therefore located on provincial lands and the right to use and occupy lands is acquired under a surface lease from the province of Saskatchewan. A surface lease carries a maximum term of 33 years, and may be extended as necessary, to allow the lessee to develop and operate the mine and plant and thereafter to carry out the reclamation of the lands involved.

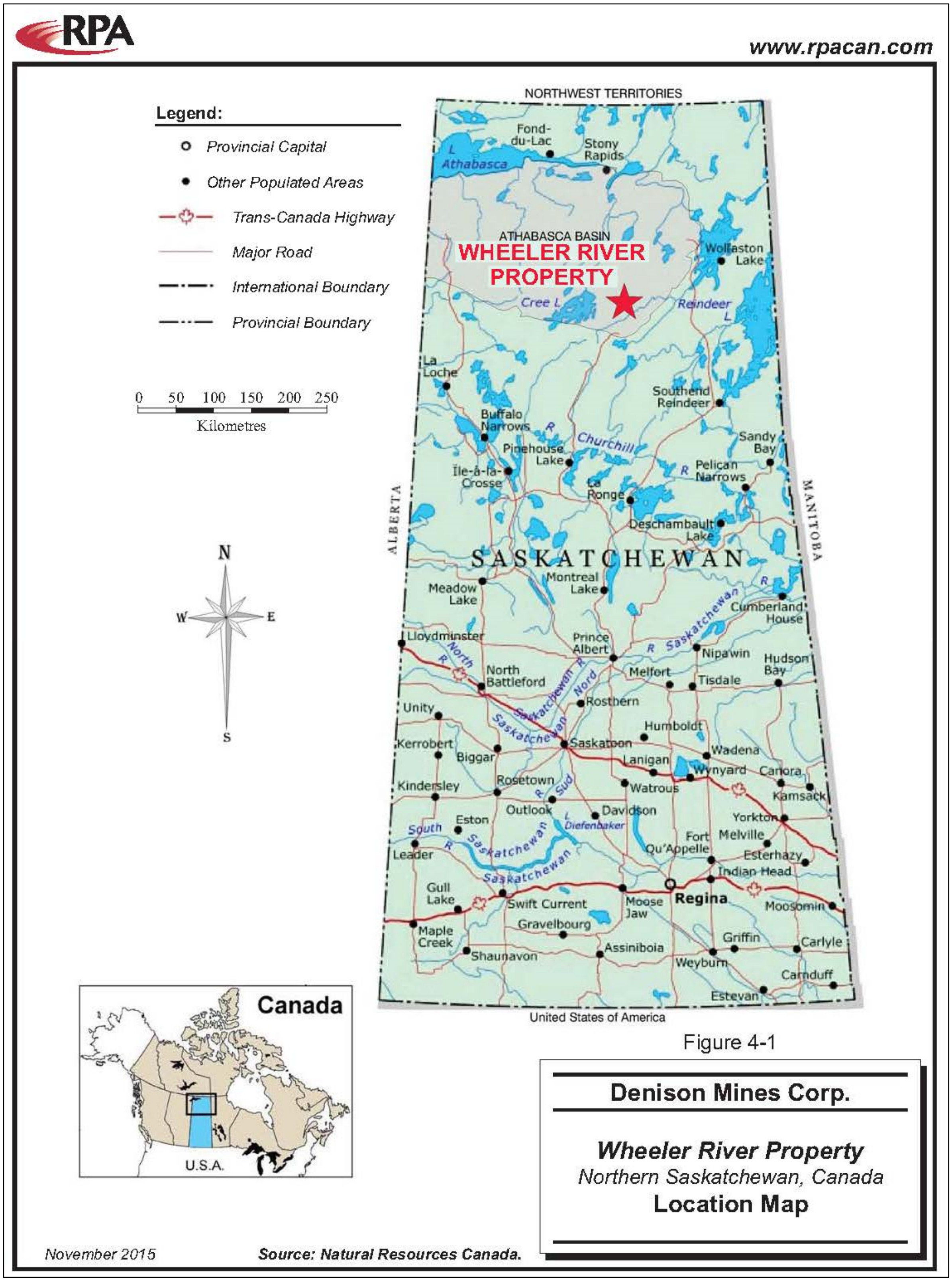


Figure 4-1. Wheeler River Project Location Map

Table 4-1. Land Tenure Details

Disposition #	Area (ha)	Annual Assessment ($)	Excess Credit ($)	Years Protected
S-97677	322	$8,050	$136,850	17
S-97678	335	$8,375	$142,375	17
S-97690	1,087	$27,175	$461,975	17
S-97894	246	$6,150	$104,550	17
S-97895	314	$7,850	$133,450	17
S-97896	356	$8,900	$151,300	17
S-97897	524	$13,100	$222,700	17
S-97907	352	$8,800	$149,600	17
S-97908	1,619	$40,475	$688,075	17
S-97909	1,036	$25,900	$440,300	17
S-98339	362	$9,050	$153,850	17
S-98340	250	$6,250	$106,250	17
S-98341	802	$20,050	$340,850	17
S-98342	1,016	$25,400	$431,800	17
S-98343	362	$9,050	$153,850	17
S-98347	939	$23,475	$399,075	17
S-98348	951	$23,775	$404,175	17
S-98349	540	$13,500	$229,500	17
S-98350	307	$7,675	$130,475	17
	11,720			

4.4 Royalties and Other Encumbrances

The property is subject to royalties on mineral sales and/or profits levied by the Province of Saskatchewan (refer to Section 22). The joint venture also includes a 10% net profits interest ("NPI") associated with the sale of mineral concentrates derived from ore mined from the property. The obligation to pay the NPI is borne by the joint venture participants in proportion to their respective participating interest. The benefit to receive the NPI is also shared by the joint venture participants, in a proportion that may be slightly different than their respective participating interest. While the NPI does not affect the economics of the project on a 100% basis, each joint venture participant could have either a net NPI asset or a net NPI obligation, which does have an impact on the economics of the individual participant's interest in the project. Denison has not included the net impact of the NPI in the economic evaluation of its own interest in the project, as the adjustment is immaterial (refer to Section 22).

Other than the items disclosed above, SRK is not aware of any other royalties due, back-in rights, or other encumbrances by virtue of any underlying agreements.

4.5 Permitting

SRK is not aware of any environmental liabilities associated with the property.

SRK understands that Denison has all the required permits to conduct the proposed work on the property. SRK is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the property.

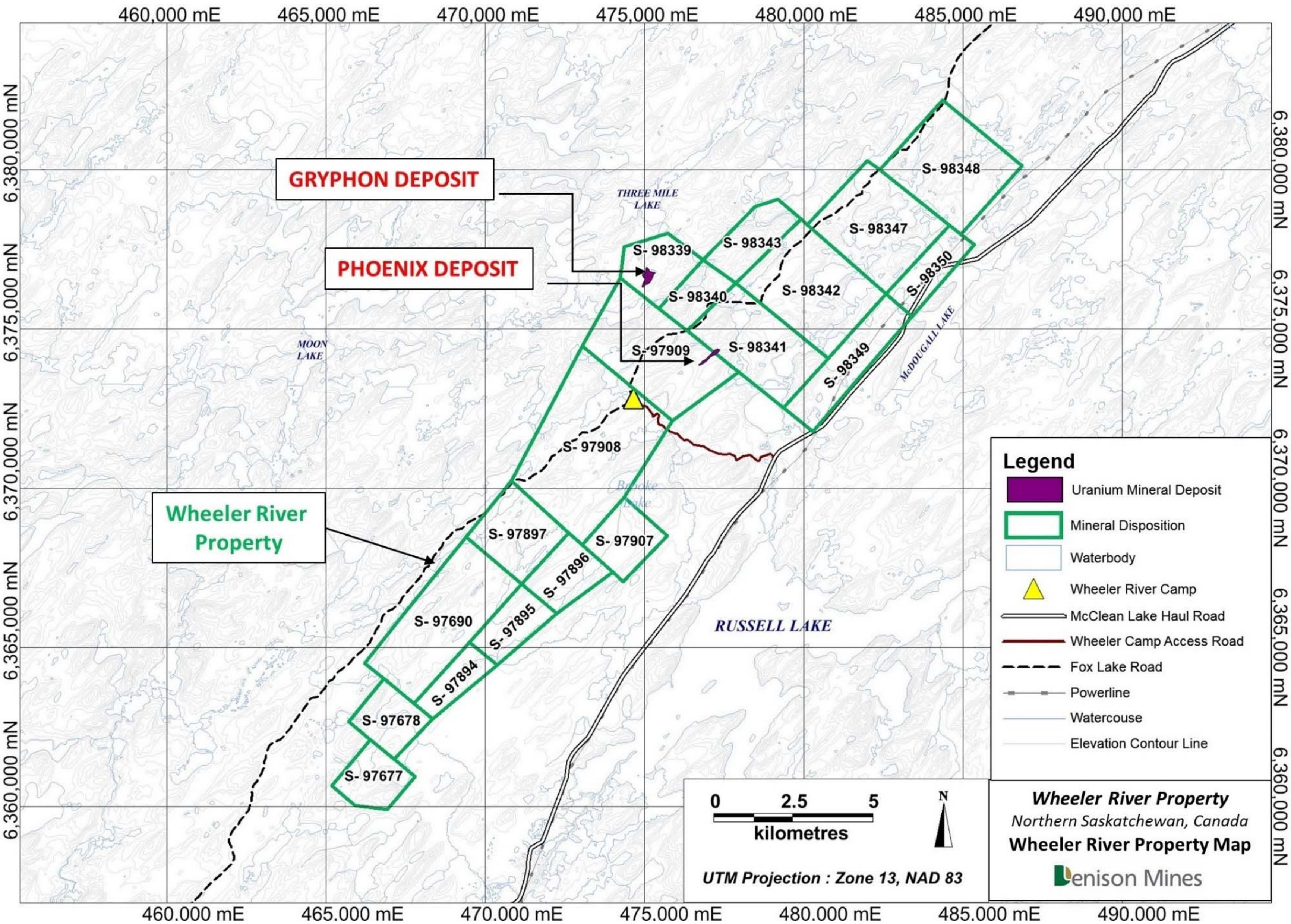


Figure 4-2: Wheeler River Property Map

5 Accessibility, Climate, Local Resources and Infrastructure, and Physiography

5.1 Accessibility

Access to the property and deposits is by road, helicopter, or fixed wing aircraft from Saskatoon. Vehicle access to the property is by Highway 914, which terminates at the Key Lake mill. The haul road between the Key Lake and McArthur River operations lies within the eastern part of the property. An older access road, the Fox Lake Road, between Key Lake and McArthur River provides access to most of the northwestern side of the property. Gravel and sand roads and drill trails provide access by either four-wheel-drive or all-terrain vehicles to the rest of the property.

5.2 Climate

The climate is typical of the continental sub-arctic region of northern Saskatchewan, with temperatures ranging from +32°C in summer to -45°C in winter. Winters are long and cold, with mean monthly temperatures below freezing for seven months of the year. Winter snow pack averages 70 cm to 90 cm. Field operations are possible year-round, with the exception of limitations imposed by lakes and swamps and the periods of break-up and freeze-up.

Freezing of surrounding lakes, in most years, begins in November and break-up occurs around the middle of May. The average frost-free period is approximately 90 days.

Average annual total precipitation for the region is approximately 450 mm, of which 70% falls as rain, with more than half occurring from June to September. Snow may occur in all months but rarely falls in July or August. The prevailing annual wind direction is from the west with a mean speed of 12 km/hr.

5.3 Local Resources and Infrastructure

La Ronge is the nearest commercial/urban centre where most exploration supplies and services can be obtained. TransWest Air offers daily, scheduled flight services between Saskatoon and La Ronge (located approximately 600 km and 260 km, respectively, south of the property). Most company employees are on a two-weeks in and two weeks out schedule.

As noted previously, the property is well located with respect to all weather roads and the provincial power grid. Most significantly, the operating Key Lake mill complex is approximately 35 km south of the property.

Field operations are currently conducted from Denison's Wheeler River camp, 4 km south of Gryphon and 3 km southwest of Phoenix (Figure 4-2). The camp, which is operated by Denison, provides accommodations for up to 40 exploration personnel. Fuel and miscellaneous supplies are stored in existing warehouse and tank facilities at the camp. The site generates its own power. Abundant water is available from the numerous lakes and rivers in the area.

5.4 Physiography

The property is characterized by a relatively flat till plain with elevations ranging from 477 to 490 MASL. Throughout the area, there is a distinctive north-easterly trend to landforms resulting from the passage of Pleistocene glacial ice from the northeast to the southwest. The topography and vegetation at the property are typical of the taiga forested land common to the Athabasca Basin area of northern Saskatchewan.

The area is covered with overburden from 0 to 130 m in thickness. The terrain is gently rolling and characterized by forested sand and dunes. Vegetation is dominated by black spruce and jack pine, with occasional small stands of white birch occurring in more productive and well-drained areas. Lowlands are generally well drained but can contain some muskeg and poorly drained bog areas with vegetation varying from wet, open, non-treed vistas to variable density stands of primarily black spruce as well as tamarack depending on moisture and soil conditions. Lichen growth is common in this boreal landscape mostly associated with mature coniferous stands and bogs.

6 History

6.1 Prior Ownership

The Wheeler River property was staked on July 6, 1977, due to its proximity to the Key Lake uranium discoveries, and was vended into an agreement on December 28, 1978 among AGIP Canada Ltd. (AGIP), E&B Explorations Ltd. (E&B), and Saskatchewan Mining Development Corporation (SMDC), with each holding a one-third interest. On July 31, 1984, all parties divested a 13.3% interest and allowed Denison Mines Limited, a predecessor company to Denison, to earn a 40% interest. On December 1, 1986, E&B allowed PNC Exploration (Canada) Co. Ltd. (PNC) to earn a 10% interest from one-half of its 20% interest. In the early 1990s, AGIP sold its 20% interest to Cameco, which was a successor to SMDC. In 1996, Imperial Metals Corporation, a successor to E&B, sold an 8% interest to Cameco and a 2% interest to PNC. Participating interests in 2004 were Cameco 48%, JCU 12% (a successor to PNC), and Denison 40%.

In late 2004, Denison entered into an agreement to earn a further 20% interest by expending $7 million within six years. When the earn-in obligations were completed, the participating interests were Denison 60%, Cameco 30%, and JCU 10%. Since November 2004, Denison has been the operator of the WRJV.

In January 2017, Denison executed an agreement with the partners of the WRJV that will result in an increase in Denison's ownership of the Wheeler River project to up to approximately 66% by the end of 2018. Under the terms of the agreement, the joint venture parties had agreed to allow for a one-time election by Cameco to fund 50% of its ordinary share of joint venture expenses in 2017 and 2018. The shortfall in Cameco's contribution are being funded by Denison, in exchange for a transfer of a portion of Cameco's interest in the project. Accordingly, Denison's share of joint venture expenses is 75% in 2017 and 2018, and Cameco and JCU's share of joint venture expenses will be 15% and 10%, respectively.

On January 31, 2018, Denison announced that it had increased its interest in the Wheeler River project, based on spending on the project during 2017, from 60% to 63.3% in accordance with this agreement. Denison expects to increase its interest to approximately 66% by the end of 2018, with Cameco’s interest declining to approximately 24% and JCU’s interest remaining at 10%.

6.2 Exploration and Development History

Excluding the years 1990 to 1994, exploration activities comprising airborne and ground geophysical surveys, geochemical surveys, prospecting, and diamond drilling have been carried out on the Wheeler River property continuously from 1978 to the present.

Subsequent to the discovery of the Key Lake mine in 1975 and 1976, the Key Lake exploration model (Dahlkamp and Tan, 1977) has emphasized the spatial association between uranium deposition at, immediately above, or immediately below the unconformity with graphitic pelitic gneiss units in the basement subcrop under the basal Athabasca sandstone. The graphitic pelitic gneiss units are commonly intensely sheared and are highly conductive in contrast to the physically more competent adjoining rock types that include semipelitic gneiss, psammite, meta-arkose, or granitoid gneiss. From the late 1970s to the present, the Key Lake model has been useful in discovering blind uranium deposits throughout the Athabasca Basin (Jefferson et al., 2007), although it is worth noting that the vast majority of electromagnetic (EM) conductors are unmineralized.

Following the Key Lake exploration model, EM techniques were the early geophysical methods of choice for the Wheeler River property area during the period 1978 to 2004 and more than 152 line-km of EM conductors have been delineated on the property. These conductive units have been delineated to depths of 1,000 m, through the quartz-rich Athabasca Group sandstones that are effectively transparent from an EM perspective.

These conductors or conductor systems were assigned a unique designation and follow-up exploration drilling successfully identified several zones of uranium mineralization.

In 1982, AGIP discovered the MAW Zone. This alteration system contains rare earth element (REE) mineralization in a structurally disrupted zone which extends from the unconformity to the present surface. There is no evidence of uranium mineralization. The REE mineralization contains yttrium values greater than 2.0%, boron values up to 2.5%, and total rare earth oxide (REO) up to 8.1%.

In 1985, SMDC (predecessor to Cameco) drilled ZK-02 to test a moderate UTEM conductor axis in a previously unexplored area along the K-North conductor, which is now known as Gryphon. The drill hole intersected several zones of hydrothermal alteration in the sandstone indicating that the conductor was likely overshot and thus lay grid east of ZK-02.

In 1986, SMDC intersected uranium mineralization associated with Ni-Co-As sulphides at the unconformity in the M Zone (DDH ZM-10, 0.79% U_3O_8 over 5.75 m), and also discovered uranium mineralization at the O Zone, which is associated with a 72 m vertical unconformity offset. The O Zone basement-hosted mineralization graded 0.048% U_3O_8 over 0.9 m at 378.8 m in drill hole ZO-02.

In 1988, Cameco drilled ZK-04 and ZK-06 on the same drill section as ZK-02 to test for the UTEM conductor and follow up on the sandstone alteration. Hole ZK-04 was drilled 120 m grid east of ZK-02, and hole ZK-06 was drilled 35 m grid west of ZK-04. In drill hole ZK-04, a major basement fault structure was intersected from 572.6 m to 603.2 m, with associated strong hydrothermal alteration and a 9.8 m radioactive zone from 581.7 m to 591.5 m. Assays from drill hole ZK-04 returned 0.08% U_3O_8 over 2.4 m at 580.0 m and 0.19% U_3O_8 over 2.3 m at 587.7 m. Moderate to strong hydrothermal alteration and associated fault gouges and fracturing continued to the end of the hole at 631 m (approximately 112 m below the unconformity surface).

The third hole on this section, ZK-06, was drilled up-dip of ZK-04 in an attempt to locate the up-dip and unconformity extension of the mineralization intersected in drill hole ZK-04. Two significant zones of weak mineralization and elevated radioactivity were intersected within a 12.1 m zone, 11 m to 50 m below the unconformity. ZK-06 returned 0.17% U_3O_8 over 7.7 m at 532.0 m and 0.06% U_3O_8 over 4.4 m at 564.6 m. Intense alteration, fracturing, and faulting in the sandstone was noted, as well as alteration and structure extending approximately 50 m into the basement rocks. At this time, ZK-06 was thought to have intersected the unconformity target and no follow-up was conducted for several years.

From 1995 to 1997, exploration by Cameco identified strong alteration and illitic and dravitic geochemical enrichment associated with major structures in both the sandstone and the basement and a significant unconformity offset associated with the “quartzite ridge” which had been delineated as a result of drilling the Q conductor system.

In 1998, further drilling was carried out at the Q Zone and also at the R Zone (the Phoenix deposit area). At the R Zone, two drill holes were abandoned in sandstone due to quartz dissolution (desilicification). The possibility that this sandstone alteration might be of significance was not emphasized at the time.

In 1999, a geological setting similar to McArthur River's P2 trend was intersected at the WC Zone, where faulted graphite-pyrite pelitic gneiss overlay the quartzite ridge. The former operator (Cameco) noted extensive dravite (boron) alteration in the overlying sandstones.

In 2001, Cameco drilled ZK-23, testing the K1A SWML conductor approximately 250 m grid east of the ZK-02\ZK-04\ZK-06 drill fence in what is now the Gryphon area. The drill hole intersected a wide zone of structural disruption within the sandstone 40 m above the unconformity. The conductive response was explained by a wide zone of moderately graphitic-pyritic pelitic gneisses. No unconformity or basement mineralization was intersected, and no follow-up drill holes were recommended.

In 2002, drill hole WR-185 intersected a 175 m unconformity offset along the west contact of the quartzite ridge. This area was the initial focus of the WRJV after Denison became operator in 2004.

In 2003, 61 shallow reverse circulation (RC) holes were drilled, targeting the sandstone/overburden interface exploring for alteration zones in the upper sandstone. No anomalies were detected. Drill hole WR-190A tested the WS UTEM conductor and was abandoned at 364 m due to deteriorating drilling conditions. This drill hole is located only 90 m from the eventual Phoenix discovery drill hole WR-249. Noticeable desilicification and bleaching of the sandstone were present, but no noteworthy geochemical anomalies were identified. A direct current (DC) resistivity survey was also completed to map trends of alteration within the Athabasca sandstones and underlying basement rocks that might be related to uranium mineralization.

In November 2004, Denison became operator of the WRJV and in 2005 carried out property-wide airborne Fugro GEOTEM EM and Falcon gravity surveys with five subsequent ground transient EM (TEM) grids completed on GEOTEM anomalies. The focus for Denison, based on a McArthur River analogy, was the quartzite ridge, particularly the west, or footwall side of the ridge. Several small regional campaigns were carried out to test EM conductors located by airborne and ground geophysical surveys.

In 2007, a 154.8 line-km geophysical induced polarization (IP) and magnetotelluric (MT) survey using Titan 24 DC resistivity technology was undertaken with the prime goals being the extension of Cameco's 2003 resistivity survey, surveying of the K and M zones, and exploration of the REA or "Millennium" (WS) Zone, which appeared to have attractive geological features in an underexplored part of the property. The results showed the following:

- A very strong resistivity high which delineated the quartzite unit;
- Two strong, well defined resistivity lows both occurring in areas where previous drill holes had been lost in the Athabasca sandstone; and
- Well defined resistivity chimneys.

Although 2007 drilling on various 2003 resistivity anomalies did not discover any significant uranium mineralization, there was some support for the concept that resistivity did "map" alteration chimneys within the Athabasca sandstone. Alteration chimneys in the Athabasca sandstone above the unconformity or basement-hosted uranium mineralization have been described from almost all Athabasca Basin uranium deposits, following the first thorough description of their occurrence at the McClean deposits (Saracoglu et al., 1983; Wallis et al., 1984). The chimneys nearly always have a prominent structural component consisting of broken and rotated sandstone and a high degree of fracturing and brecciation. These structural features are accompanied by alteration consisting of variable amounts of bleaching (removal of diagenetic hematite), silicification, desilicification, druzy quartz-lined fractures, secondary hematite, dravite, and/or clay minerals which can cause resistivity anomalies.

During the winter and spring of 2008, the North Grid resistivity survey data was reinterpreted and three drill targets, A, B, and C, were proposed. These targets were well defined alteration or resistivity chimneys situated close to the hangingwall of the quartzite unit in areas where previous attempts to drill ground EM conductors (the WS and the REA) had failed to reach the unconformity. In 2008, drill hole WR-249 led to the discovery of the Phoenix deposit. Subsequent drilling identified four mineralized zones over a strike length of more than one kilometer: Phoenix zones A, B, C, and D.

In March 2014, drill hole WR-556 resulted in discovery of the Gryphon deposit, intersecting uranium mineralization averaging 15.33% U_3O_8 over 4.0 m in basement graphitic gneiss, 200 m below the sub-Athabasca unconformity. Since the discovery of the Phoenix deposit in 2008, exploration efforts have been focused on the K-Zone trend which exhibits numerous favourable exploration criteria including basement quartzite and graphitic gneisses, basement structures, reverse offsets of the unconformity, weak basement hosted mineralization near the unconformity, and anomalous sandstone geochemistry and alteration. Historical holes ZK-04 and ZK-06 drilled in the late 1980s, targeting unconformity-related mineralization, intersected favourable sandstone structure and alteration as well as alteration and weak mineralization in the basement approximately 35 m below the unconformity. Follow-up drilling campaigns attempted to locate unconformity mineralization up dip of the weak basement mineralization. Gryphon deposit discovery drill hole WR-556 was the first to evaluate the down dip projection of these intersections.

Subsequent drilling on the property from 2014 to present has focused on delineating the extent and continuity of the Gryphon deposit as well as evaluating additional high priority areas along the K-North trend.

Table 6-1 is a summary of the exploration activities that have been carried out on the Wheeler River property.

Table 6-1. Exploration and Development History

Period (Year)	Activity
1978-Present	The area was previously explored by AGIP and SMDC (Cameco). Since 1978, several airborne and ground geophysical surveys have defined 152 km of conductor strike length in 14 conductive zones.
1986-1988	AGIP, SMDC, and Cameco drilled a total of 192 drill holes encountering sub-economic uranium mineralization in the M Zone (1986), O Zone (1986), and K Zone (1988). Rare earth element mineralization was also discovered in the MAW Zone (1982).
2004	Denison assumed operatorship in 2004 and initially focused on the footwall side of the quartzite ridge (west side of the property) intersecting sub-economic uranium mineralization.

Period (Year)	Activity
2008	In 2008, three resistivity targets were drilled leading to the discovery of the Phoenix deposit.
2008-2014	During the period 2008 to 2014, drilling predominantly focused on defining the Phoenix deposits.
2014-Present	Subsequent drilling has discovered and delineated the Gryphon deposit.

6.3 Previous Mineral Resource Estimates

An initial mineral resource estimate was reported for the Phoenix deposit in a NI 43-101 technical report by SRK dated November 17, 2010 (Table 6-2). An updated mineral resource estimate for the Phoenix deposit Zones A and B was prepared by RPA on December 31, 2012 (Table 6-3). A further updated mineral resource estimate for the Phoenix deposit Zones A and B was prepared by RPA on May 28, 2014 (Table 6-4) and an initial mineral resource estimate for the Gryphon deposit was prepared by RPA on September 25, 2015 (Table 6-5). All previous mineral resource estimates are superseded by the updated mineral resource estimate in the current Wheeler River technical report, which incorporates additional drilling completed at Gryphon since 2015.

Table 6-2. SRK Mineral Resource Estimate as of November 17, 2010 (100% Basis) Denison Mines Corp. – Phoenix Deposit

Deposit	Classification	Tonnes (000)	lbs U_3O_8 (000)	Average Grade (% U_3O_8)
Zone A	Indicated	89.9	35,638	18.0
Zone B	Inferred	23.8	3,811	7.3

Source: Arseneau and Revering, 2010

Table 6-3. RPA Mineral Resource Estimate as of December 31, 2012 (100% Basis) Denison Mines Corp. – Phoenix Deposit

Category	Tonnes	Grade (% U_3O_8)	Million lbs U_3O_8
Indicated	152,400	15.6	52.3
Inferred	11,600	29.8	7.6

Source: Roscoe, 2012

Table 6-4. RPA Mineral Resource Estimate as of May 28, 2014 (100% Basis) Denison Mines Corp. – Phoenix Deposit

Category	Tonnes	Grade (% U_3O_8)	Million lbs U_3O_8
Indicated	166,400	19.13	70.2
Inferred	8,600	5.80	1.1

Source: Roscoe, 2014

Table 6-5. RPA Mineral Resource Estimate as of September 25, 2015 (100% Basis) Denison Mines Corp. – Phoenix and Gryphon Deposits

Deposit	Classification	Tonnes	Million lbs U_3O_8	Average Grade (% U_3O_8)
Phoenix	Indicated	166,400	70.2	19.14
Phoenix	Inferred	8,600	1.1	5.80
Gryphon	Inferred	834,000	44.1	2.31

Source: Roscoe, 2015

The current report includes the Phoenix mineral resource estimate documented in the RPA (2015) technical report as well as the updated mineral resource estimate for the Gryphon deposit.

6.4 Past Production

To date, no production has occurred on the property and the property is still at the advanced exploration stage.

7 Geological Setting and Mineralization

7.1 Regional Geology

7.1.1 General

The Phoenix and Gryphon uranium deposits are located near the southeastern margin of the Athabasca Basin in the southwest part of the Churchill Structural Province of the Canadian Shield (Figure 7-1). The Athabasca Basin is a broad, closed, and elliptically shaped cratonic basin with an area of 425 km (east-west) by 225 km (north-south). The bedrock geology of the area consists of Archean and Paleoproterozoic gneisses unconformably overlain by up to 1,500 m of flat-lying, unmetamorphosed sandstones and conglomerates of the mid-Proterozoic Athabasca Group. The property is located near the transition zone between two prominent litho-structural domains within the Precambrian basement: the Mudjatik Domain to the west and the Wollaston Domain to the east.

The Mudjatik Domain is characterized by elliptical domes of Archean granitoid orthogenesis separated by keels of metavolcanic and metasedimentary rocks, whereas the Wollaston Domain is characterized by tight to isoclinal, north-easterly trending, doubly plunging folds developed in Paleoproterozoic metasedimentary rocks of the Wollaston Supergroup (Yeo and Delaney, 2007), which overlie Archean granitoid orthogenesis identical to those of the Mudjatik Domain.

The area is cut by a major northeast-striking fault system of Hudsonian Age. The faults occur predominantly in the basement rocks but often extend up into the Athabasca Group due to several periods of post-depositional movement. Diabase sills and dikes up to 100 m in width and frequently associated with the faulting have intruded into both the Athabasca rocks and the underlying basement.

7.1.2 The Metamorphosed Basement

The basement rocks underlying the Athabasca Group have been divided into three tectonic domains: the Western Craton, the Cree Lake Mobile Zone, and the Rottenstone Complex (Figures 7-1 and 7-2). The central Cree Lake Mobile Zone is bounded in the northwest by the Virgin River Shear and Black Lake fault and in the southeast by the Needle Falls Shear Zone.

The Cree Lake Mobile Zone has been further subdivided into the Mudjatik Domain in the west half and the Wollaston Domain in the east half. The lithostructural character of these domains is the result of the Hudsonian Orogeny in which an intense thermo-tectonic period remobilized the Archean age rocks and led to intensive folding of the overlying Aphebian-age supracrustal metasedimentary units. The Mudjatik Domain represents the orogenic core and comprises non-linear, felsic, granitoid to gneissic rocks surrounded by subordinate thin gneissic supracrustal units. These rocks, which have reached granulite-facies metamorphic grades, usually occur as broad domal features. The adjacent Wollaston Domain consists of Archean granitoid gneisses overlain by an assemblage of Aphebian pelitic, semipelitic, and arkosic gneisses, with minor interlayered calc-silicate rocks and quartzites. These rocks are overlain by an upper assemblage of semipelitic and arkosic gneisses with magnetite bearing units.

The Wollaston Domain basement rocks are unconformably overlain by flat lying, unmetamorphosed sandstones, and conglomerates of the Helikian age Athabasca Group, which is a major aquifer in the area.

7.1.3 The Athabasca Group

The Athabasca Group sediments consist of unmetamorphosed pink to maroon quartz-rich pebbly conglomerate and red siltstone of the Read Formation and maroon quartz-pebble conglomerate, maroon to white pebbly sandstone, sandstone and clay-clast-bearing sandstone belonging to the Manitou Falls Formation. The sandstone is poorly sorted near the base, where conglomerates form discontinuous layers of variable thickness. Minor shale and siltstone occur in the upper half of the succession. Locally, the rocks may be silicified and indurated or partly altered to clay and softened. In spite of their simple composition, their diagenetic history is complex (Jefferson et al., 2007). The predominant regional background clay is dickite.

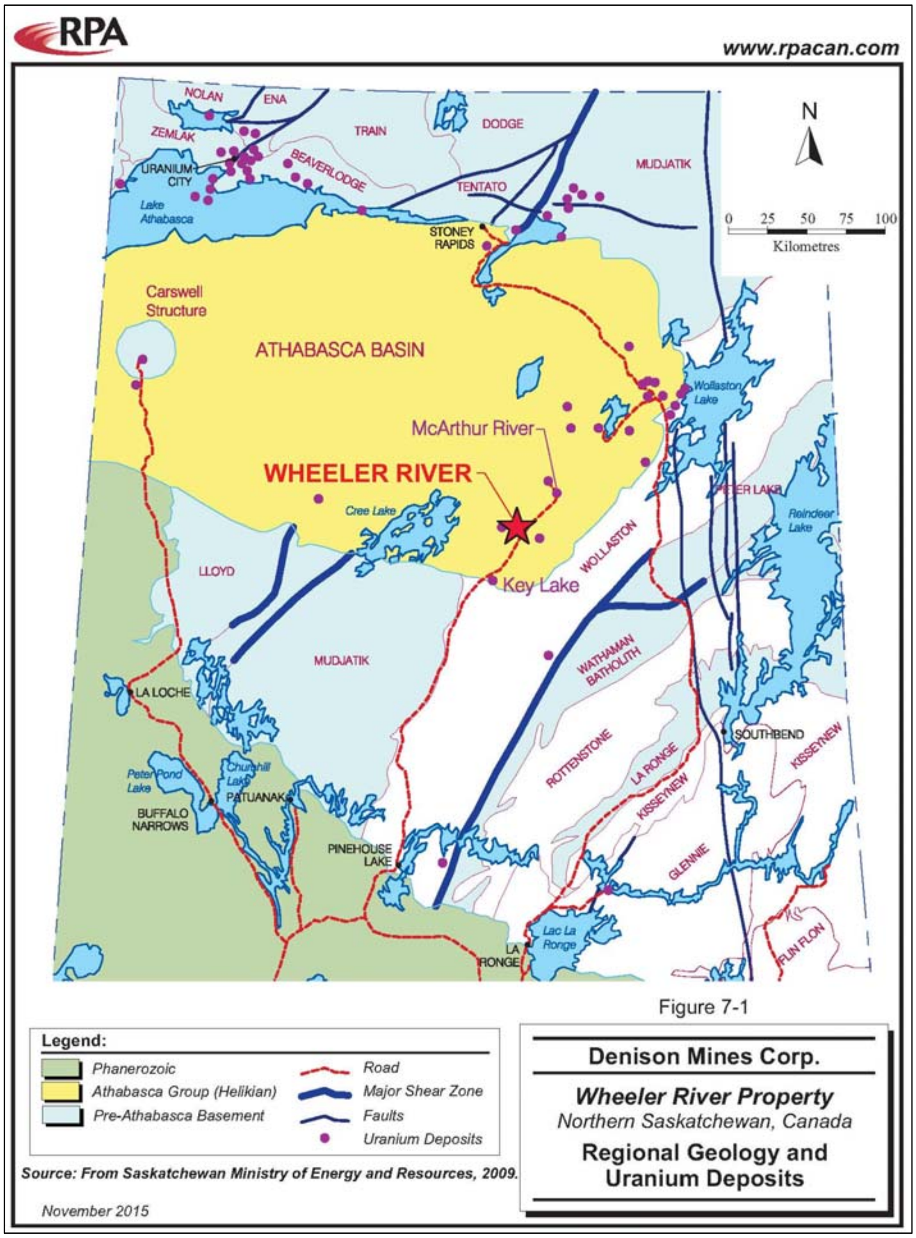


Figure 7-1. Regional Geology and Uranium Deposits

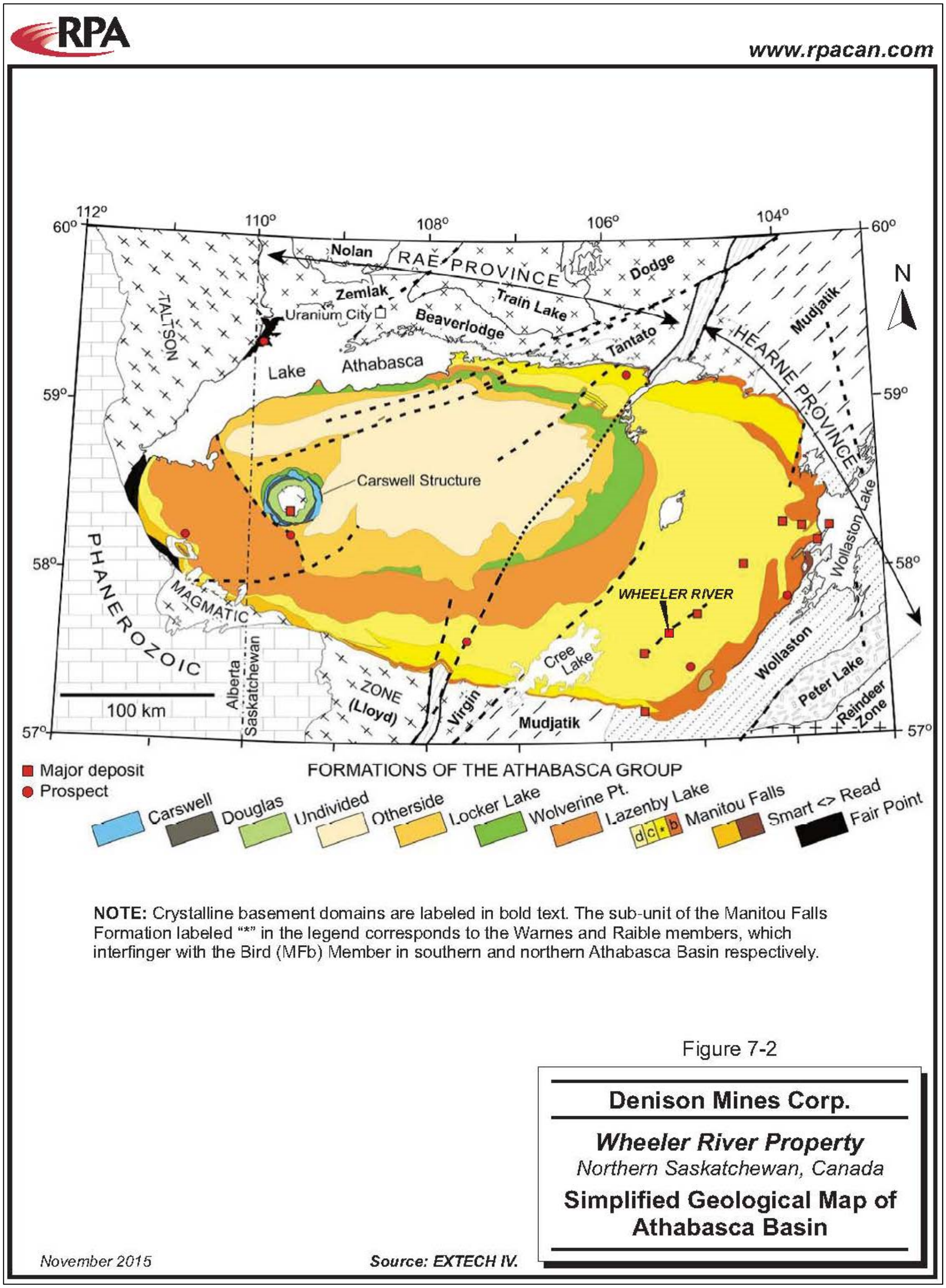


Figure 7-2. Simplified Geological Map of Athabasca Basin

The basin is interpreted to have developed from a series of early northeast-trending fault-bounded sub-basins that coalesced. The topographic profile of the unconformity suggests a gentle inward slope in the east, moderate to steep slopes in the north and south, and a steeper slope in the west.

Subdivisions of the Athabasca Group in the eastern part of the basin (Figure 7-2) include four members from bottom to top:

- Read Formation (formerly the MFa Member) - a sequence of poorly sorted sandstone and minor conglomerate.
- Bird Member (MFb) - interbedded sandstone and conglomerate distinguished from the underlying MFa and overlying MFc by the presence of at least 1% to 2% conglomerate in beds thicker than 2 cm.
- Collins Member (MFc) - a sandstone with rare clay intraclasts.
- Dunlop Member (MFd) - a fine-grained sandstone with abundant (>1%) clay intraclasts.

7.2 Quaternary Deposits

In the eastern Athabasca Basin, Quaternary glacial deposits up to 100 m thick drape bedrock topography of ridges, typically associated with granitic gneiss domes, and structurally controlled valleys (Campbell, 2007). At least three tills, locally separated by stratified gravel, sand, and silt, can be distinguished. The dominant ice-flow direction was southwesterly, but a late glacial re-advance was southerly in eastern parts of the basin and westerly along its northern edge.

7.3 Local and Property Geology

7.3.1 General

The Wheeler River property lies in the eastern part of the Athabasca Basin where undeformed, late Paleoproterozoic to Mesoproterozoic sandstone, conglomerate, and mudstone of the Athabasca Group unconformably overlie early Paleoproterozoic and Archean crystalline basement rocks, as described below. The local geology of the property is very much consistent with the regional geology described above.

7.3.2 Quaternary Deposits

The property is partially covered by lakes and muskeg, which overlie a complex succession of glacial deposits up to 130 m in thickness. These include eskers and outwash sand plains, well-developed drumlins, till plains, and glaciofluvial plain deposits (Campbell, 2007). The orientation of the drumlins reflects southwesterly ice flow.

7.3.3 Athabasca Group

Little-deformed late Paleoproterozoic to Mesoproterozoic Athabasca Group strata comprised of Manitou Falls Formation sandstones and conglomerates unconformably overlie the crystalline basement and have a considerable range (Figure 7-3) from 170 m over the quartzite ridge to at least 560 m on the western side of the property.

The Manitou Falls Formation is locally separated from the underlying Read Formation (formerly the MFa) by a paraconformity, and comprises three units, the Bird Member (MFb), Collins Member (MFc), and Dunlop Member (MFd), which are differentiated based on conglomerates and clay intraclasts (Bosman and Korness, 2007; Ramaekers et al., 2007). Thickness of the Read Formation ranges from zero meters at the north end of the property and over parts of the quartzite ridge to 200 m west of the quartzite ridge. The thickness of the MFb, which is absent above the quartzite ridge, is as much as 210 m in the northeastern part of the property. The MFc unit is a relatively clean sandstone with locally scattered granules or pebbles and one-pebble-thick conglomerate layers interpreted to be pebble lag deposits. The MFc ranges in thickness from 30 m to 150 m. The MFd is distinguished from the underlying MFc sandstone by the presence of at least 0.6% clay intraclasts (Bosman and Korness, 2007). The MFd is up to 140 m thick. The upper 100 m to 140 m of sandstone is typically buff coloured, medium to coarse grained, quartz rich and cemented by silica, kaolinite, illite, sericite, or hematite. Alteration of the sandstone is noted along much of the Phoenix deposit trend.

Variations in thickness of the Athabasca sub-units reflect syndepositional subsidence. In particular, the thinning of the Read Formation towards the quartzite ridge, and the absence of both the Read and the MFb Member over much of the ridge, indicate syn-Read uplift of the latter along the thrust fault that bounds it to the west. This is supported by the Read Formation sedimentary breccia, interpreted as a fault-scarp talus deposit, along the western margin of the ridge.

Although the predominant regional background clay in the Athabasca Basin is dickite, the property lies within a broad illite anomaly trending north-easterly from Key Lake through the McArthur River area (Earle and Sopuck, 1989). Chlorite and dravite are also relatively common in sandstones within this zone.

The topography of the sub-Athabasca basement varies dramatically across the property. From elevations of 160 MASL to 230 MASL along its southeastern edge, the unconformity rises gently to a pronounced north-easterly trending ridge up to 350 MASL, coincident with the subcrop of a quartzite unit in the crystalline basement. The unconformity surface drops steeply westward to as low as 30 m below sea level. The unconformity surface is less variable in the northern part of the property, ranging from 40 MASL in the northeast to 200 MASL in the northwest.

The west side of the quartzite unit forms a prominent topographic scarp, rising up to 200 m above the sub-Athabasca unconformity lying to the west. The breccia of angular quartzite blocks, centimeters to meters in size, with a finely laminated sandstone matrix, has been intersected in numerous drill holes along the western margin (footwall) of the quartzite ridge. The quartzite breccia is often intimately associated with uranium mineralization that occurs at numerous locations along the footwall of the quartzite unit.

The Athabasca sandstones were deposited as a succession of sandy and gravelly braided river deposits in westward-flowing streams. The conglomerates typical of MFb indicate increased stream competence, due either to increased flow (i.e. higher precipitation) or increased subsidence. The mud chips typical of MFd are fragments of thin mud beds deposited from suspension during the late stages of a flood and re-worked by the next one. Hence, they indicate intermittent, possibly seasonal, stream flow (Liu et al., 2011).

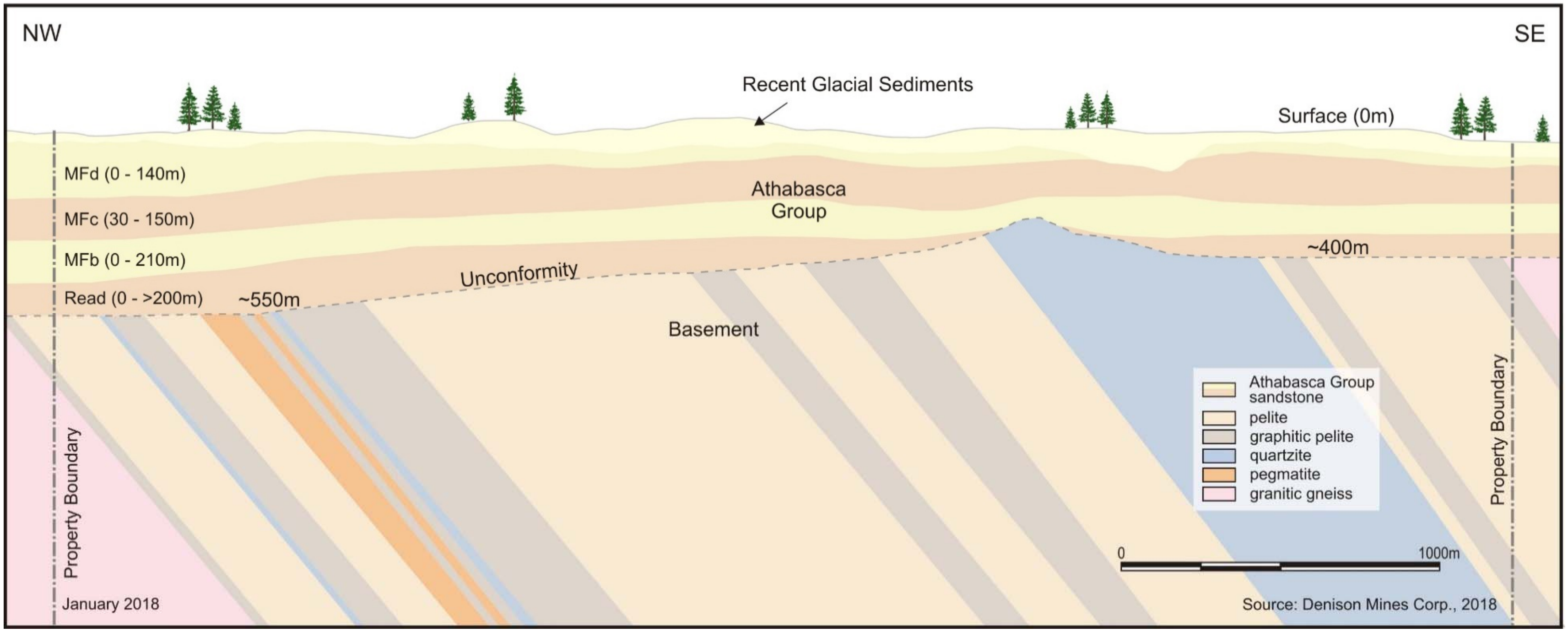


Figure 7-3. Cross-section of Wheeler River Athabasca and Basement Rock Types

7.3.4 Basement Geology

Basement rocks beneath the Phoenix and Gryphon deposits are part of the Wollaston Domain and are comprised of metasedimentary and granitoid gneisses (Figure 7-4). The metasedimentary rocks belong to the Wollaston Supergroup and include graphitic and non-graphitic pelitic and semipelitic gneisses, meta-quartzite, and rare calc-silicate rocks together with felsic and quartz feldspathic granitoid gneisses. These metasedimentary rocks are interpreted to belong to the Daly Lake Group (Yeo and Delaney, 2007). Pegmatitic segregations and intrusions are common in all units with garnet, cordierite, and sillimanite occurring in the pelitic strata, indicating an upper amphibolite grade of metamorphism.

Graphitic pelitic gneiss and quartzite units appear to play important roles in the genesis of Athabasca Basin unconformity-type deposits (Jefferson et al., 2007). Thus, the presence of extensive subcrop of both units: 18 km of quartzite and 152 line-km of conductors (assumed to be graphitic pelitic gneiss), greatly enhances the economic potential of the Wheeler River property.

All of these rock types have a low magnetic susceptibility. The metasedimentary rocks are flanked by and intercalated with granitoid gneisses, some of which have a relatively high magnetic susceptibility. Some of these granitoid gneisses are Archean (Card et al., 2007). Prior to extensive drilling, interpretation of basement geology depends heavily on airborne magnetic data combined with airborne and ground EM interpretation.

A “Paleoweathered Zone”, generally from 3 m to 10 m thick, is superimposed on the crystalline rocks and occurs immediately below the unconformity.

7.3.5 Phoenix Deposit

The quartzite ridge, an interpreted impermeable and structural barrier forming the footwall to the mineralization, dominates the basement geology at the Phoenix deposit. The quartzite unit exhibits variable dips from -45º to -75º to the southeast, averaging -50º, and with an undulating, but generally 055º azimuth. Immediately overlying the quartzite is a garnetiferous pelitic gneiss, which varies from seven meters to 60 m in thickness. This generally competent and unmineralized unit contains distinctive porphyroblastic garnets and acts as a marker horizon. Overlying the garnetiferous pelitic gneiss is a graphitic pelitic gneiss in which the graphite content varies from 1% to 40%. The graphitic pelitic gneiss is approximately 5 m wide in the southwest, increases to approximately 70 m near drill hole WR-249, and is 50 m wide at the northeast extremity. Overlying the graphitic pelitic gneiss is a massive, non-graphitic, unaltered pelitic gneiss unit.

Mineralization at Phoenix generally occurs at the Athabasca unconformity with basement rocks at depths ranging from 390 m to 420 m. It is interpreted to be structurally controlled by the northeast-southwest trending (055º azimuth) WS Fault which dips -55º to the southeast on the east side of the quartzite ridge (Figure 7-5).

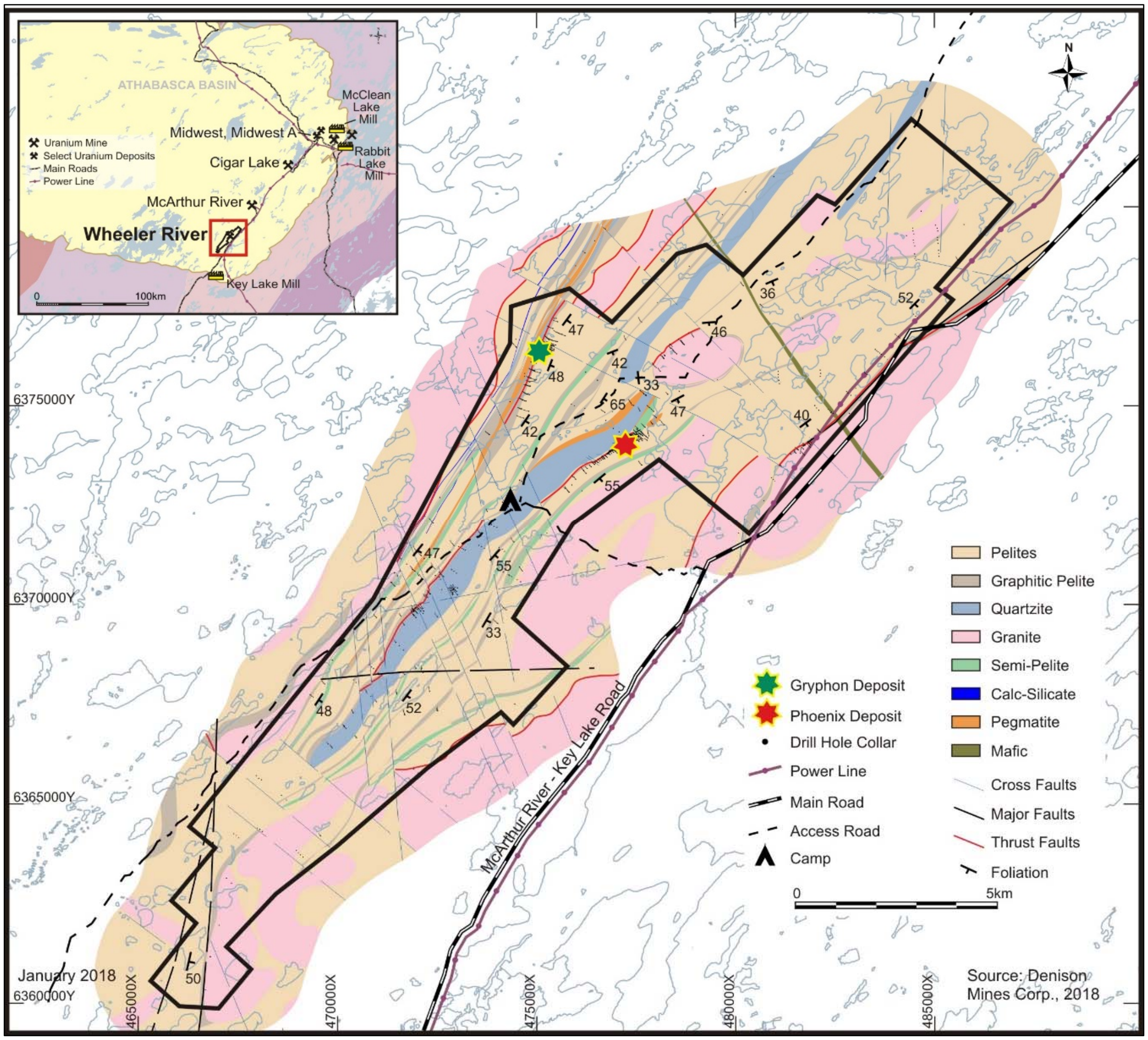


Figure 7-4. Wheeler River Property Basement Geology

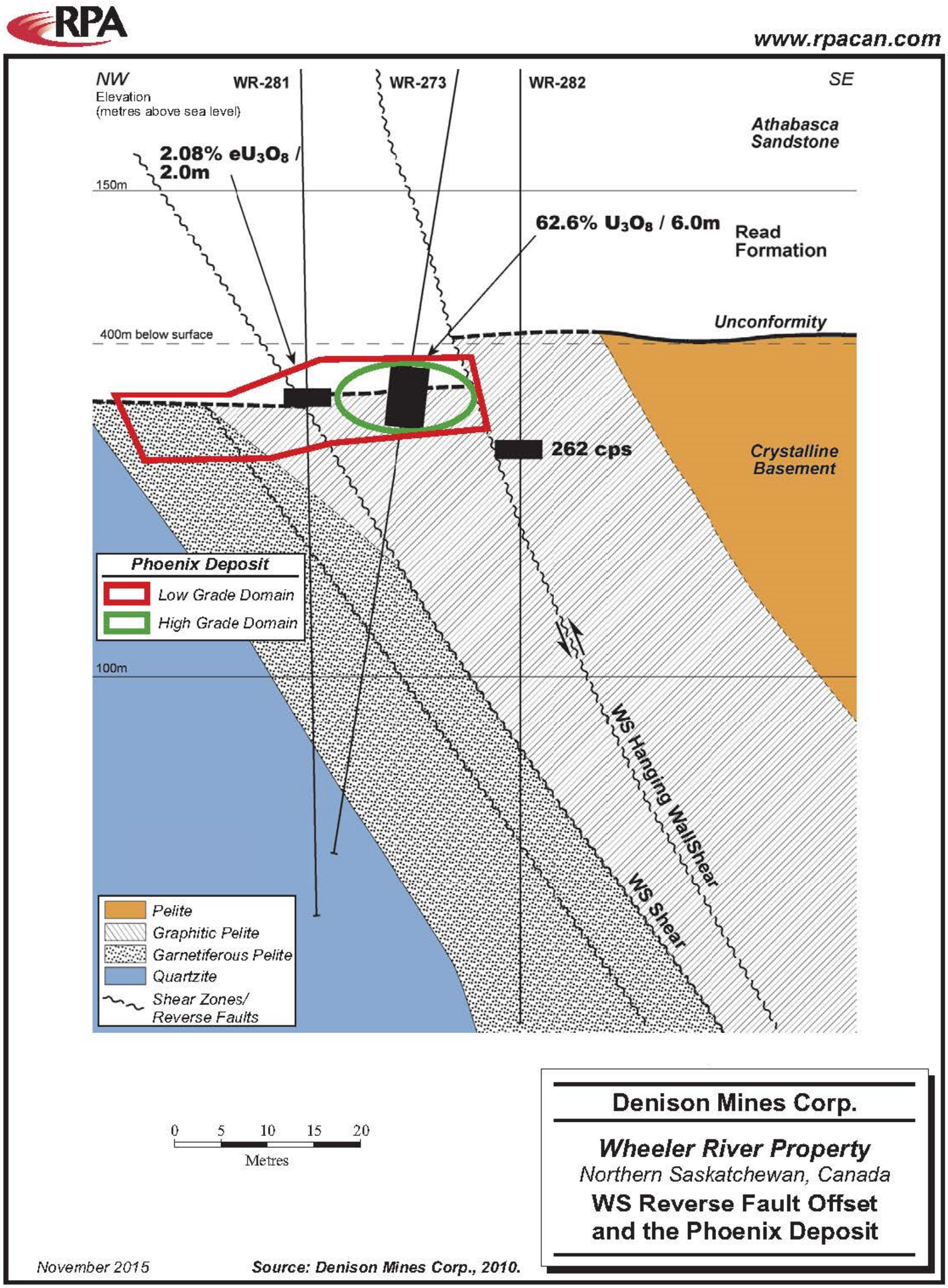


Figure 7-5. WS Reverse Fault and the Phoenix Deposit

7.3.6 Gryphon Deposit

The geology of the Gryphon deposit comprises highly deformed crystalline basement rocks overlain by the relatively undeformed Athabasca sandstone. There are four main sandstone members of the Manitou Falls ("MF") Formation present (from youngest to oldest): MFd, MFc, MFb, and the Read Formation. At the Gryphon deposit, the thickness of the Athabasca sandstone cover ranges from 480 m in the southeast to 540 m in the northwest. The unconformity surface down-drops in a series of steps to the northwest. There is approximately 60 m of vertical displacement over 250 m across strike.

Four major basement lithological units have been defined at Gryphon which dip moderately to the southeast (Figure 7-6):

1. Upper Graphite – The Upper Graphite is approximately 110 m thick, occurs furthest stratigraphically to the southeast, and is located hangingwall to the mineralization. The A and E series of mineralized lenses occur at the base of the unit along a major fault zone, the G-Fault. This pelitic gneiss unit averages 5% to 8% graphite in the upper portion of the unit grading to 10% to 15% in the lower portion of the unit. The unit is well foliated and strikes at 022° dipping at 50° to the southeast.

2. Quartz-Pegmatite Assemblage – Stratigraphically below the Upper Graphite is the Quartz-Pegmatite Assemblage, interpreted to be zone of silicification either pre- or syn-mineralization. This unit is approximately 55 m thick and consists of several smaller (five meter to nine meter) discrete sub-units of alternating quartzite, quartz-rich pegmatite, pegmatite, and graphite-bearing pelitic gneisses. The unit hosts the B series of mineralized lenses which occur along foliation-parallel faults related to the G-Fault.

3. Lower Graphite – Underlying the Quartz-Pegmatite Assemblage is the Lower Graphite. This pelitic gneiss unit is approximately 15 m thick and averages 10% to 15% graphite. It is well foliated and strikes approximately 022° and dips 45° to the southeast and is host to the C series of mineralized lenses which are interpreted to occur along foliation-parallel faults related to the G-Fault or within tensional fractures.

4. Basal Pegmatite – Stratigraphically below the Lower Graphite is the Basal Pegmatite. This is a pegmatite to coarse grained granitic unit which is competent and relatively unaltered. Within the Basal Pegmatite, there are multiple minor (1 m to 2m) variably-graphitic pelitic gneiss intervals. The pelitic gneiss intervals pinch and swell along strike and no not maintain a continuous thickness throughout the deposit area. The D series of mineralized lenses occurs within this unit within tensional fractures within the pegmatites/granites or concordant with the variably-graphitic pelitic gneisses.

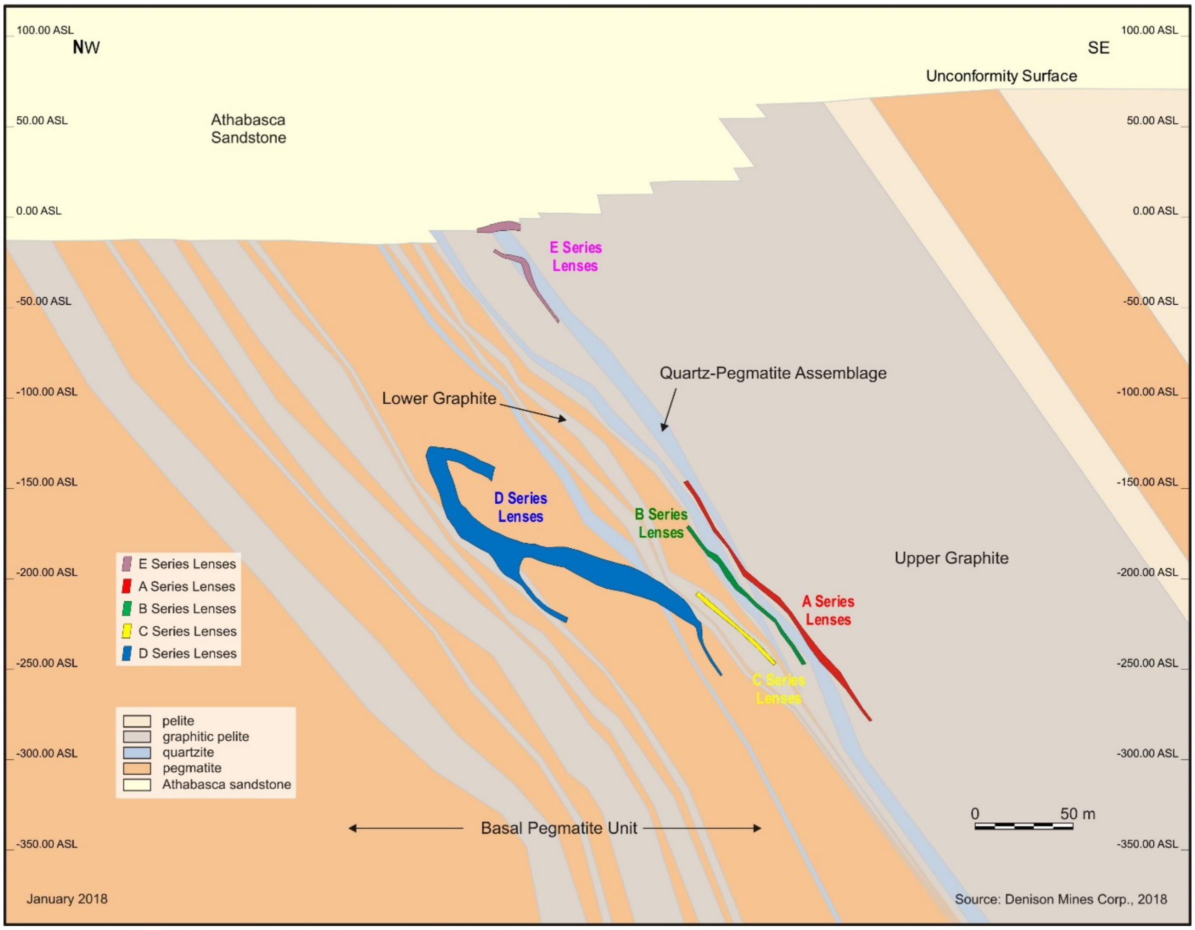


Figure 7-6. Gryphon Representative Cross-section

7.4 Alteration

7.4.1 Phoenix Deposit

At Phoenix, typical unconformity-associated alteration is evident, with a form and nature similar to other Athabasca Basin unconformity-associated deposits. The sandstones are altered for as much as 250 m above the unconformity and exhibit varying degrees of silicification and desilicification (which causes many technical drilling problems), as well as dravitization, kaolinitization, chloritization, and illitization. In addition, hydrothermal hematite and druzy quartz are present in the sandstone and commonly in the basement rocks. Alteration is focused along structures propagating upward from the WS shear and associated splays, and probably does not exceed 100 m width across strike, making this a relatively narrow exploration target. The basement in the northeast part of the Phoenix deposit is much more extensively bleached and clay altered than that to the southwest.

Sandstone alteration is typically much stronger and widely distributed above Zone D and Zone A associated with a reduced environment indicated by the strong presence of sooty pyrite. Alteration diminishes in intensity along strike to the southwest. Sandstone alteration above Zone B and Zone C, in general, is half the amplitude and intensity of Zone A with a less pronounced damage zone above the unconformity. Zone B and Zone C also exhibit a pronounced oxidized environment as indicated by the strong presence of hydrothermal hematite primarily overprinting the basement sequence directly underlying the unconformity.

7.4.2 Gryphon Deposit

At Gryphon, alteration in the Athabasca sandstone is quite variable relative to the basement-hosted mineralization. Directly above Gryphon, the typical alteration sequence above the unconformity (from surface to the unconformity) is described as follows:

- The upper 100 m to 150 m of sandstone is typically weakly bleached and silicified (interpreted as a regional feature);
- From approximately 150 m to 440 m from surface, there is no significant alteration. Diagenetic hematite banding is predominant; and
- From approximately 440 m to 540 m from surface, variable amounts of alteration occur, which include:
  - Moderate bleaching, irregular bands of hydrothermal hematite, and patchy silicification from 490 m to 540 m;
  - Pervasive silicification and strong dravitic interstitial clays from 515 m to 540 m; and
  - Alternating silicification and desilicification with strong grey alteration, pyrite development, and dravite rich breccias from 440 m to 540 m.

Sandstone alteration is generally lacking in the hangingwall (southeast) to the Gryphon mineralization and exhibits a background dickitic signature, although drill holes that intersected an up-faulted basement exhibit moderate silicification with preserved diagenetic hematite.

Sandstone alteration in the footwall (northwest) to the Gryphon mineralization consists of isolated alteration zones with strong bleaching, grey alteration, silicification, and vuggy quartz that occur upwards of 60 m above the unconformity. Footwall sandstone is also dominated by a strong kaolinitic signature with moderate amounts of dravite, primarily controlled by basement structural splays propagating into the sandstone. Although sandstone alteration in the footwall area of the Gryphon deposit exhibits strong visual and clay alteration, its geochemical signature is much less pronounced with sandstone uranium partial values seldom exceeding 1 ppm. These isolated zones of alteration are assumed to be related to the up-dip projection of the offsetting basement reverse faults to the southeast, notably the G-Fault itself and associated hangingwall splays. The Gryphon E series of mineralized lenses occurs at the intersection of the G-Fault and the unconformity and directly underlies the structurally disrupted zone of sandstone alteration.

Directly below the unconformity and distal to basement structures, the typical paleoweathering profile is preserved. The basement paleoweathering profile is gradually overprinted by various forms and intensities of hydrothermal alteration proximal to the various structures associated with Gryphon.

Basement clay alteration exhibits a zoned sequence around mineralization associated with the various mapped structures and varies in intensity in relation to each series of mineralized lenses and the host lithology. Notably stronger and widely distributed alteration sequences are present around the A and B series lenses with less intense and pronounced alteration noted in the vicinity of the C, D, and E series lenses. There is no direct correlation between intensity of alteration and uranium grade.

Distal alteration associated with Gryphon mineralization includes weak chlorite and sericite. A distinct halo of phengite is also present hangingwall to the G-Fault and footwall to the Basal Fault, essentially indicating an oxidized and relatively weak to unaltered envelope surrounding the Gryphon mineralizing system.

Proximal alteration signatures associated with the Gryphon series of lenses include various amounts of weak to strong bleaching, dravite and druzy quartz formation. There is a distinct zonation of cordierites with progressively stronger alteration proximal to mineralization. Distal to mineralization cordierites are weakly altered and exhibit a characteristic blue-green phengitic illite-chlorite clay partly replacing the cordierite itself. Proximal to mineralization the cordierites are replaced by a brown muscovitic illite and weak chlorite pseudomorphs, which are generally stretched and elongated along foliation. A distinct halo of paragonite surround the mineralization proximal to the G-Fault and Basal Faults, being indicative of a reducing environment. Quartz flooding and silicification is quite common proximal to high-grade mineralization. Intense pervasive silicification, which variably is destructive to basement rock textures, occurs within two to ten meters of mineralization and has a close spatial associated with the G-Fault and Basal Fault. Silicification is locally associated with pink silica and pink sericite which is interpreted to be a product of active beta decay, which produces visible spectral absorptions and changes in refractive index. Clay-sericite also exhibits a distinct zonation around mineralization at Gryphon. Distal to mineralization green sudoite generally replace subhedral feldspars. Medial to mineralization feldspars are replaced by a 'whispy' paragonitic white sericite grading to an intense pervasive white dravite-illite-kaolinite alteration proximal to mineralization. The latter is especially prominent along the Basal Fault in proximity to the D series mineralized lenses.

7.5 Structural Geology

The Wheeler River property lies in the Wollaston Domain, a northeast trending fold and thrust belt with recumbently folded, early Paleoproterozoic, Wollaston Supergroup metasedimentary rocks intercalated with granitoid gneisses, some of which are of Archean age.

Numerous hypothetical structural models have been proposed for the property. The simplest model infers a southeast dipping homocline. The presence of mechanically competent quartzite units, as well as the bounding units of competent granitoid gneiss, together with the many kilometers of relatively incompetent graphitic pelitic gneiss provides a situation for the extensive development of thrust and strike slip/wrench fault tectonics, as well as later normal faults, at competent/incompetent interfaces (Liu et al., 2011). A northwesterly trending diabase dyke, probably part of the 1.27 Ga Mackenzie dyke swarm, cuts across the sandstones on the northern part of the property.

7.5.1 Phoenix Deposit

The major structural feature at the Phoenix deposit is the northeast-southwest trending (055º azimuth) WS reverse fault which dips -55º to the southeast and lies within or at the base of the graphitic pelitic gneiss unit along the east edge (hangingwall) of the quartzite ridge, which appears to have acted as a buttress for thrusting and reverse faulting (Kerr, 2010; Kerr et al., 2011). Deformation within the WS Fault has occurred partly by ductile shearing, but mainly by fracturing. A progressive sequence of fracturing is evident by variations in the strike and dip of slickensides. The principal stress directions responsible for early deformation were northwest-southeast. A change in the principal stress to an east-west direction led to later strike-slip movement along the WS shear. Later extension is indicated by northwest-striking normal faults, which dip steeply to the southwest.

With the limited structural data currently available (as the majority of drill holes were vertical), it appears that the WS structure was most active during deposition of the Read Formation; however, continued uplift is indicated by westward tilting of MFc strata along the fault zone. Reverse fault displacements on the western edge of the quartzite ridge occurred primarily within the highly resistant quartzite unit. Within the Wheeler River area, vertical offset on the footwall of the quartzite unit can be as much as 60 m; however, at the Phoenix deposit, known vertical displacements in the hangingwall sequence are always less than 10 m (Figure 7-5).

Mineralization hosted in the lower 15 m of the Athabasca sandstone appears to have some relationship to the extensions of the WS Fault and its various hangingwall splays; hence, movement on these faults must have continued after deposition of rocks of the Read Formation and probably the MFd member of the Manitou Falls Formation. The WS Fault and its various interpreted hangingwall splays may have been the main conduit for the mineralizing fluids. Thus, determining favourable locations along the WS Fault, where zones of long-lived permeability are present, is of critical importance. Five east-west oriented cross faults or tear faults are also observed at Phoenix. These features are not well documented in core as the majority of the structures have been replaced by high-grade mineralization. They are inferred by changes in geologic strike or flexures in the geology underlying the deposit. These cross faults are believed to have enhanced the permeability of select portions of the deposit during deposition, subsequently allowing for the formation of thicker and high-grade uranium mineralization.

7.5.2 Gryphon Deposit

On a property scale, the Gryphon deposit is situated within a dilation jog or releasing bend along the K-North trend, a highly prospective northeast striking metasedimentary corridor along the Wheeler River property's northwest boundary. Regionally the K-North trend geology strikes 035° to the northeast and dips moderately at -50° to the southeast. In the immediate vicinity of Gryphon, there is a prominent change in geologic strike from the regional 035° to 020°. The mineralization at Gryphon is interpreted to have formed from the mixing of oxidized basinal uraniferous fluids with reduced basement ferrous fluids resulting in the co-precipitation of uraninite and hematite. To facilitate this mixing of fluid within the basement, a dilational structural setting is required to allow for the ingress of basinal fluids. It is interpreted that the subtle change in strike, or jog, coupled with the regional northwest directed compression allowed for basement dilation at Gryphon. This is supported by core observations which support a reverse-sinistral sense of movement proximal to the deposit.

On a deposit scale, the plunge of the deposit to the northeast is controlled by structural dilation as a result of reverse-sinistral faulting over shallower foliation dips. Higher grades and thicknesses tend to correspond with larger fault displacements. Five main fault groups are recognized, though several other minor faults are also present throughout the deposit area (Figure 7-7). These structures are generally located at the contact between the less competent graphitic pelitic gneisses and more competent quartz-pegmatites, pegmatites, and pelitic gneiss units. The faults are brittle in nature and can be described as a combination of cataclasites and gouges, and intervals of blocky and friable core.

1. The Offset Fault and associated splays occur at the contact with the Upper Graphite and its overlying pelitic gneiss. It is interpreted to be conformable with the local geology having a strike of 020° and dip of -50°. The Offset Fault and its associated splays are responsible for over 60 m of known unconformity displacement. The unconformity is displaced downward to the northwest in a series of steps over a 100 m cross strike distance. To date, no mineralization has been found to be associated with the Offset Fault.

2. The G-Fault and associated splays occur at the lower contact of the Upper Graphite unit and its underlying Quartz-Pegmatite Assemblage. In general, its orientation is conformable to the geology with a strike of 020° and dip of -50°. However, mineralization generally occurs along the

G-Fault and its associated fault strands where a shallowing of stratigraphic foliation is observed, between -30° and -50°. The shallowing of foliation in combination with reverse sinistral movement have provided a zone of dilation, amenable to fluid movement and uranium precipitation. Five to ten meters of unconformity displacement have been recorded along its strike. The G-Fault form the principal and most significant structure related to the Gryphon deposit.

3. The Basal Fault, subordinate to but sharing many structural characteristics with the G-Fault occurs over 200 m to the northwest of the G-Fault within the pegmatite-dominated footwall units with minor variably graphitic pelitic gneiss. Similar to the G-Fault, mineralization is associated with a shallowing of foliation, though it is less pronounced within the pegmatite-dominated sequence. No appreciable unconformity offset is associated with the subcrop of the Basal fault at the unconformity.

4. The Linkage Faults, representing tension fractures, occur within the Basal Pegmatite unit and as the name suggests link the Basal Fault and G-Fault through a network of fault splays occurring discordant to the deposit geology. It is interpreted that the Linkage Faults formed as a result of prominent reverse faulting along the G-Fault and subsequent tensional fracture development at high angles into the Basal Pegmatite unit (Riedel shear model). To date three primary Linkage Faults (or fault zones) have been identified that vary in thickness from two meters to 20 m and have a minimum strike of 50 m. They follow the deposit strike of 020° but are generally much shallower in dip, ranging from -10° to -30° to the southeast. Higher grade uranium intersections are common where the Linkage Faults intersect the G-Fault and Basal Fault but are quite variable along the Linkage Faults themselves.

5. Five cross-cutting fault zones have also been noted within the deposit area. These spatially defined zones are characterized by a high-frequency of west to northwest striking faults and fractures with steep dips of variable orientation. The zones are somewhat regularly spaced across the deposit every 100 m to 150 m. The timing and kinematics of these fault zones is not well understood; however, they are interpreted to have been reactivated over time and most commonly display a normal sense of movement. The most northeastern and southwestern sub-vertical faults appear to play a role in the morphology of the mineralized lenses, primarily the A and B series lenses. Where mineralization occurs in proximity to these sub-vertical structures its primary plunge of 30°, as observed from an inclined longitudinal section, shallows considerably to 010° to 015°, suggesting that the structures are pre- or syn-mineralization. Faults associated with these zones have also been interpreted to offset mineralization, compartmentalize mineralization, or in some cases are mineralized themselves.

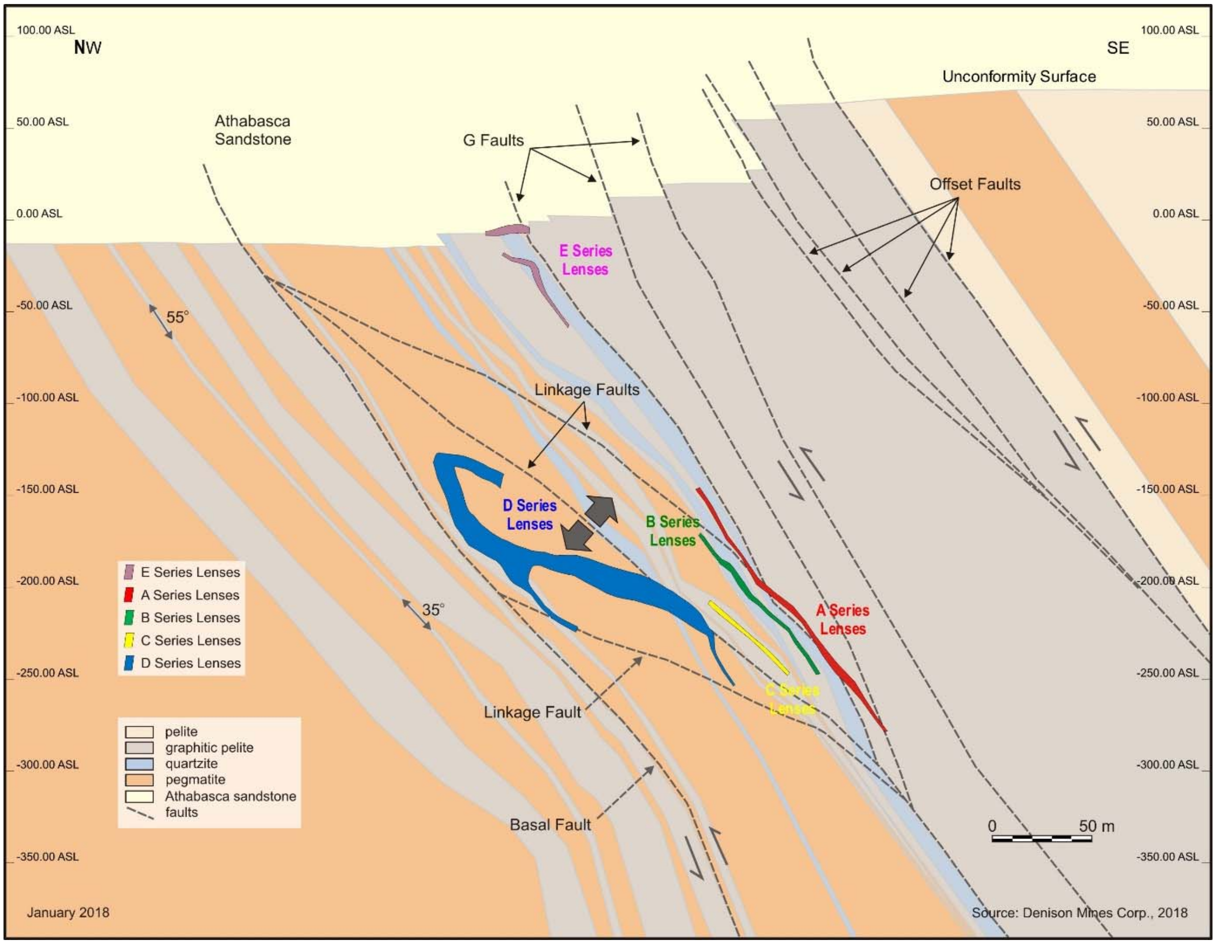


Figure 7-7. Cross-section of the Gryphon Deposit Showing Significant Interpreted Structures

7.6 Mineralization

7.6.1 Phoenix Deposit

The Phoenix uranium deposit can be classified as an unconformity-associated deposit of the unconformity-hosted variety. The deposit straddles the sub-Athabasca unconformity approximately 400 m below surface and comprises three zones (A, B, C) which cover a strike length of 1.1 km. The deposit's A and B zones comprise an exceptionally high-grade core surrounded by a lower grade shell. The deposit is interpreted to be structurally-controlled by the WS shear, a prominent basement thrust fault which occurs footwall to a graphitic-pelite and hangingwall to a garnetiferous pelite and quartzite unit. A minor amount of basement, fracture hosted mineralization is evident extending below the north part of Zone A.

Mineralization within the Phoenix deposit lenses is dominated by massive to semi-massive uraninite associated with an alteration assemblage comprising hematite, dravitic tourmaline, illite and chlorite. Secondary uranium minerals, including uranophane, and sulphides are trace in quantity.

Average trace metal concentrations for Phoenix assay samples greater than 0.2% U_3O_8 are as follows: 576 ppm Ni, 194 ppm Co, 319 ppm As, 2,092 ppm Zn, 18 ppm Ag, 7,176 ppm Cu, 9,143 ppm Pb, 266 ppm Mo and 35 ppm Se. Average concentrations of Ni, Co, and As are at the low end of the range found in other uranium deposits in the Athabasca Basin.

7.6.2 Gryphon Deposit

The Gryphon uranium deposit can be classified as an unconformity-associated deposit of the basement-hosted variety. The majority of the deposit occurs within southeasterly dipping crystalline basement rocks of the Wollaston Supergroup below the regional sub-Athabasca Basin unconformity. The deposit is located from 520 m to 850 m below surface and has an overall strike length of 610 m, dip length of 390 m and varies in thickness between two meters and 70 m, depending on the number of mineralized lenses present. The mineralized lenses are controlled by reverse fault structures which are largely conformable to the basement stratigraphy and dominant foliation. The A, B, and C series of lenses comprise stacked, parallel lenses which plunge to the northeast along a fault zone (the G-Fault) which occurs between hangingwall graphite-rich pelitic gneisses and a more competent pegmatite-dominated footwall. A ubiquitous zone of silicification ("Quartz-Pegmatite Assemblage") straddles the G-Fault and the A, B, and C series of lenses occur in the hangingwall of, within, and in the footwall of the Quartz-Pegmatite Assemblage respectively. The D series lenses occur within the pegmatite-dominated footwall along a secondary fault zone ("Basal Fault") or within extensional relay faults which link to the G-Fault. The E series lenses occur along the G-Fault, up-dip and along strike to the northeast of the A and B series lenses, within the upper basement or at the sub-Athabasca unconformity. The E series of lenses differ from the remaining sets of lenses as they are the only ones to not follow the local scale plunge of the deposit, rather the mineralization is located planar to foliation and tight to the unconformity (Figure 7-8). To date, the E series lenses are the only lenses to host unconformity mineralization at Gryphon.

Mineralization within the Gryphon deposit lenses is dominated by massive, semi-massive, or fracture-hosted uraninite associated with an alteration assemblage comprising hematite, dravitic tourmaline, illite, chlorite, and kaolinite. Secondary uranium minerals, including uranophane and carnotite, are trace in quantity.

Gangue mineralogy is dominated by alteration clays (illite, kaolinite, chlorite), dravite, and hematite with minor relict quartz, biotite, graphite, zircon, and ilmenite. Only trace concentrations of sulphides are noted comprising galena, chalcopyrite, and pyrite. Notable concentrations of molybdenum and lithium are also noted within and around the mineralization, represented visually as lepidolite and molybdenite respectively.

Average trace metal concentrations for Gryphon assay samples greater than 0.2% U_3O_8 are as follows: 107 ppm Ni, 62 ppm Co, 30 ppm As, 18 ppm Zn, 14 ppm Ag, 301 ppm Cu, 3,525 ppm Pb, 498 ppm Mo and 13 ppm Se. These concentrations are typically lower than those recorded for the Phoenix deposit.

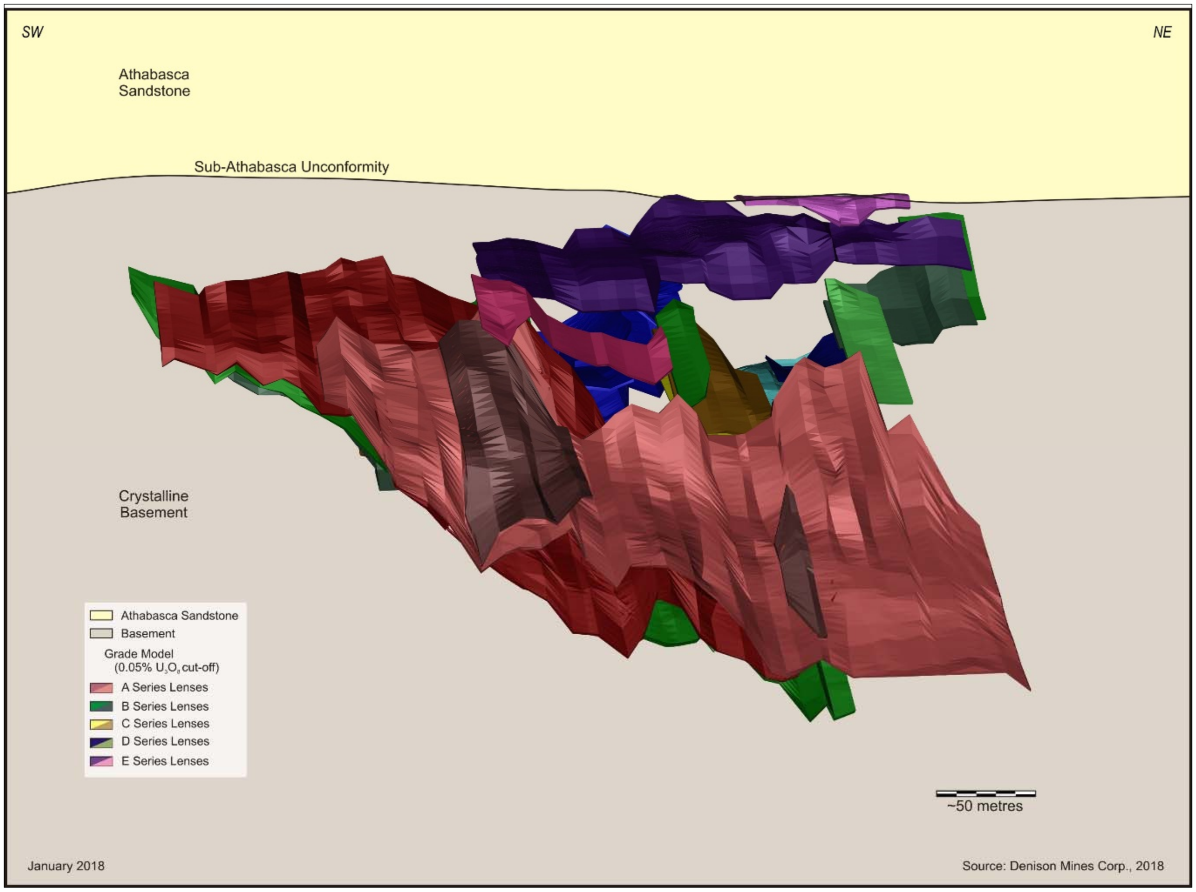


Figure 7-8. 3D Isometric Longitudinal View of the Gryphon Deposit (shown as mineralized wireframes using a 0.05% U_3O_8 cut-off and minimum thickness of 2 meters)

8 Deposit Types

Both the Phoenix and Gryphon deposits are classified as Athabasca Basin unconformity-associated (also unconformity-related and –type) uranium deposits. Phoenix straddles the unconformity contact between the Athabasca sandstone and underlying basement, while Gryphon is primarily hosted in the basement rocks with minor portions of the deposit situated at the unconformity.

Jefferson et al. (2007) offered the following definition for the geological environment of this type of mineralization:

Unconformity-associated uranium deposits are pods, veins, and semi-massive replacements consisting of mainly uraninite, close to basal unconformities, in particular those between Proterozoic conglomeratic sandstone basins and metamorphosed basement rocks. Prospective basins in Canada are filled by thin, relatively flat-lying, and apparently unmetamorphosed but pervasively altered, Proterozoic (~1.8 Ga to <1.55 Ga), mainly fluvial, red-bed quartzose conglomerate, sandstone, and mudstone. The basement gneiss was intensely weathered and deeply eroded with variably preserved thicknesses of reddened, clay-altered, hematitic regolith grading down through a green chloritic zone into fresh rock. The basement rocks typically comprise highly metamorphosed interleaved Archean to Paleoproterozoic granitoid and supracrustal gneiss including graphitic metapelitic gneiss that hosts many of the uranium deposits. The bulk of the U-Pb isochron ages on uraninite are in the range of 1,600 Ma to 1,350 Ma. Monometallic, generally basement-hosted uraninite fills veins, breccia fillings, and replacements in fault zones. Polymetallic, commonly sub horizontal, semi-massive replacement uraninite forms lenses just above or straddling the unconformity, with variable amounts of uranium, nickel, cobalt, and arsenic, and traces of gold, platinum-group elements, copper, rare-earth elements, and iron.

The uranium deposits in the Athabasca Basin occur below, across, and immediately above the unconformity, which can lie within a few meters of surface at the rim of the Basin, to over 1,000 m deep near its centre. The deposits formed by extensive hydrothermal systems occurring at the unconformity's structural boundary between the older and younger rock units. Major deep-seated structures are also interpreted to have played an important role in the hydrothermal process, likely acting as conduits for hot mineralized fluids that eventually pooled and crystallized in the structural traps provided by the unconformity. One of the necessary reducing fluids originates in the basement and flows along basement faults. A second, oxidizing fluid originates within the Athabasca sandstone stratigraphy and migrates through the inherent porosity. In appropriate circumstances, these two fluids mix and precipitate uranium in a structural trap at or near the basal Athabasca unconformity with basement rocks.

Two end-members of the deposit model have been defined (Quirt, 2003). A sandstone-hosted egress-type model (i.e. Midwest A) involved the mixing of oxidized, sandstone brine with relatively reduced fluids issuing from the basement into the sandstone. Basement-hosted, ingress-type deposits (i.e. Rabbit Lake) formed by fluid-rock reactions between oxidizing sandstone brine entering basement fault zones and the local wall rock. Both types of mineralization and associated host-rock alteration occurred at sites of basement–sandstone fluid interaction where a spatially stable redox gradient/front was present.

Although either type of deposit can be high-grade, ranging in grade from a few percent to 20% U_3O_8, they are not volumetrically large and typically occur as narrow, linear lenses often at considerable depth. In plain view, the deposits can be 100 m to 150 m long and a few meters to 30 m wide and/or thick. Egress-type deposits tend to be polymetallic (U-Ni-Co-Cu-As) and typically follow the trace of the underlying graphitic pelitic gneisses and associated faults, along the unconformity. Ingress-type, essentially monomineralic U deposits, can have more irregular geometry.

Unconformity-type uranium deposits are surrounded by extensive alteration envelopes. In the basement, these envelopes are generally relatively narrow but become broader where they extend upwards into the Athabasca Group for tens of meters to even 100 m or more above the unconformity. Hydrothermal alteration is variously marked by chloritization, tourmalinization (high boron, dravite), hematization (several episodes), illitization, silicification/desilicification, and dolomitization. Modern exploration for these types of deposits relies heavily on deep-penetrating geophysics and down-hole geochemistry.

Since the discovery of Key Lake in 1975-1976, the Key Lake exploration model has emphasized the occurrence of uranium mineralization proximal to the sub-Athabasca unconformity at locations where graphitic pelite units in the basement meet the basal Athabasca sandstone. The graphitic pelite units are commonly intensely sheared in contrast to the physically more competent rock types that include non-graphitic pelite, semi-pelite, psammite, meta-arkose, or granite gneiss. Airborne and ground electromagnetic systems are commonly used to map conductive graphitic pelite units versus the relatively resistive and non-conductive quartz-feldspathic rock types.

However, since the discovery of the McArthur River deposit in 1988, the McArthur River exploration model has emphasized the importance of basement quartzites occurring in proximity to uranium mineralization. Highly competent quartzites provide a strong rheological contrast to other metasediments and therefore control the sites of major thrust, reverse, and strike-slip faults. Although these faults are loci for mineralization, the poor conductivity, low magnetic susceptibilities, and specific gravity (density) values associated with quartzite, as well as other quartz-feldspathic rocks, limits the effectiveness of airborne and ground geophysical methods in mapping these basement units. This is particularly so when they are covered by hundreds of meters of Athabasca sandstone. Alteration haloes are typically larger than the deposit footprints and are characterized by changes in mineralogy and major and trace elements. Therefore, the detection of alteration halos through geophysics, primarily DC resistivity surveys, and drill core lithogeochemistry and reflectance spectrometry, have become increasingly important exploration methodologies.

Recently, basement-hosted deposits have become more recognized as a viable exploration target through the development of Eagle Point mine and the discovery of deposits such as Millennium, Triple R, and Arrow. Exploration typically requires the recognition of significant fault zones within basement metasediments (often associated with graphite) with associated clay and geochemical alteration haloes.

Figures 8-1 and 8-2 illustrate various models for unconformity-type uranium deposits of the Athabasca Basin. The geology of both the Phoenix and the Gryphon deposits and the controls on mineralization are sufficiently well understood for mineral resource estimation, in RPA’s opinion.

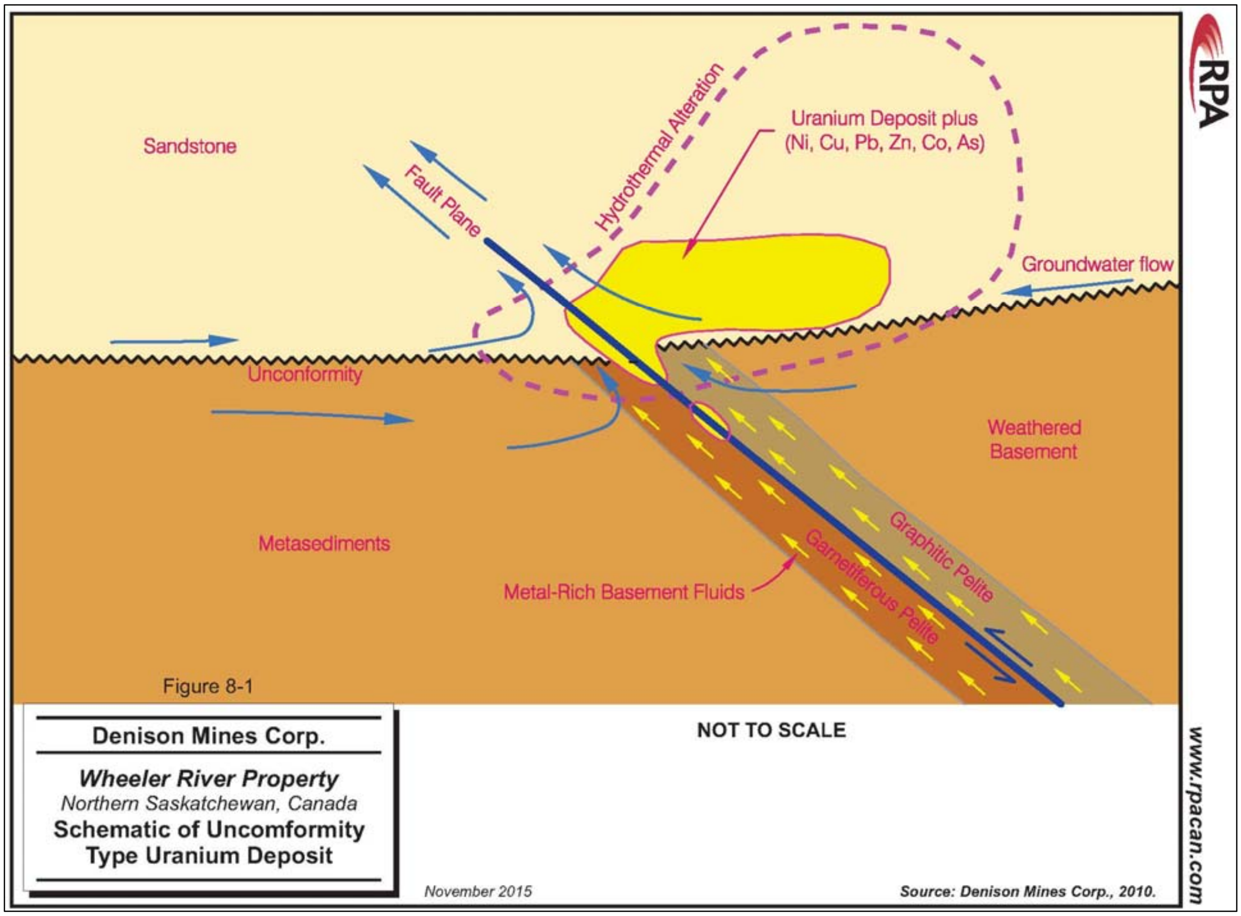


Figure 8-1. Schematic of Unconformity Type Uranium Deposit

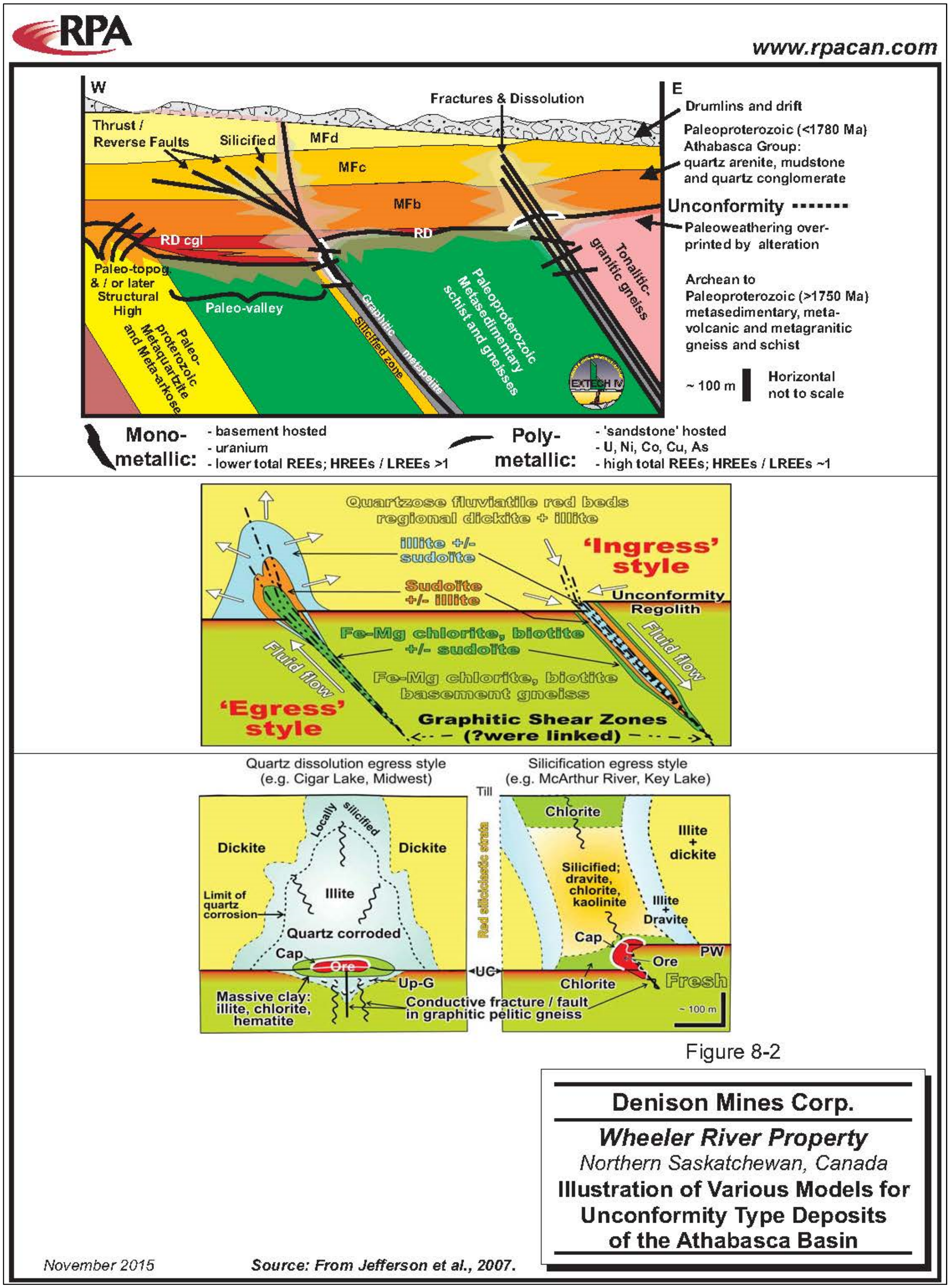


Figure 8-2. Various Models for Unconformity Type Deposits of the Athabasca Basin

9 Exploration

With the exception of drilling, exploration work performed on the property by Denison since 2008 is summarized in this section. Work completed on the property and its immediate vicinity by other parties prior to 2008 is summarized in Section 6 of this report. Drilling completed on the Phoenix and Gryphon deposits is summarized in Section 10.

9.1 Ground Geophysical Surveys

9.1.1 2009 Induced Polarization Survey

Following the discovery of the Phoenix deposit in 2008, Denison, as operator of the Wheeler River joint venture, completed DC Resistivity/IP surveys comprising 60.2 line-km in 2009.

9.1.2 2010 Transient Electromagnetic (TEM) Survey

During February and March 2010, a geophysical program consisting of 25.2 km of a fixed loop surface TEM survey and 51.0 km of a step loop TEM survey was completed on three lines of the previously established 2007 Wheeler River grid. Three lines of step-wise moving loop (SWML) TEM surveying was completed on three previously defined resistivity anomalies in an attempt to better define any conductive axis associated with graphitic basement features that could act as conduits for mineralizing events. The resistivity signature located on L40+00N is known to be associated with the uranium mineralization associated with the Gryphon deposit.

9.1.3 2011-2012 Induced Polarization Survey

The 2011 exploration program on the property carried out by Denison included a 120.6 line-km Titan 24 DC/IP survey. Additional Titan 24 surveying (48.8 line-km) was completed in 2012.

9.1.4 2013 Induced Polarization Survey

In 2013, the WRJV completed a 127.0 line-km Titan 24 DC/IP survey over two areas previously not covered (R North and K West areas).

9.1.5 2014 Induced Polarization, Gravity, and SWML EM Surveys

Geophysical exploration in 2014 consisted of the following work, with primary focus being the K-North area and its close vicinity:

- 46.05 line-km over three lines of infill SWML EM in the K-North area to complete areas previously not covered.
- 43 line-km over two lines of SWML in the WS South area covering areas of interest from the 2013 Titan 24 DC/IP survey.
- 48 line-km of ground gravity covering the O Zone, where historic drilling showed a large unconformity offset with weak uranium mineralization.
- A 52.0 line-km ground gravity survey was carried out in 2014 over the K-North area to test if the unconformity offset seen in drill core could be defined by this method.
- A 67.2 km extension of the 2007 North Titan 24 DC/IP survey to complete the coverage over the K-North area.
- A 3D DC/IP survey to attempt to resolve a 2 km by 2 km geologically/geophysically complex area north of Phoenix Zone A.

9.1.6 2015 Induced Polarization Survey

In 2015, the WRJV completed a 149.5 line-km Titan 24 DC/IP survey over two areas previously not covered (O Zone and the southern parts of the K and Q Zones).

9.1.7 2016 Induced Polarization, Gravity, and Borehole Surveys

In 2016, geophysical surveys were conducted as follows:

- 42.0 km of Infill Gravity Survey on WR-16-G2 grid by MWH Geo-Surveys Ltd. The objective of this work was to develop a density model that was consistent with physical property constraints including wireline density logs, a geological model built from large amounts of drilling data, and two types of gravity data with overlapping coverage. The work demonstrates a solution to a very complex constrained gravity inversion problem.
- 83.3 km of DC-IP Deep Earth Imaging survey on the WR-16-G1 grid by Quantec Geoscience Ltd Spartan. The exploration objectives were to map and detect alteration related to unconformity-type uranium mineralization within the project area for drill targeting, delineation and structural control identification.

9.1.8 2017 Borehole Surveys

DGI Geoscience Inc. conducted down hole logging of physical properties including; density, acoustic velocity, magnetic susceptibility, natural gamma, fluid temperature, apparent resistivity, and neutron on 13 historical drill holes. The main objectives of this project were to quantitatively domain boreholes using rock properties and geochemical data, to expand knowledge of geophysical rock properties to other boreholes where petrophysical data was not acquired, and to extract new value and insights from geophysical and geochemical data.

9.2 Airborne Surveys

9.2.1 2013 VTEM Survey

In 2013, a helicopter borne versatile time-domain electromagnetic (VTEM)-magnetic-radiometric survey was conducted over the property. The survey comprised 990 line-km at a 300 m line-spacing covering an area of approximately 249 km^2. This survey used a larger loop than previously in an attempt to remove noise that caused difficulties in interpretation of a previous survey.

10 Drilling

Diamond drilling on the Wheeler River property is the principal method of exploration and delineation of uranium mineralization after initial geophysical surveys. Drilling can generally be conducted year-round on the property. Drill holes on the property are labelled with a prefix of the project name, WR, followed by the hole number.

Since 1979, a total of 810 diamond drill holes and 84 reverse circulation (RC) drill holes totalling 393,881 m have been completed within the property (Table 10-1). The following sections provide details of the holes drilled on the Phoenix and Gryphon deposits.

Table 10-1. Wheeler River Property Drilling Statistics

Year	Company	# Diamond Drill Holes (including wedge holes and re-starts)	# Rotary Drill Holes	Total Drilled (m)
1979	AGIP Canada Ltd.	6	0	2,111
1980	AGIP Canada Ltd.	6	0	1,968
1981	AGIP Canada Ltd.	14	0	5,352
1982	AGIP Canada Ltd.	13	0	4,974
1983	AGIP Canada Ltd.	9	0	2,255
1984	AGIP Canada Ltd.	13	0	2,986
1985	SMDC	13	0	3,395
1986	SMDC	11	0	4,174
1987	SMDC	12	23	6,362
1988	SMDC	12	0	5,882
1989	SMDC	9	0	4,617
1995	Cameco	4	0	1,890
1996	Cameco	5	0	2,544
1997	Cameco	7	0	3,218
1998	Cameco	7	0	3,074
1999	Cameco	3	0	1,263
2001	Cameco	2	0	1,213
2002	Cameco	4	0	2,099
2003	Cameco	4	61	3,470
2004	Cameco	1	0	494
2005	Denison Mines Inc.	12	0	4,837
2006	Denison Mines Inc.	27	0	10,514
2007	Denison Mines Corp.	18	0	6,147
2008	Denison Mines Corp.	14	0	6,104
2009	Denison Mines Corp.	43	0	18,950
2010	Denison Mines Corp.	60	0	28,264
2011	Denison Mines Corp.	80	0	38,428
2012	Denison Mines Corp.	58	0	26,810
2013	Denison Mines Corp.	52	0	25,656
2014	Denison Mines Corp.	50	0	30,833
2015	Denison Mines Corp.	72	0	42,243
2016	Denison Mines Corp.	78	0	47,199
2017	Denison Mines Corp.	91	0	44,556
TOTAL		**810**	**84**	**393,881**

10.1 Phoenix Deposit Exploration Drilling

During the summer of 2008, WR-249 was drilled on geophysics line 4300 to test resistivity target "A". WR-249 was spotted 90 m northwest of WR-190A, which had been lost in the sandstone 34 m above the unconformity in 2003. The hole encountered strong desilicification, silicification, hydrothermal hematite, druzy quartz, and increased fracture density, with progressively more intense alteration towards the unconformity, together with a strong grey bleached zone consisting of extremely fine-grained pyrite which provided a strong visual contrast to bleached zones in other nearby holes. At the unconformity, disseminated and massive uranium mineralization was present from 406.65 m to 409 m. The assay grade was 1.06% U_3O_8 over 2.35 m. This was the highest grade intercept on the property to date. This hole was located seven kilometers northeast of the previous work in the WR-204 area and, more significantly, was drilled on the hangingwall rather than the footwall side of the quartzite ridge.

Target "B" was tested by WR-251, which was located 600 m along strike from WR-249. It intersected similar alteration along with three mineralized zones occurring both at the unconformity and in the basement. The best intersection graded 0.78% U_3O_8 over 2.25 m.

All 2008 follow-up drilling was located in the WR-251 area. Additional uranium mineralization (1.4% U_3O_8 over 4.0 m and 1.75% U_3O_8 over 0.5 m) was intersected in WR-253, which was drilled to test for mineralization 15 m to the southeast of WR-251.

All drill holes completed during the summer of 2008 intersected either uranium mineralization or very strong alteration located in the hangingwall to the quartzite unit. This new discovery was termed Phoenix.

During 2009, three drill programs, consisting of a total of 31 diamond drill holes (114,549 m), were carried out, each of which established significant milestones in the advancement of the property. During the winter program, the first indications of higher grade mineralization came from hole WR-258, which returned 11.8% U_3O_8 over 5.5 m from a depth of 397 m. The summer drill program continued to test the Phoenix discovery, with hole WR-273 returning a value of 62.6% U_3O_8 over 6.0 m at a depth of 405 m. Mineralization was monomineralic pitchblende with very low concentrations of accessory minerals and was reported to be remarkably similar to the high-grade McArthur River P2 deposits. Most of the mineralization occurs as a horizontal sheet at the base of the Athabasca sandstone proximal to where a graphitic pelitic gneiss unit in the basement intersects the unconformity. In addition, the alteration changes to the northeast with intense and strong basement bleaching becoming more prominent, and the strongest graphitic faulting observed. More significantly, the new mineralized zone returned the highest grades intersected in more than 40 years of continuous exploration on the property.

A further drill program in the fall of 2009 established continuity of the high-grade portion of the mineralized zone and extended the overall zone as a possibly continuous unit for a strike length of greater than one kilometer.

During 2010, 62 diamond drill holes totalling 28,362.3 m were carried out on two claims along the Phoenix deposit trend. Of the 62 drill holes, 55 totalling 25,949 m were completed to the desired depth. Twenty-seven holes were drilled on claim S-98341 during two drill seasons from January to April and June to August. Thirty-five holes were drilled on claim S-97909 during two drill seasons from January to April and June to August. The two-phase drilling program was carried out during the periods of January to April 2010 and June to August 2010.

During 2011, a two-phase drilling program of 80 diamond drill holes totalling 38,426.6 m was carried out on mineral dispositions S-97908, S-97909, and S-98341. Of the 80 drill holes completed, 63 totalling 29,988 m were successfully completed to design depth at Phoenix.

During 2012, Denison completed 51 diamond drill holes totalling 23,073 m on the Phoenix deposit during two drilling campaigns.

In 2013, 53 diamond drill holes totaling 25,651 m were carried out on mineral dispositions across the property of which 18 totalling 8,270 m were completed as infill delineation drilling on Phoenix Zone A.

In 2014, an additional nine diamond drill holes were completed on Phoenix Zone A to extend higher grade portions of the deposit.

In 2016, three diamond drill holes were completed on Phoenix Zone A to test the ground condition of the proposed Phoenix deposit infrastructure.

In 2017, five diamond drill holes were completed on Phoenix Zone A to collect samples for metallurgical testing as well as test the ground conditions of proposed Phoenix deposit infrastructure.

Since 2008, 251 drill holes totalling 115,948 m of drilling have delineated the Phoenix deposit (Figure 10-1 and Table 10-2). Well-established drilling industry practices were used in the drilling programs.

Table 10-2. Phoenix Drilling Statistics

Deposit	Year	Company	# Holes	Total Drilled (m)
Phoenix	2008	Denison	14	6,499
	2009	Denison	31	14,549
	2010	Denison	55	25,949
	2011	Denison	63	29,988
	2012	Denison	51	23,073
	2013	Denison	18	8,270
	2014	Denison	9	3,791
	2015	Denison	2	1,557
	2016	Denison	3	1,748
	2017	Denison	5	524
Phoenix Total			**251**	**115,948**

Target	# Holes	Total Drilled (m)
Zone A	137	63,202
Zone B	55	25,347
Zone C	24	10,438
Zone D	27	15,214

To date, the Phoenix deposit area has been systematically drill tested over approximately one kilometer of strike length at a nominal 25 m to 50 m section spacing (Figure 10-1).

Delineation diamond drilling at Phoenix was primarily done with NQ sized core (47.6 mm diameter) in holes WR-249 through WR-275 and HQ sized core (63.5 mm diameter) reducing down to NQ at 350 m in holes WR-276 through WR-561A, with most holes successfully penetrating into the basement. In general, drilling in the higher grade areas of the Phoenix deposit has been conducted on a nominal drill hole grid spacing of 25 m northeast-southwest by 10 m northwest-southeast. Some additional infill holes were drilled primarily to test the spatial continuity of the mineralization. The most notable results from drilling to date are the intersections of 6.0 m of 62.6% U_3O_8 in hole WR-273, 3.5 m of 58.2% U_3O_8 in hole WR-305, 8.4 m of 38.4% U_3O_8 in hole WR-401, and 10.5 m of 50.1% U_3O_8 in hole WR-525. The bulk of the flat lying high-grade mineralization is positioned at and sub-parallel to the unconformity.

All holes were logged for lithology, structure, alteration, mineralization, and geotechnical characteristics. Data were entered into DHLogger software on laptops in the field. The DHLogger data was transferred into a Fusion database. All drill hole data was validated throughout the drilling program and as an integral component of the current recent resource estimation work. Hard copies of drill logs are stored at site.

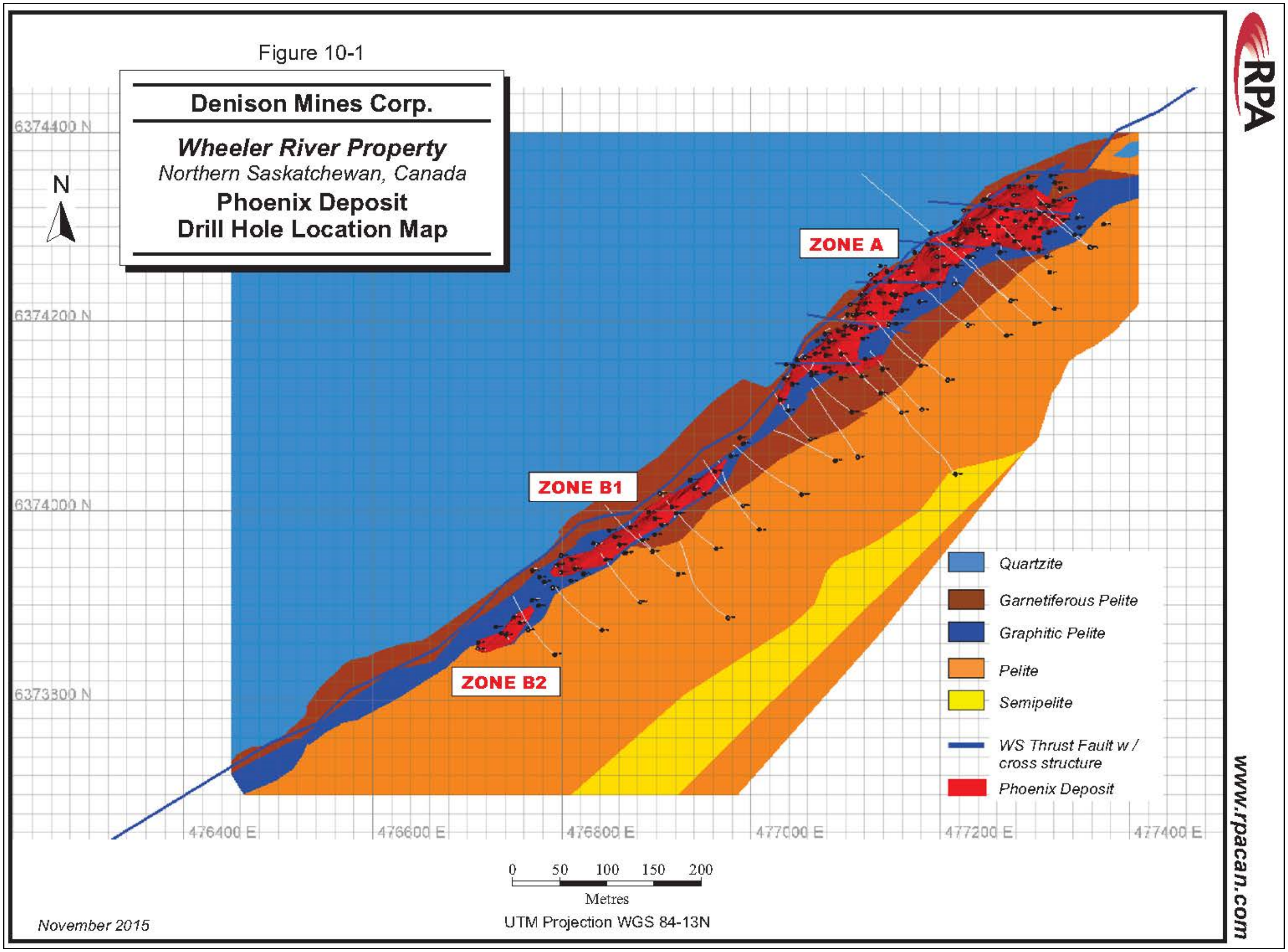


Figure 10-1. Phoenix Deposit Drill Hole Location Map

10.2 Gryphon Deposit Exploration Drilling

The first exploration drilling in the Gryphon area began in 1988 and continued intermittently through 2013.

In 2013, Denison drilled two holes, WR-507D1 and WR-509. WR-507D1 was drilled approximately 40 m up dip on section northwest of hole ZK-23, to test for more favourable geology (Figure 10-2). No significant mineralization was intersected at the unconformity or in the basement, but similar lithological units and structure were intersected which hosted mineralization in the ZK-02/ZK-04/ZK-06 drill fence. WR-509 was drilled approximately 100 m grid west of the ZK-02/ZK-04/ZK-06 drill fence within the K1a conductive corridor to test for unconformity mineralization. No significant unconformity alteration or mineralization was intersected, however, there was some weak basement mineralization intersected over approximately 0.5 m from 634.2 m within a pelitic lens in a large pegmatite body. No further follow-up was recommended for either hole at that time.

In 2014, Denison completed a drilling campaign of 25 holes for 18,546 m which included the Gryphon discovery hole WR-556. WR-556 was drilled on the ZK-02/ZK-04/ZK-06 fence to test two targets:

- The unconformity down-dip of a sandstone structure intersected in ZK-06; and
- The down-dip projection of basement hosted mineralization intersected in ZK-04 and ZK-06.

No unconformity mineralization was intersected, but high-grade mineralization was intersected at the contact of a graphitic pelitic gneiss and a quartzite unit down dip from hole ZK-06. The mineralization graded 15.3% U_3O_8 over 4.0 m from 697.5 m (approximately 207 m below the unconformity). This mineralization formerly termed the Upper Lens, is now part of the Gryphon A series of lenses.

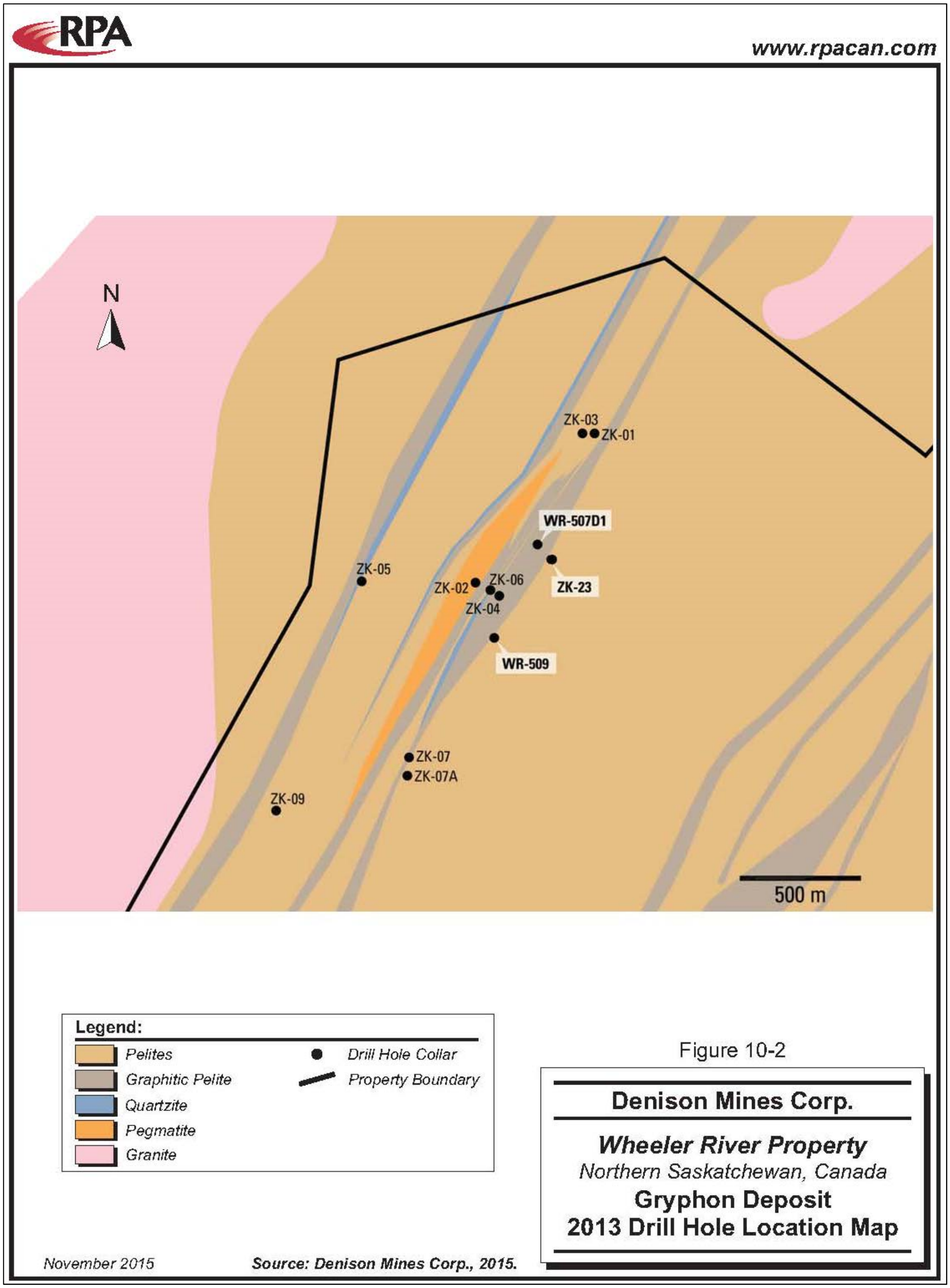


Figure 10-2. Gryphon Deposit 2013 Drill Hole Location Map

In 2014, Denison also drilled holes WR-558 and WR-560. WR-558 was drilled to target the contact of the unconformity with the western most graphitic unit northwest of ZK-02. While no unconformity mineralization was encountered, basement mineralization was intersected in a pegmatite unit approximately 54 m below the unconformity, now considered to be part of the Gryphon D series of lenses. The mineralization graded 7.3% U_3O_8 over 0.5 m from 611.7 m and was considered peripheral mineralization at that time. WR-560 was drilled 35 m up dip of the WR-556 intersection. WR-560 intersected high-grade mineralization at a lower stratigraphic position to that found in WR-556 and what was formerly termed the Lower Lens, is now considered part of the Gryphon C series of lenses. The WR-560 mineralization graded 21.2% U_3O_8 over 4.5 m from 759 m (approximately 234 m below the unconformity).

Since the discovery of Gryphon, definition drilling has continued on all lenses (A through E series). The A through C series lenses have been defined as a body of multiple stacked high-grade lenses that plunge toward the northeast, approximately 80 m to 370 m below the sub-Athabasca unconformity. Denison followed up the 2014 drilling with 2015, 2016 and 2017 winter and summer drilling campaigns. As of January 30, 2018, the effective date of the current mineral resource estimate, Denison and predecessor companies have drilled a total of 251 holes totalling 141,740 m in the Gryphon area of which 214 totalling 120,351 m have delineated the Gryphon deposit. Table 10-3 lists the holes by drilling program and Figure 10-3 shows the location of drilling at Gryphon.

Table 10-3. Gryphon Drilling Statistics

Deposit	Year	Company	# Holes	Total Drilled (m)
Gryphon	1985	SMDC	1	560
	1988	SMDC	3	1,837
	1989	Cameco	2	960
	2001	Cameco	1	584
	2013	Denison	3	1,515
	2014	Denison	25	18,546
	2015	Denison	53	30,990
	2016	Denison	72	43,476
	2017	Denison	91	43,273
GRYPHON TOTAL			**251**	**141,740**

Diamond drilling at Gryphon was primarily done with NQ sized core (47.6 mm diameter) with most holes angled between 60° and 79° to the northwest, 11 of the holes are drilled vertically.

Highlights from the Gryphon drilling program are listed Table 10-4.

Table 10-4. Gryphon Deposit Mineral Intersections

Hole No.	From (m)	To (m)	Thick (m)	% U_3O_8	GT
WR-560	759.0	763.5	4.5	21.21	95.46
WR-556	697.5	701.5	4.0	15.33	61.33
WR-573D1	548.5	551.0	2.5	22.16	55.39
WR-569A	680.0	683.5	3.5	13.16	46.07
WR-604	779.0	784.5	5.5	6.34	34.86
WR-584B	641.6	646.1	4.5	7.50	33.75
WR-569A	702.5	705.5	3.0	10.27	30.82
WR-574	696.5	698.5	2.0	14.60	29.19
WR-571	757.5	760.0	2.5	8.79	21.98
WR-571D2	512.0	517.5	5.5	3.95	21.72
WR-641	718.5	729.5	11.0	5.30	58.30

Notes: Intersection interval is composited at cut-off grade of 1.0% U_3O_8 and minimum thickness of 1 m

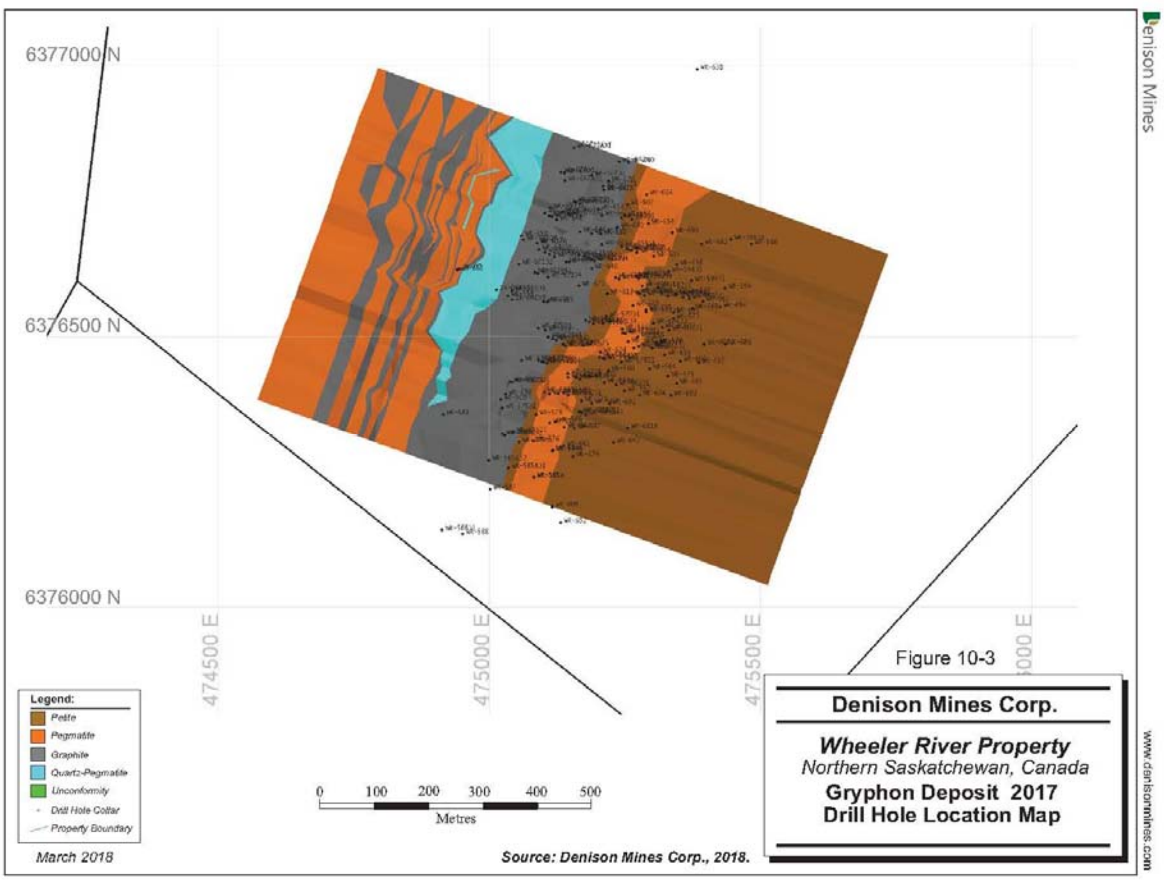


Figure 10-3. Gryphon Deposit 2017 Drill Hole Location Map

10.3 Drill Hole Surveying

The collar locations of drill holes are spotted on a grid established in the field, and collar sites are surveyed by differential base station GPS using the NAD83 UTM zone 13N reference datum. The drill holes have a concise naming convention with the prefix WR denoting Wheeler River followed by the number of the drill hole. Where directional drilling methods were employed, involving the drilling of a 'parent' hole and multiple 'daughter' holes drilled part way down the parent hole, the suffix D was used to denote the 'daughter' nature of the hole location. Subsequent collar locations of daughter holes are derived using a combination of GPS'ed parent hole locations and down hole Reflex survey tests utilizing GEMS software. In general, most of the drilling was completed on northwest-southeast oriented profiles spaced approximately 25 m apart. At Gryphon, 121 of the 214 drill holes were completed as subsurface 'daughter' holes drilled as off-cuts from surface 'parent' holes.

The trajectory of all drill holes is determined with a Reflex instrument in single point mode, which measures the dip and azimuth at 50 m intervals down the hole with an initial test taken six meters below the casing and a final measurement at the bottom of the hole. All mineralized and non-mineralized holes within the Phoenix deposit are cemented from approximately 25 m below the mineralized zone to approximately 25 m above the zone. All mineralized and non-mineralized holes within the Gryphon deposit are cemented for the entire basement column to approximately 25 m above the unconformity.

10.4 Radiometric Logging of Drill Holes

All drill holes on the property are logged with a radiometric probe to measure the natural gamma radiation, from which an indirect estimate of uranium content can be made. Most of the U_3O_8 grade data (76%) used for the Phoenix mineral resource estimate are obtained from chemical assays of the rock. The remainder of the data are derived from radiometric probe results, typically when poor drill core recovery prevents representative sampling for chemical assays. Core recovery at Gryphon is typically 100% and therefore radiometric equivalent U_3O_8 grades ("e U_3O_8") are seldom required as a substitute for chemical U_3O_8 assays. For the updated Gryphon mineral resource estimate, reported herein, 7.0% of the assay intervals relied on eU_3O_8 grades where core recovery was less than 80%.

10.4.1 Radiometric Probing

Probing with a Mount Sopris gamma logging unit employing a triple gamma probe (2GHF-1000) was completed systematically on every drill hole. The probe measures natural gamma radiation using three different detectors: one 0.5 in by 1.5 in sodium iodide (NaI) crystal assembly and two Geiger Mueller (G-M) tubes installed above the NaI detector. These G-M tubes have been used successfully to determine grade in very high concentrations of U_3O_8. By utilizing three different detector sensitivities (the sensitivity of the detectors is very different from one detector to another), these probes can be used in both exploration and development projects across a wide spectrum of uranium grades. Accurate concentrations can be measured in uranium grades ranging from less than 0.1% to as high as 80% U_3O_8. Data are logged from all three detectors at a speed of 10 m/min down hole and 15 m/min up hole through the drill rods. Speeds are generally slowed down while logging through the mineralized intervals at approximately 5 m/min.

The radiometric or gamma probe measures gamma radiation which is emitted during the natural radioactive decay of uranium (U) and variations in the natural radioactivity originating from changes in concentrations of the trace element thorium (Th) as well as changes in concentration of the major rock forming element potassium (K).

Potassium decays into two stable isotopes (argon and calcium) which are no longer radioactive and emits gamma rays with energies of 1.46 MeV. Uranium and thorium; however, decay into daughter products which are unstable (i.e. radioactive). The decay of uranium forms a series of about a dozen radioactive elements in nature which finally decay to a stable isotope of lead. The decay of thorium forms a similar series of radioelements. As each radioelement in the series decays, it is accompanied by emissions of alpha or beta particles or gamma rays. The gamma rays have specific energies associated with the decaying radionuclide. The most prominent of the gamma rays in the uranium series originate from decay of bismuth 214 (^{214}Bi), and in the thorium series from decay of thallium 208 (^{208}Tl).

The natural gamma measurement is made when a detector emits a pulse of light when struck by a gamma ray. This pulse of light is amplified by a photomultiplier tube, which outputs a current pulse which is accumulated and reported as counts per second ("cps"). The gamma probe is lowered to the bottom of a drill hole and data are recorded as the tool travels to the bottom and then is pulled back up to the surface. The current pulse is carried up a conductive cable and processed by a logging system computer which stores the raw gamma cps data.

Since the concentrations of these naturally occurring radioelements vary between different rock types, natural gamma ray logging provides an important tool for lithologic mapping and stratigraphic correlation. For example, in sedimentary rocks, sandstones can be easily distinguished from shales due to the low potassium content of the sandstones compared to the shales. The greatest value of the gamma ray log in uranium exploration, however, is in determining equivalent uranium grade.

The basis of the indirect uranium grade calculation (referred to as eU_3O_8 for equivalent U_3O_8) is the sensitivity of the detector used in the probe which is the ratio of cps to known uranium grade and is referred to as the probe calibration factor. Each detector's sensitivity is measured when it is first manufactured and is also periodically checked throughout the operating life of each probe against a known set of standard test pits, with various known grades of uranium mineralization or through empirical calculations. Application of the calibration factor, along with other probe correction factors, allows for immediate grade estimation in the field as each drill hole is logged.

Down-hole total gamma data are subjected to a complex set of mathematical equations, taking into account the specific parameters of the probe used, speed of logging, size of bore hole, drilling fluids, and presence or absence of any type of drill hole casing. The result is an indirect measurement of uranium content within the sphere of measurement of the gamma detector. A Denison in-house computer program known as GAMLOG converts the measured counts per second of the gamma rays into 10 cm increments of equivalent percent U_3O_8 (%e U_3O_8). GAMLOG is based on the Scott's Algorithm developed by James Scott of the Atomic Energy Commission (AEC) in 1962 and is widely used in the industry.

The conversion coefficients for conversion of probe counts per second to %e U_3O_8 equivalent uranium grades are based on the calibration results obtained at the SRC uranium calibration pits (sodium iodide crystal) and empirical values developed in-house (Sweet and Petrie, 2010) for the triple-gamma probe (Figure 10-4).

SRC down-hole probe calibration facilities are located in Saskatoon, Saskatchewan. The calibration facilities test pits consist of four variably mineralized holes, each approximately four meters thick. The gamma probes are calibrated a minimum of two times per year, usually before and after both the winter and summer field seasons.

Drilling procedures, including collar surveying, down-hole Reflex surveying, and radiometric probing are standard industry practice.

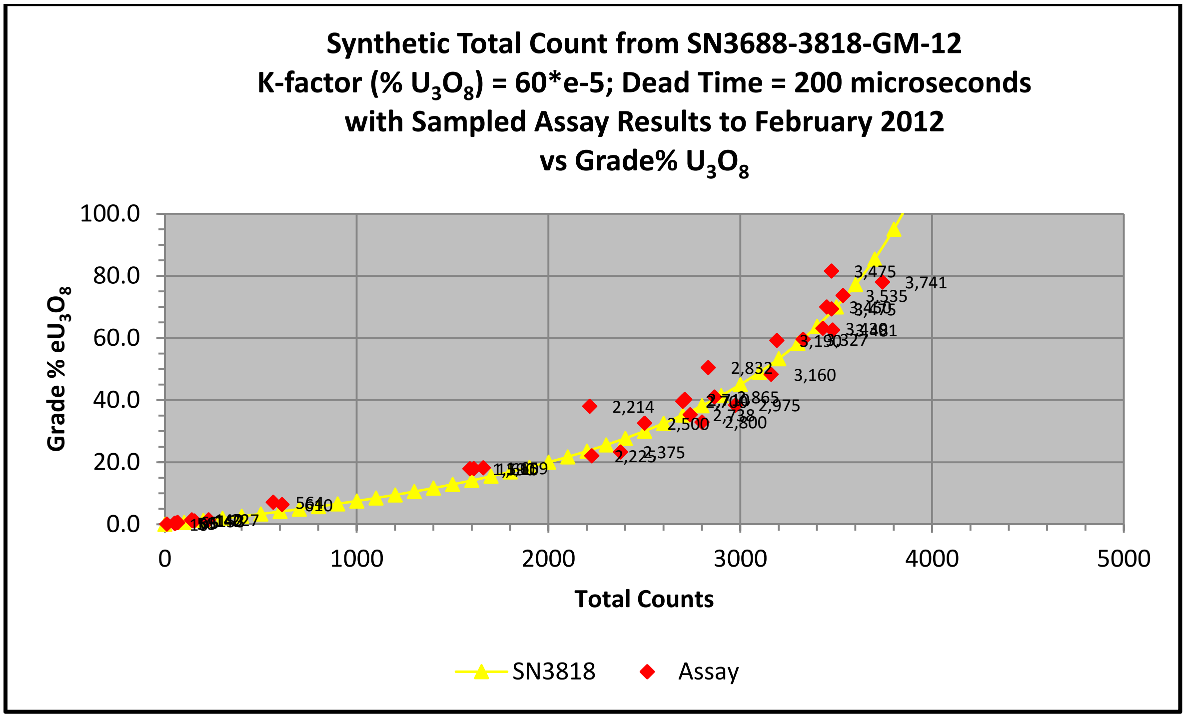


Figure 10-4. Calibration Curve for Geiger-Meuller SN 3818 Probe

10.5 Sampling Method and Approach

10.5.1 Drill Core Handling and Logging Procedures

At each drill site, core is removed from the core tube by the drill contractors and placed directly into three row NQ wooden core boxes with standard 1.5 m length (4.5 m total) or two row HQ wooden boxes with standard 1.5 m (3.0 m total). Individual drill runs are identified with small wooden blocks, onto which the depth in meters is recorded. Diamond drill core is transported at the end of each drill shift to an enclosed core handling facility at Denison's Wheeler River camp. The core handling procedures at the drill site are industry standard. Drill holes are logged at the Wheeler River camp core logging facilities by Denison personnel.

Before the core is split for assay, the core is photographed, descriptively logged, measured for structures, surveyed with a scintillometer, and marked for sampling. Sampling of the holes for assay is guided by the observed geology, radiometric logs, and readings from a hand-held scintillometer.

The general concept behind the scintillometer is similar to the gamma probe except the radiometric pulses are displayed on a scale on the instrument and the respective count rates are recorded manually by the technician logging the core or chips. The hand-held scintillometer provides quantitative data only and cannot be used to calculate uranium grades; however, it does allow the geologist to identify uranium mineralization in the core and to select intervals for geochemical sampling, as described below.

Scintillometer readings are taken throughout the hole as part of the logging process, usually over 3 m intervals, and are averaged for the interval. In mineralized zones, where scintillometer readings are above five times background (approximately 500 cps depending on the scintillometer being used), readings are recorded over 10 cm intervals and tied to the run interval blocks. The scintillometer profile is then plotted on strip logs to compare and adjust the depth of the down-hole gamma logs. Core trays are marked with aluminum tags as well as felt marker.

10.5.2 Drill Core Sampling

Assay Sampling

Denison submits assay samples for geochemical analysis for all the cored sections through mineralized intervals, where core recovery permits. All mineralized core is measured with the scintillometer described above by removing each piece of drill core from the ambient background, noting the most pertinent reproducible result in counts per second, and carefully returning it to its correct place in the core box. Any core registering over 500 cps is flagged for splitting and sent to the laboratory for assay. Early drill holes were sampled using variable intervals (0.2 m to 1.0 m); after drill hole WR-253, holes were sampled using 0.5 m lengths. Barren samples are taken to flank both ends of mineralized intersections, with flank sample lengths at least 0.5 m on either end, which; however, may be significantly more in areas with strong mineralization.

Assay sampling is undertaken by two or more Denison staff geologists with the oversight of the Denison Project Manager or Project Geologist in an onsite dedicated sampling facility for mineralized core. All core samples are split with a hand splitter according to the sample intervals marked on the core. One-half of the core is returned to the core box for future reference and the other half is bagged, tagged, and sealed in a plastic bag. Bags of mineralized samples are sealed for shipping in metal or plastic pails depending on the radioactivity level. Samples collected on 0.5 m spacing through the mineralized zone are analyzed using inductively coupled plasma optical emission spectroscopy (“ICP-OES”). The sealed metal pails containing the mineralized samples are then transported directly from site to Saskatchewan Research Council (“SRC”) GeoAnalytical Laboratories by vehicle by a Denison employee. The pails are delivered directly to a receiving SRC employee ensuring the security and integrity of the samples. Denison employees follow Transportation of Dangerous Goods protocols as outlined by the provincial government.

Other Sampling

Three other types of drill core samples are collected as follows:

1. Composite geochemical samples are collected over approximately 10 m intervals in the upper Athabasca sandstone and in fresh lithologies beneath the unconformity (basement) and over 5 m intervals in the basal sandstone and altered basement units. The samples consist of 1 cm to 2 cm disks of core collected at the top or bottom of each row of core in the box over the specified interval. Care is taken not to cross lithological contacts or stratigraphic boundaries.
2. Representative/systematic core disks (one to five centimeters in width) are collected at regular 5 m to 10 m intervals throughout the entire length of core until basement lithologies become unaltered. These samples are analyzed for clay minerals using reflectance spectroscopy.
3. Select spot samples are collected from significant geological features (i.e. radiometric anomalies, structure, alteration, etc.). Core disks 1 cm to 2 cm thick are collected for reflectance spectroscopy and split core samples, over the desired interval, are sent for geochemical analysis. Ten-centimeter wide core samples may also be collected for density measurement.

These sampling types and approaches are typical of uranium exploration and definition drilling programs in the Athabasca Basin. The drill core handling and sampling protocols are industry standard.

10.6 Core Recovery and Use of Probe Data

At Phoenix, the mineralized zones (sandstones or basement) are moderately to strongly altered, and occasionally disrupted by fault breccias. In places, the core can be broken and blocky, however, recovery is generally good with an overall average of 89.65%. Local intervals of up to 5 m with less than 80% recovery have been encountered due to washouts during the drilling process. Where 80% or less of a composited interval is recovered during drilling (>20% core loss), or where no geochemical sampling has occurred across a mineralized interval, uranium grade determination has been supplemented by radiometric probe data. Radiometric probe data accounts for approximately 23% of the drill holes used for the mineral resource estimate at Phoenix. There are 1,708 U_3O_8 assay records totalling 848 m in the Phoenix deposit database. Of these, 1,464 U_3O_8 assay records totalling 726 m are in Zone A and 244 U_3O_8 assay records totalling 122 m are in Zone B.

Core recovery at Gryphon is typically 100% and therefore radiometric eU_3O_8 grades are seldom required as a substitute for chemical U_3O_8 assays. There are 5,591 U_3O_8 assay records totalling 2,796 m in the Gryphon deposit database of which 3,393 totalling 1,596 m were used in the resource estimate. For the updated Gryphon mineral resource estimate, reported herein, 7.0% of the assay intervals relied on eU_3O_8 grades where core recovery was less than 80%.

RPA is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

11 Sample Preparation, Analyses, and Security

As described in Section 10, core from the property is photographed, logged, marked for sampling, split, bagged, and sealed for shipment by Denison personnel at the Wheeler River field logging facility. All samples for assay or geochemical analyses are sent to the SRC Geoanalytical Laboratories (SRC) in Saskatoon, Saskatchewan. Samples for reflectance clay analyses have been analyzed using a PIMA spectrometer or an ArcSpectro FT-NIR ROCKET spectrometer and sent to Rekasa Rocks Inc. ("Rekasa") based in Saskatoon, Saskatchewan or AusSpec International Ltd. ("AusSpec") based in Arrowtown, New Zealand, respectively, for interpretation. SRC, Rekasa, and AusSpec are independent facilities that offer analytical services to the mineral exploration and mining industry. All samples for geochemical or clay analyses are shipped to Saskatoon by airfreight or ground transport. All samples for U_3O_8 assays are transported by land to the SRC laboratory by Denison personnel. A sample transmittal form is prepared that identifies each batch of samples. SRC performs sample preparation on all samples submitted. There is no sample preparation, apart from drying, involved for the samples sent for clay analyses.

11.1 Geochemical Sample Preparation Procedures

11.1.1 Sample Receiving

Samples are received at the SRC laboratory as either dangerous goods (qualified Transport of Dangerous Goods [TDG] personnel required) or as exclusive use only samples (no radioactivity documentation attached). On arrival, samples are assigned an SRC group number and are entered into the Laboratory Information Management System (LIMS).

All received sample information is verified by sample receiving personnel: sample numbers, number of pails, sample type/matrix, condition of samples, request for analysis, etc. The samples are then sorted by radioactivity level. A sample receipt and sample list are then generated and e-mailed to the appropriate authorized personnel at Denison. Denison is notified if there are any discrepancies between the paperwork and samples received.

11.1.2 Sample Sorting

To ensure that there is no cross contamination between sandstone and basement, non-mineralized, low-level, and high-level mineralized samples, they are sorted by their matrix and radioactivity level. Samples are firstly sorted in their group into matrix type (sandstone and basement/mineralized).

The samples are then checked for their radioactivity levels. Using a Radioactivity Detector System, the samples are classified into one of the following levels:

- "Red Line" (minimal radioactivity) <500 cps.
- "1 Dot" 500 – 1,999 cps.
- "2 Dots" 2,000 – 2,999 cps.
- "3 Dots" 3,000 – 3,999 cps.
- "4 Dots" 4,000 – 4,999 cps.
- "UR" (unreadable) >5,000 cps.

The samples are then sorted into ascending sample numerical order and transferred to their matrix designated drying oven.

11.1.3 Sample Preparation

After the drying process is complete, “Red line” and “1 Dot” samples are sent for further processing (crushing and grinding) in the main SRC laboratory. All radioactive samples at “2 Dots” or higher are sent to a secure radioactive facility at SRC for the same sample preparation. Plastic snap top vials are labelled according to sample numbers and sent with the samples to the appropriate crushing room. All highly radioactive materials are kept in a radioactive bunker until they can be transported by TDG trained individuals to the radioactivity facility for processing.

Rock samples are jaw crushed to 60% passing -2 mm. Samples are placed into the crusher (one at a time) and the crushed material is put through a splitter. The operator ensures that the distribution of the material is even, so there is no bias in the sampling. One portion of the material is placed into the plastic snap top vial and the other is put in the sample bag (reject). The first sample from each group is checked for crushing efficiency by screening the vial of rock through a 2 mm screen. A calculation is then carried out to ensure that 60% of the material is -2 mm. If the quality control (QC) check fails, the crushing is redone and checked for crushing efficiency; if it still fails, the QC department is notified and corrective action is taken.

The crusher, crusher catch pan, splitter, and splitter catch pan are cleaned between each sample using compressed air.

The reject material is returned to its original sample bag and archived in a plastic pail with the appropriate group number marked on the outside of the pail. The vials of material are then sent to grinding; each vial of material is placed in pots (six pots per grind) and ground for two minutes. The material is then returned to the vials. The operator shakes the vial to check the fineness of the material by looking for visible grains and listening for rattling. The sample is then screened through a 106-micron sieve, using water. The sample is then dried and weighed; to pass the grinding efficiency QC, there must be over 90% of the material at -106 micron. The material is then transferred to a labelled plastic snap top vial.

The pots are cleaned out with silica sand and blown out with compressed air at the start of each group. In the radioactive facility, the pots are cleaned with water. Once sample pulps are generated, they are returned to the main laboratory to be chemically processed prior to analysis. All containers are identified with sample information and their radioactivity status at all times. When the preparation is completed, the radioactive pulps are returned to a secure radioactive bunker, until they can be transported back to the radioactive facility. All rejected sample material not involved in the grinding process is returned to the original sample container. All highly radioactive materials are stored in secure radioactive designated areas.

Sample preparation methods for the samples used in the Gryphon and Phoenix mineral resource estimates meet or exceed industry standards.

11.2 Analytical Methods

All assay core samples from Gryphon and Phoenix were analyzed by the ICP1 package offered by SRC. Composite geochemical samples, up to and including WR-269, were also analyzed using this method after which the method was changed to ICP-MS1 because of a lower detection limit.

11.2.1 Method: ICP1

(Uranium multi-element exploration analysis by ICP-OES).

Method Summary: In ICP-OES analysis, the atomized sample material is ionized, and the ions then emit light (photons) of a characteristic wavelength for each element, which is recorded by optical spectrometers. Calibrations against standard materials allow this technique to provide a quantitative geochemical analysis.

The analytical package includes 62 analytes (46 total digestion, 16 partial digestion), with nine analytes being analyzed for both partial and total digestions (Ag, Co, Cu, Mo, Ni, Pb, U, V, and Zn) plus boron. These samples are also sometimes analyzed for Au by fire assay.

Partial Digestion: For partial digestion analysis, samples were crushed to 60% -2 mm and a 100 g to 200 g sub-sample was split out using a riffler. The sub-sample pulverized to 90% -106 µm using a standard puck and ring grinding mill. The sample was then transferred to a plastic snap top vial. An aliquot of pulp is digested in a digestion tube in a mixture of HNO_3:HCl, in a hot water bath for approximately one hour, then diluted to 15 mL using de-ionized water. The samples were then analyzed using a Perkin Elmer ICP-OES instrument (models DV4300 or DV5300).

Total Digestion: An aliquot of pulp is digested to dryness in a hot block digestor system using a mixture of concentrated HF:HNO_3:$HClO_4$. The residue is dissolved in 15 mL of dilute HNO_3 and analyzed using the same instrument(s) as above.

11.2.2 Method: ICPMSI

(The multi-element determination by ICP-MS).

Method Summary: The analytical package includes the analysis of 47 elements and oxides using a three acid (HF/HNO_3/$HClO_4$) "total" digestion and a suite of 42 elements using a two acid (HNO_3/HCl) "partial" digestion. Analysis of the lead isotopes (204Pb, 206Pb, 207Pb, and 208Pb) are also included in the package. Boron is determined by ICP-OES analysis after fusion with NaO_2/$NaCO_3$. PerkinElmer instruments (models Optima 300DV, Optima 4300DV, and Optima 5300DV) are currently in use. The samples generally analyzed by this package are non-radioactive, non-mineralized sandstones and basement rocks with low concentrations of uranium (<100 ppm).

Partial Digestion: An aliquot of pulp is digested in a mixture of ultra-pure concentrated nitric and hydrochloric acids (HNO_3:HCl) in a digestion tube in a hot water bath, then diluted to 15 mL using de-ionized water prior to analysis. As, Ge, Hg, Sb, Se, and Te are subject to partial digestion only, as these elements are not suited to total digestion analysis. The ICP-MS instruments used are PerkinElmer Elan DRC II.

Total Digestion: An aliquot of pulp is digested to dryness in a hot block digestor system using a mixture of ultra-pure concentrated acids HF:HNO_3:$HClO_4$. The residue is dissolved in 15 mL of 5% HNO_3 and made to volume using de-ionized water prior to analysis.

11.2.3 Method: U_3O_8 wt% Assay

(The determination of U_3O_8 wt% in solid samples by ICP-OES).

Method Summary: When ICP1 U partial values are ≥1,000 ppm, sample pulps are re-assayed for U_3O_8 using SRC's ISO/IEC 17025:2005-accredited U_3O_8 (wt%) method. In the case of uranium assay by ICP-OES, a pulp is already generated from the first phase of preparation and assaying (discussed above).

Aqua Regia Digestion: An aliquot of sample pulp is digested in a 100 mL volumetric flask in a mixture of 3:1 HCl:HNO_3 on a hot plate for approximately one hour, then diluted to volume using de-ionized water. Samples are diluted prior to analysis by ICP-OES.

Instrument Analysis: Instruments in the analysis are calibrated using certified commercial solutions. The instruments used were PerkinElmer Optima 300DV, Optima 4300DV, or Optima 5300DV.

Detection Limits: 0.001% U_3O_8

11.2.4 Method: U_3O_8 wt% Assay

(The determination of U_3O_8 wt% in solid samples by delayed neutron counting).

SRC in 2009 documented the method summary for the Delayed Neutron Counting (DNC) technique as follows. Samples previously prepared as pulps for ICP total digestion are used for the DNC analysis. The pulps are irradiated in a Slowpoke 2 nuclear reactor for a given period of time. After irradiation, the samples are pneumatically transferred to a counting system equipped with six helium-3 detectors. After a suitable delay period, neutrons emanating from the sample are counted. The proportion of delayed neutrons emitted is related to the uranium concentration. For low concentrations of uranium, a minimum of one gram of sample is preferred, and larger sample sizes (two to five grams) will improve precision. Several blanks and certified uranium standards are analyzed to establish the instrument calibration. In addition, control samples are analyzed with each batch of samples to monitor the stability of the calibration. At least one in every ten samples is analyzed in duplicate. The results of the instrument calibration, blanks, control samples, and duplicates must be within specified limits otherwise corrective action is required.

Analysis for uranium by DNC incorporates four separate flux/site conditions of varying sensitivity to produce an effective range of analysis from zero to 150,000 µg U per capsule (samples of up to 90% U can be analyzed by weighing a fraction of a gram to ensure that there is no more than 150,000 µg U in the capsule). Each condition is calibrated using between three and seven reference materials. For each condition, one of these materials is designated as a calibration check sample. As well, there is an independent control sample for each condition.

11.2.5 Drill Core Bulk Density Analysis

Drill core samples collected for bulk density measurements were sent to SRC. Samples were first weighed as received and then submerged in de-ionized water and re-weighed. The samples were then dried until a constant weight was obtained. The sample was then coated with an impermeable layer of wax and weighed again while submersed in de-ionized water. Weights were entered into a database and the bulk density of each sample was calculated. Water temperature at the time of weighing was also recorded and used in the bulk density calculation.

11.2.6 Reflectance Clay Analyses

Prior to 2015, core chip samples for clay analysis were analyzed using a PIMA II spectrometer. This included all analyses performed on samples from the Phoenix deposit. Short wave infrared (SWIR) spectra were sent to Rekasa, a private facility in Saskatoon, for interpretation. Samples were air or oven dried prior to analysis in order to remove any excess moisture. Reflective spectra for the various clay minerals present in the sample were compared to the spectral results from Athabasca samples for which the clay mineral proportions have been determined in order to obtain a semi-quantitative clay estimate for each sample.

From 2015, core chip samples for clay reflectance analysis were analyzed using an ArcSpectro FT-NIR (Fourier transform near-infrared) ROCKET spectrometer. This included all analyses performed on samples from the Gryphon deposit. Sample collection and preparation is identical to procedures used for PIMA analysis. The transmission spectra of the reflectance samples were sent to AusSpec, based in New Zealand. The spectra are analyzed using an aiSIRIS automated spectral interpretation system. The mineral assemblage for each sample is listed in order of spectral dominance and represents the spectral contribution of the mineral to the spectrum. The results compared well with previous PIMA spectra interpretations undertaken by Rekasa.

11.3 Quality Assurance and Quality Control

Quality assurance and quality control (QA/QC) programs provide confidence in the geochemical results and help ensure that the database is reliable to estimate mineral resources. Denison has developed and documented several QA/QC procedures and protocols for all exploration projects which include the following components:

- Determination of precision – achieved by regular insertion of duplicates for each stage of the process where a sample is taken or split.
- Determination of accuracy – achieved by regular insertion of standards or materials of known composition.
- Checks for contamination – achieved by insertion of blanks.

RPA reviewed Denison's procedures and protocols and considers them to be reasonable and acceptable.

11.3.1 Sample Standards, Blanks, and Field Duplicates

Uranium Assay Standards

Analytical standards are used to monitor analytical precision and accuracy, and field standards are used as an independent monitor of laboratory performance. Six uranium assay standards have been prepared for use in monitoring the accuracy of uranium assays received from the laboratory. Due to the radioactive nature of the standard material, insertion of the standard materials is preferable at SRC instead of in the field. During sample processing, the appropriate standard grade is determined, and an aliquot of the appropriate standard is inserted into the analytical stream for each batch of materials assayed.

Denison uses standards provided by its former Wheeler River joint venture partner Cameco for uranium assays. Cameco standards are added to the sample groups by SRC personnel, using the standards appropriate for each group. As well, for each assay group, an aliquot of Cameco's blank material is also included in the sample run. In a run of 40 samples, at least one will consist of a Cameco standard and one will consist of a Cameco blank. Accuracy of the analyses and values obtained relative to the standard values, based on the analytical results of the six reference standards used, is acceptable for mineral resource estimates. Chronological plots for the six standards are shown in Figures 11-1 to 11-6 with upper limit (UL) and lower limit (LL) being equal to the mean plus or minus three standard deviations respectively. Note that in Figure 11-1 and Figure 11-6 the standards were changed during 2011.

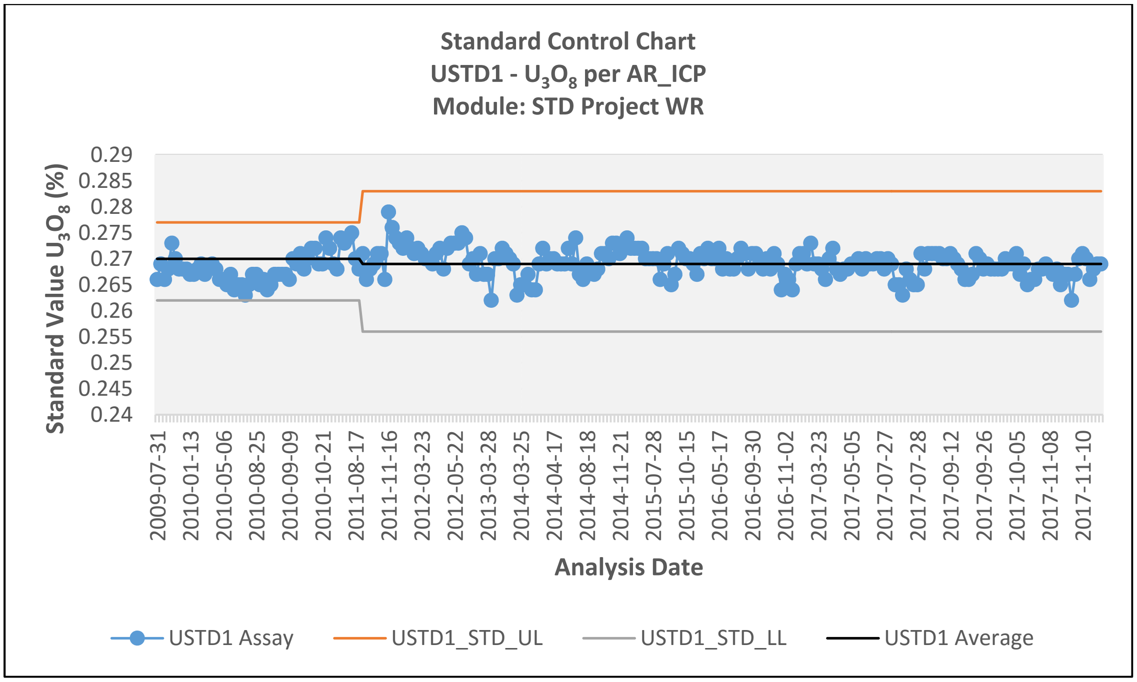


Figure 11-1. USTD1 Analyses

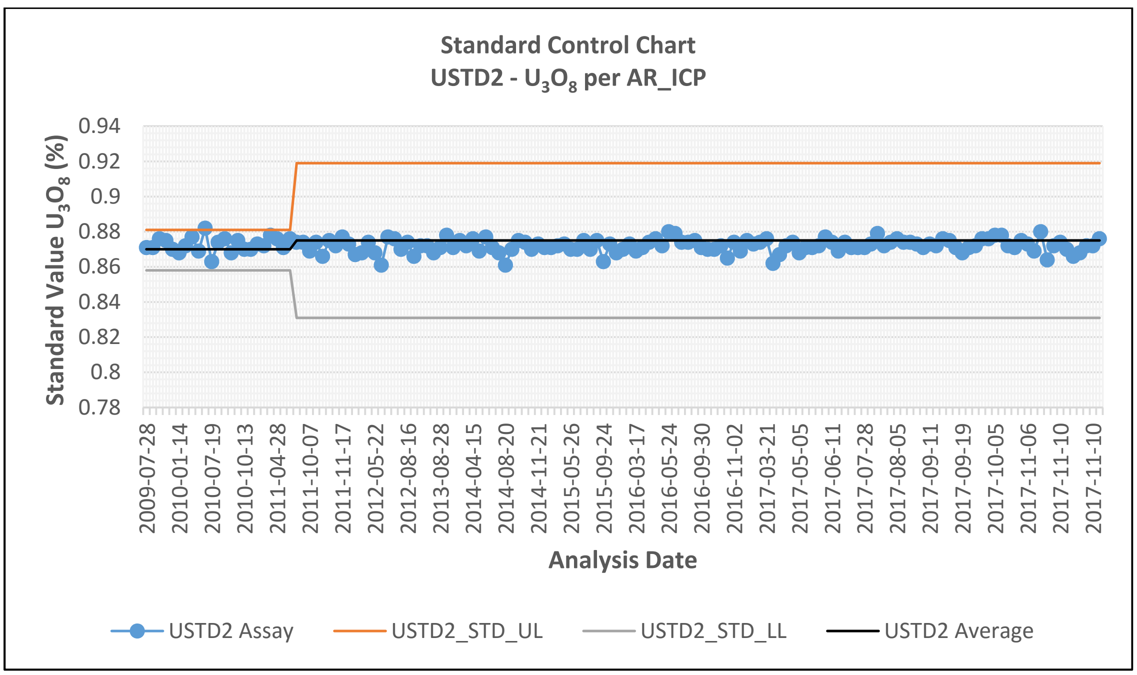


Figure 11-2. USTD2 Analyses

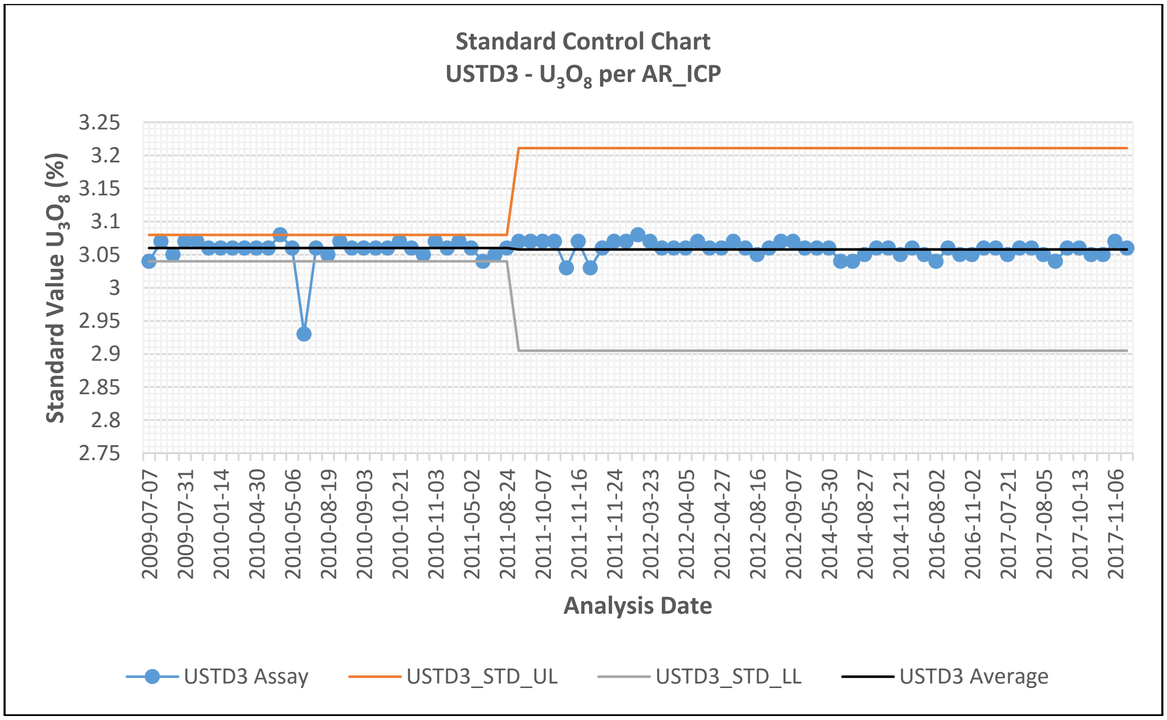


Figure 11-3. USTD3 Analyses

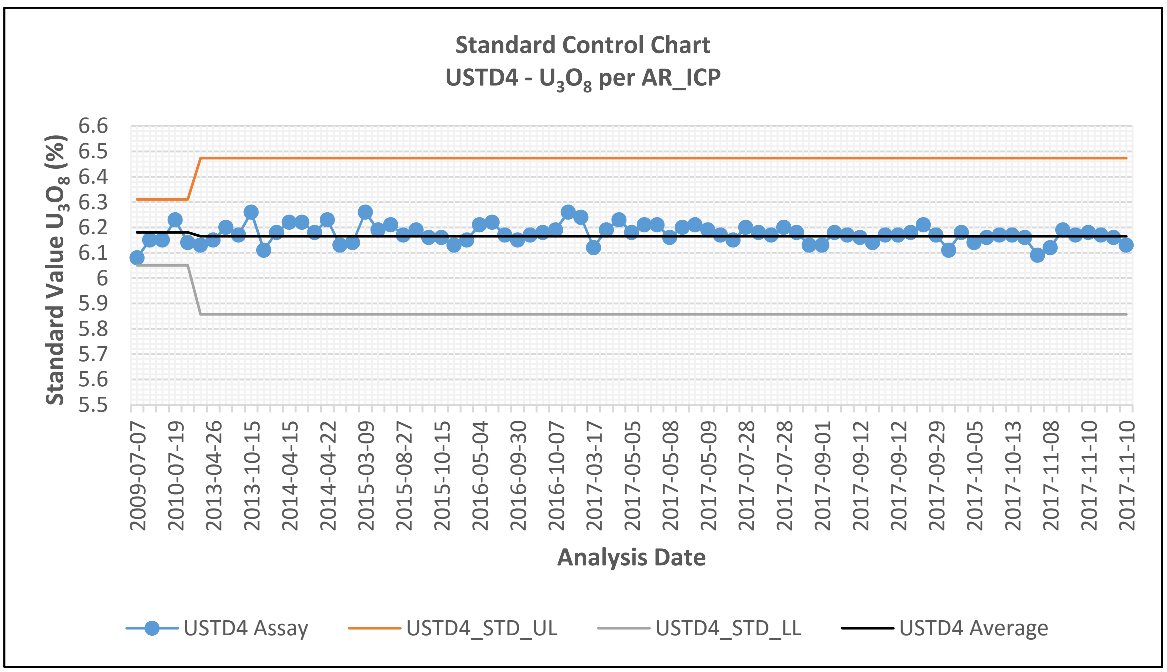


Figure 11-4. USTD4 Analyses

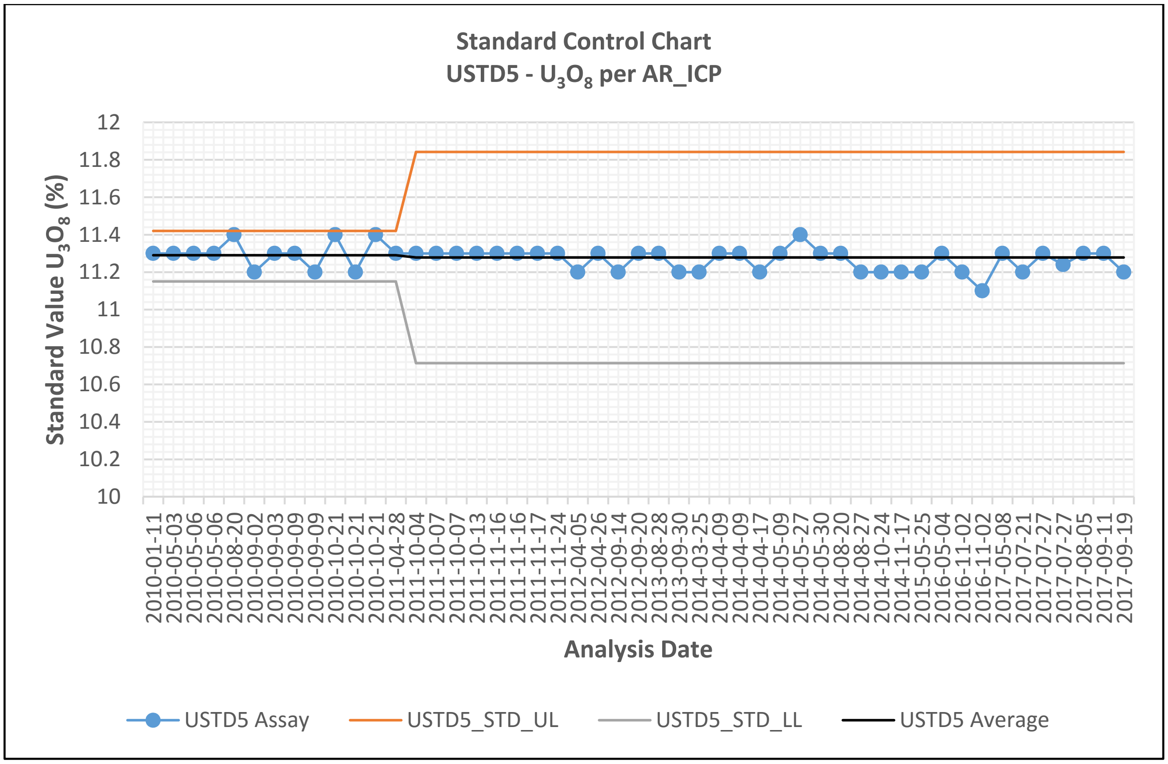


Figure 11-5. USTD5 Analyses

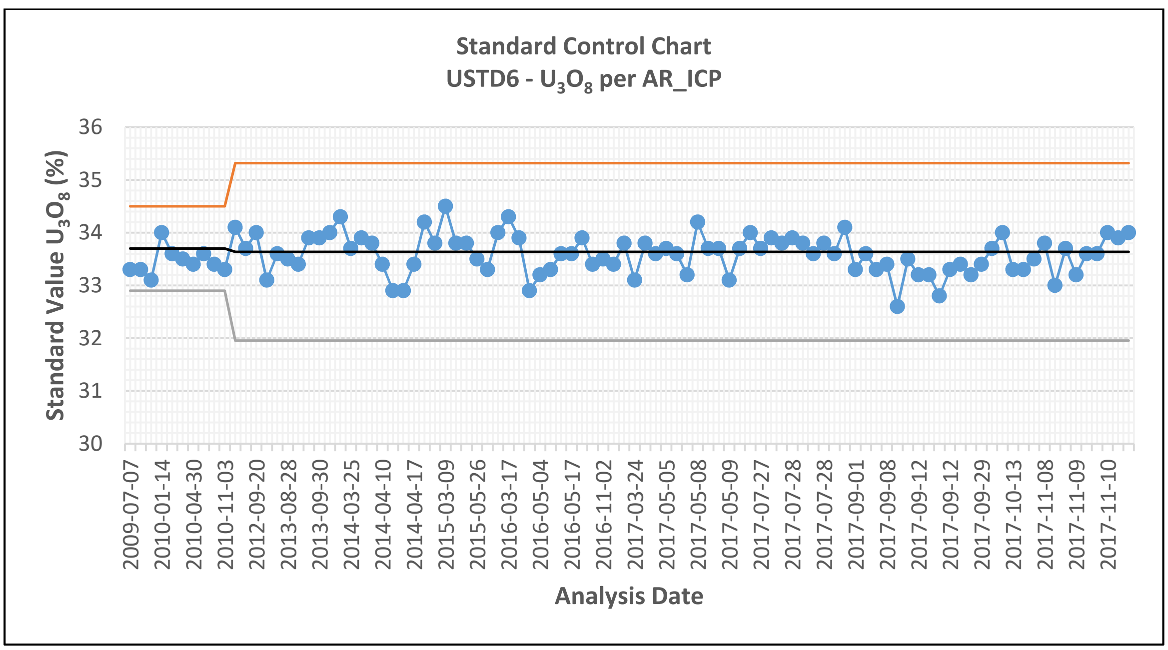


Figure 11-6. USTD6 Analyses

Blanks

Denison employs a lithological blank composed of quartzite to monitor the potential for contamination during sampling, processing, and analysis. The selected blank consists of a material that contains lower contents of U_3O_8 than the sample material but is still above the detection limit of the analytical process. Due to the sorting of the samples submitted for assay by SRC based on radioactivity, the blanks employed must be inserted by the SRC after this sorting takes place, in order to ensure that these materials are ubiquitous throughout the range of analytical grades. In effect, if the individual geologists were to submit these samples anonymously, they would invariably be relegated to the minimum radioactive grade level, preventing their inclusion in the higher radioactive grade analyses performed by SRC. Figure 11-7 shows results of analyses of blank samples. It can be seen that most are below the upper limit of 0.013% U_3O_8, with a maximum analysis of 0.036% U_3O_8.

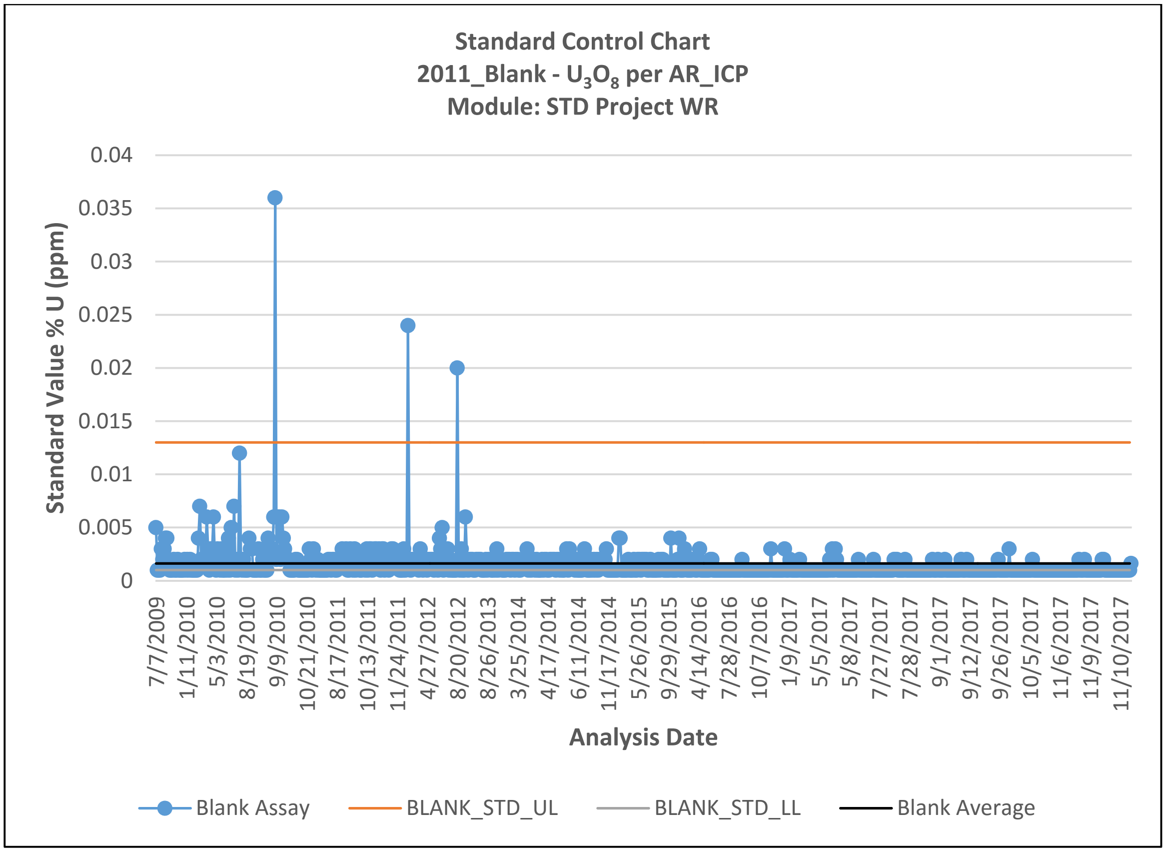


Figure 11-7. Blank Sample Analyses Results

Field Assay Duplicates

Analyses of duplicate samples are a mandatory component of quality control. Duplicates are used to evaluate the field precision of analyses received and are typically controlled by rock heterogeneity and sampling practices. Core duplicates are prepared by collecting a second sample of the same interval, through splitting the original sample, or other similar technique, and are submitted as an independent sample. Duplicates are typically submitted at a minimum rate of one per 20 samples in order to obtain a collection rate of 5%. The collection may be further tailored to reflect field variation in specific rock types or horizons. Figure 11-8 shows results of analyses of field core duplicates plotted against original analyses. It can be seen that results are satisfactory with a correlation coefficient of 92%.

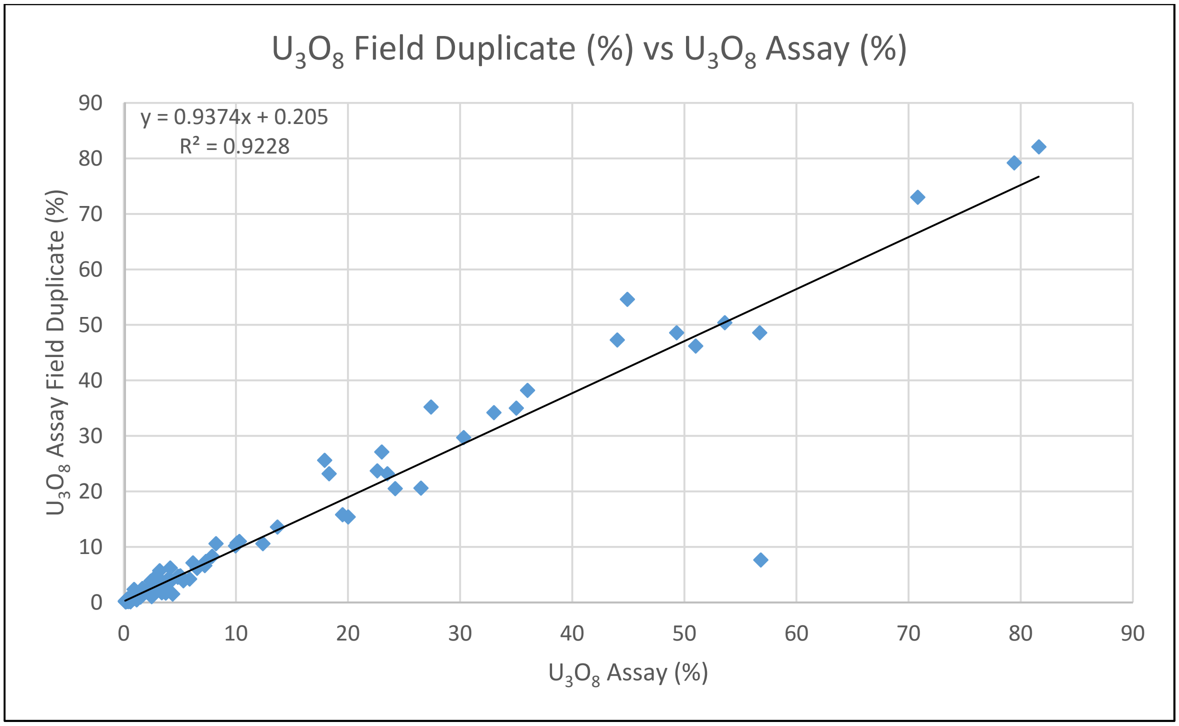


Figure 11-8. Field Duplicate Analyses

11.3.2 SRC Internal QA/QC Program

The SRC laboratory has a quality assurance program dedicated to active evaluation and continual improvement in the internal quality management system. The laboratory is accredited by the Standards Council of Canada as an ISO/IEC 17025 Laboratory for Mineral Analysis Testing and is also accredited ISO/IEC 17025:2005 for the analysis of U_3O_8. The laboratory is licensed by the Canadian Nuclear Safety Commission (CNSC) for possession, transfer, import, export, use, and storage of designated nuclear substances by CNSC Licence Number 01784-1-09.3. As such, the laboratory is closely monitored and inspected by the CNSC for compliance.

All analyses are conducted by SRC, which has specialized in the field of uranium research and analysis for over 30 years.

SRC is an independent laboratory, and no associate, employee, officer, or director of Denison is, or ever has been, involved in any aspect of sample preparation or analysis on samples from the Gryphon or Phoenix deposits.

The SRC uses a laboratory management system (LMS) for quality assurance. The LMS operates in accordance with ISO/IEC 17025:2005 (CAN-P-4E) "General Requirements for the Competence of Mineral Testing and Calibration Laboratories" and is also compliant to CAN-P-1579 "Guidelines for Mineral Analysis Testing Laboratories". The laboratory continues to participate in proficiency testing programs organized by CANMET (CCRMP/PTP-MAL).

All instruments are calibrated using certified materials. Quality control samples were prepared and analyzed with each batch of samples. Within each batch of 40 samples, one to two quality control samples were inserted. Five U_3O_8 reference standards are used: BLA2, BL3, BL4A (Figure 11-9), BL5, and SRCUO2 which have concentrations of 0.502%, 1.21% U_3O_8, 0.148% U_3O_8, 8.36% U_3O_8, and 1.58%

U_3O_8, respectively. One in every 40 samples is analyzed in duplicate; the reproducibility of this is 5%. Before the results leave the laboratory, the standards, blanks, and split replicates are checked for accuracy, and issued provided the senior scientist is fully satisfied. If for any reason there is a failure in an analysis, the sub-group affected will be re-analyzed, and checked again. A corrective action report is issued, and the problem is investigated fully to ensure that any measures to prevent the re-occurrence can and will be taken. All human and analytical errors are, where possible, eliminated. If the laboratory suspects any bias, the samples are re-analyzed, and corrective measures are taken.

Quality control samples (reference materials, blanks, and duplicates) are included with each analytical run, based on the rack sizes associated with the method. The rack size is the number of samples (including QC samples) within a batch. Blanks are inserted at the beginning, standards are inserted at random positions, and duplicates are analyzed at the end of the batch. Quality control samples are inserted based on the analytical rack size specific to the method (Table 11-1).

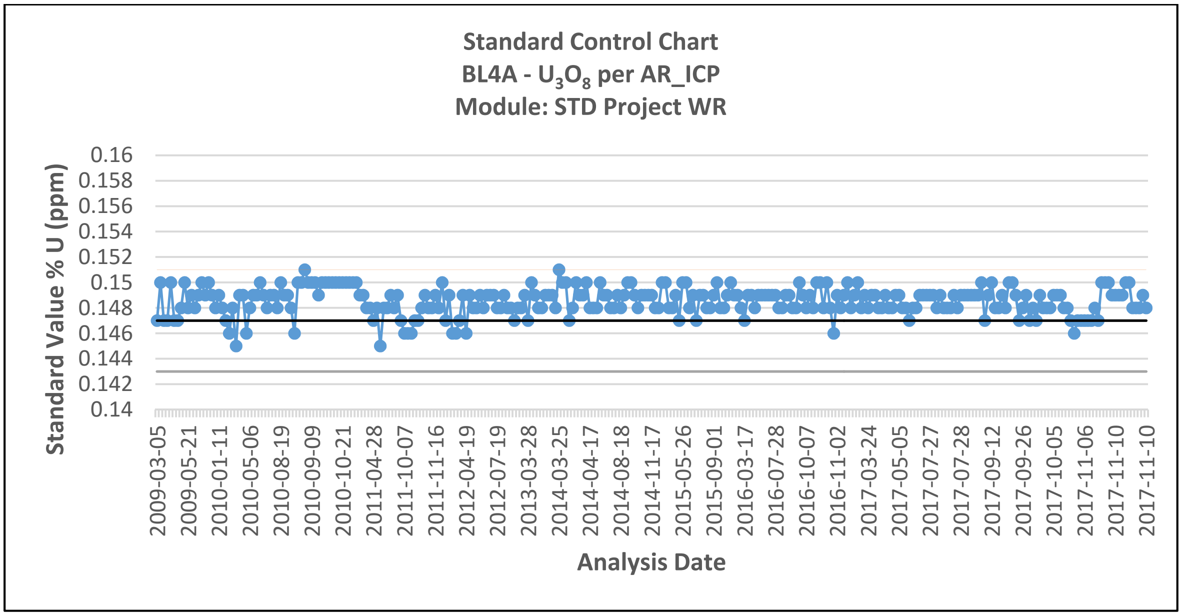


Figure 11-9. BLA4 Analyses

Table 11-1. Quality Control Sample Allocations

Rack Size	Methods	Quality Control Sample Allocation
20	Specialty methods including specific gravity, bulk density, and acid insolubility	2 standards, 1 duplicate, 1 blank
28	Specialty fire assay, assay-grade, umpire and concentrate methods	1 standard, 1 duplicate, 1 blank
40	Regular AAS, ICP-AES and ICP-MS methods	2 standards, 1 duplicate, 1 blank
84	Regular fire assay methods	2 standards, 3 duplicates, 1 blank

11.3.3 External Laboratory Check Analysis

In addition to the QA/QC described above, Denison sends one in every 25 samples to SRC's DNC laboratory, a separate facility located at SRC Analytical Laboratories in Saskatoon, to compare the uranium values using two different methods, by two separate laboratories. The DNC method is specific for uranium and no other elements are analyzed by this technique. The DNC system detects neutrons emitted by the fission of U-235 in the sample, and the instrument response is compared to the response from known reference materials to determine the concentration of uranium in the sample. In order for the analysis to work, the uranium must be in its natural isotopic ratio. Enriched or depleted, uranium cannot be analyzed accurately by DNC.

There are 295 assay pairs that used both ICP-OES total digestion and the DNC assay technique. Figure 11-10 shows the correlation between the SRC Geoanalytical and the SRC DNC laboratories. It can be seen that correlation is excellent. Uranium grades obtained with the DNC technique were used only as check assays and were not directly used for mineral resource estimation.

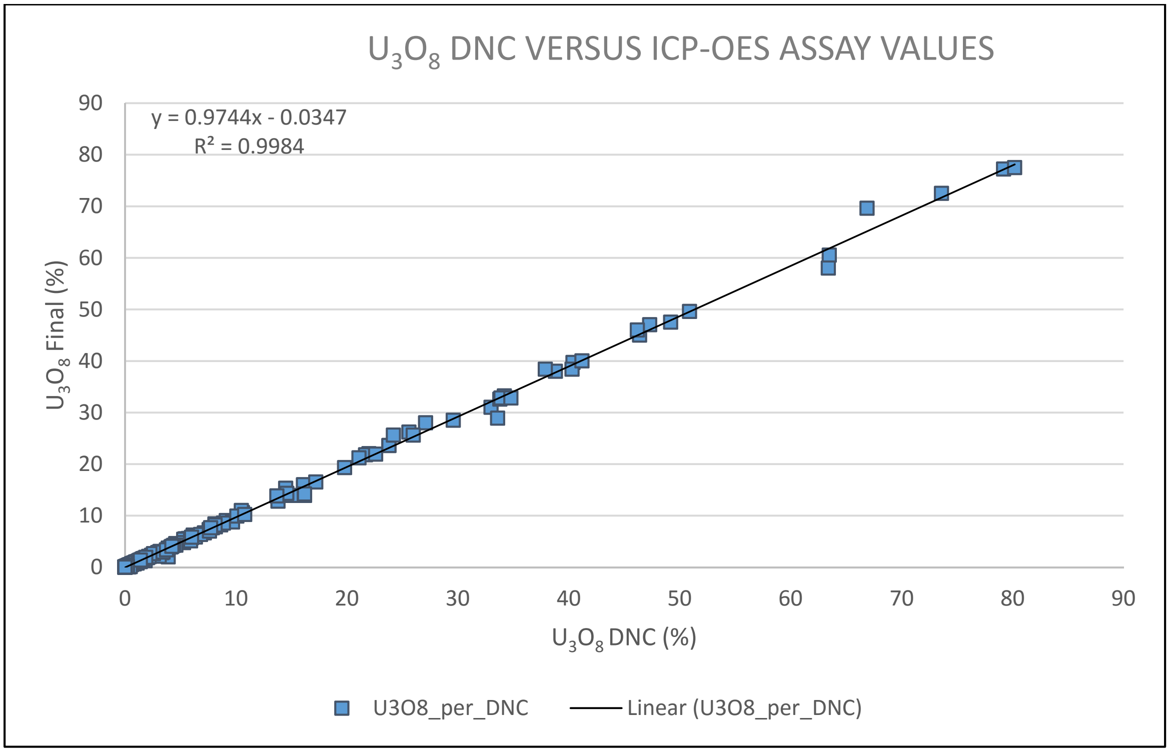


Figure 11-10. U_3O_8 DNC vs ICP-OES Assay Values

11.3.4 Security and Confidentiality

SRC considers customer confidentially and security to be of utmost importance and takes appropriate steps to protect the integrity of sample processing at all stages from sample storage and handling to transmission of results. All electronic information is password protected and backed up on a daily basis. Electronic results are transmitted with additional security features. Access to SRC's premises is restricted by an electronic security system. The facilities at the main laboratory are regularly patrolled by security guards 24 hours a day.

After the analyses are completed, analytical data are securely sent using electronic transmission of the results by SRC to Denison. The electronic results are secured using WINZIP encryption and password protection. These results are provided as a series of Adobe PDF files containing the official analytical results and a Microsoft Excel file containing only the analytical results.

In RPA's opinion, sample preparation, security, and analytical procedures meet industry standards, and the QA/QC program as designed and implemented by Denison is adequate. Consequently, the assay results within the drill hole database are suitable for use in a mineral resource estimate.

12 Data Verification

Based on the data validation by Denison and RPA and the results of the standard, blank, and duplicate analyses, RPA is of the opinion that the assay database is of sufficient quality for mineral resource estimation.

RPA reviewed and verified the resource database used to estimate the mineral resources for both the Phoenix and Gryphon deposits. The verification included a review of the QA/QC methods and results, verifying assay certificates against the database assay table, standard database validation tests, and site visits to both the Gryphon and Phoenix deposits.

Denison has developed and documented several QA/QC procedures and protocols for all exploration projects operated by Denison. The review of the QA/QC program and results is presented in Section 11. RPA reviewed Denison's procedures and protocols and considers them to be reasonable and acceptable.

12.1 Site Visit and Core Review

Dr. Roscoe visited the property on June 16, 2014 in connection with the Phoenix deposit mineral resource estimate and held discussions with technical personnel in RPA's Toronto office on May 4, 2014. Mr. Mathisen visited the property on March 23 to 25, 2015, during the winter drill program in connection with the initial Gryphon mineral resource estimate and again from September 21 to 22, 2017 during the summer drill program in relation to the most recent updated Gryphon resource estimate discussed herein. RPA visited several drill sites and reviewed all core handling, logging, sampling, and storage procedures. RPA examined core from several drill holes and compared observations with assay results and descriptive log records made by Denison geologists. As part of the review, RPA verified the occurrences of mineralization visually and by way of a hand-held scintillometer.

12.2 Database Validation

RPA conducted audits of historic records to ensure that the grade, thickness, elevation, and location of uranium mineralization used in preparing the current uranium resource estimate correspond to mineralization. RPA performed the following digital queries. No significant issues were identified.

- Header table: searched for incorrect or duplicate collar coordinates and duplicate hole IDs.
- Survey table: searched for duplicate entries, survey points past the specified maximum depth in the collar table, and abnormal dips and azimuths.
- Core recovery table: searched for core recoveries greater than 100% or less than 80%, overlapping intervals, missing collar data, negative widths, and data points past the specified maximum depth in the collar table.
- Lithology and probe tables: searched for duplicate entries, intervals past the specified maximum depth in the collar table, overlapping intervals, negative widths, missing collar data, missing intervals, and incorrect logging codes.
- Geochemical and assay table: searched for duplicate entries, sample intervals past the specified maximum depth, negative widths, overlapping intervals, sampling widths exceeding tolerance levels, missing collar data, missing intervals, and duplicated sample IDs.

12.3 Independent Verification of Assay Table

The assay table contains 5,591 laboratory records. RPA verified approximately 2,215 records representing 40% of the data for uranium values against 39 different laboratory certificates. No discrepancies were found.

12.4 Disequilibrium

Radioactive isotopes lose energy by emitting radiation and transition to different isotopes in a "decay series" or "decay chain" until they eventually reach a stable non-radioactive state. Decay chain isotopes are referred to as "daughters" of the "parent" isotope. When all the decay products are maintained in close association with uranium-238 for the order of a million years, the daughter isotopes will be in equilibrium with the parent. Disequilibrium occurs when one or more decay products is dispersed as a result of differences in solubility between uranium and its daughters, and/or escape of radon gas.

Knowledge of, and correction for, disequilibrium is important for deposits for which the grade is measured by gamma-ray probes, which measure daughter products of uranium. Disequilibrium is considered positive when there is a higher proportion of uranium present compared to daughters. This is the case where decay products have been transported elsewhere or uranium has been added by, for example, secondary enrichment. Positive disequilibrium has a disequilibrium factor which is greater than 1.0. Disequilibrium is considered negative where daughters are accumulated, and uranium is depleted. This so called "negative" disequilibrium has a disequilibrium factor of less than 1.0 but not less than zero.

Disequilibrium is determined by comparing uranium grades measured by chemical analyses with the "gamma only" radiometric grade of the same samples measured in a laboratory. There are practical difficulties in comparing chemical analyses of uranium from drill hole samples with corresponding values from borehole gamma logging, because of the difference in sample size between drill core (average grades in core or chip samples) and radiometric probe measurements (gamma response from spheres of influence up to 1 m in diameter). Also, any probe calibration (and/or assay) error can be misinterpreted as disequilibrium. If the gamma radiation emitted by the daughter products of uranium is in balance with the actual uranium content of the measured interval (assay), then uranium grade can be calculated solely from the gamma intensity measurement.

Denison routinely compares borehole natural gamma data to chemical assays as part of its QA/QC program as illustrated in the example in Figures 12-1 to 12-9 (Phoenix) and Figures 12-10 to 12-13 (Gryphon). The down-hole depths for gamma results in Figures 12-1 to 12-13 have not been corrected for depth so they do not correspond exactly to the chemical assay depths. Reasonable uranium grades can be calculated from the triple gamma probe (Geiger Mueller, or GM, tube) empirical data up to 80%. Above 80%, the counts (the maximum count rate is about 3,500 cps) increase very little with increased grades due to the physical characteristics of the GM tube (Sweet and Petrie, 2010). In general, radiometric grades are somewhat lower than chemical assay grades because:

- The GM tube can become saturated at very high grades and it cannot count any higher; and
- Some gamma rays are captured by the uranium, converted to photons, and absorbed (self-absorption) (i.e. they are not available to the detector).

Denison and RPA carried out a check of the digital probe database used for resource estimation by verifying the resource database against original assay data. Denison and RPA concluded that, in instances where core recovery was less than 80%, radiometric data could be substituted for chemical assays and that the assay database was of sufficient quality for mineral resource estimation.

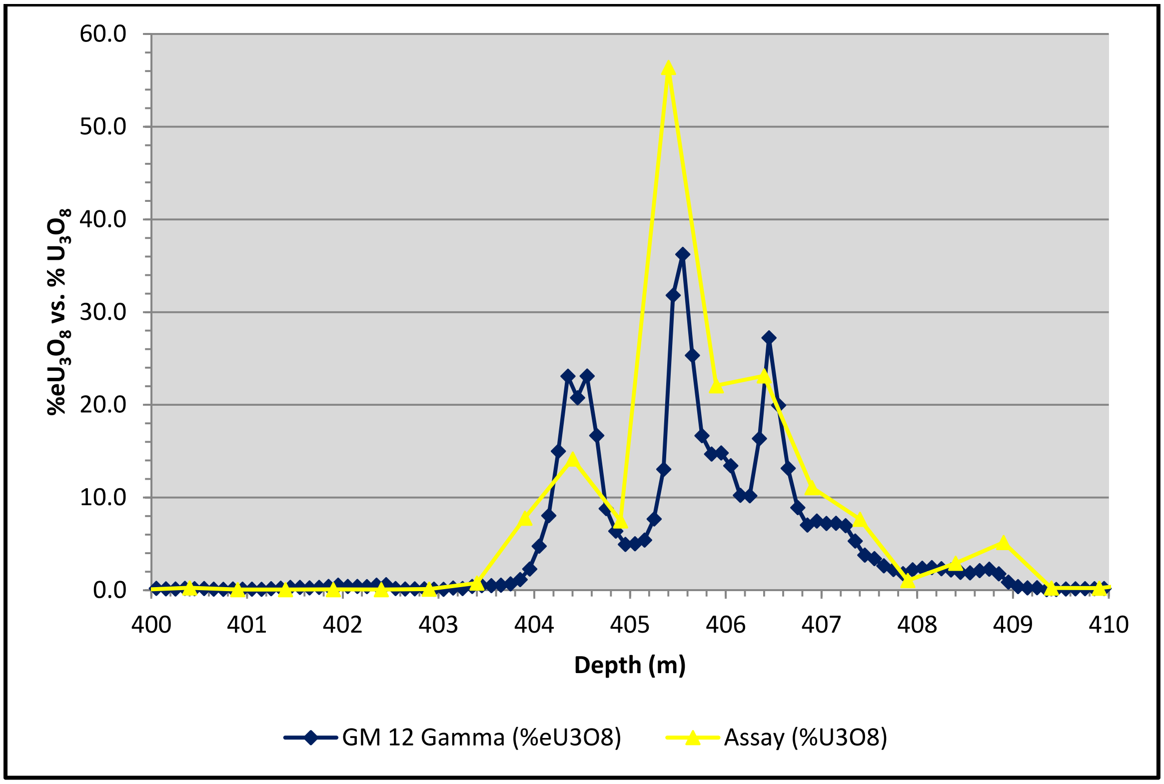


Figure 12-1. WR-318 Radiometric vs Assay % U_3O_8 Values

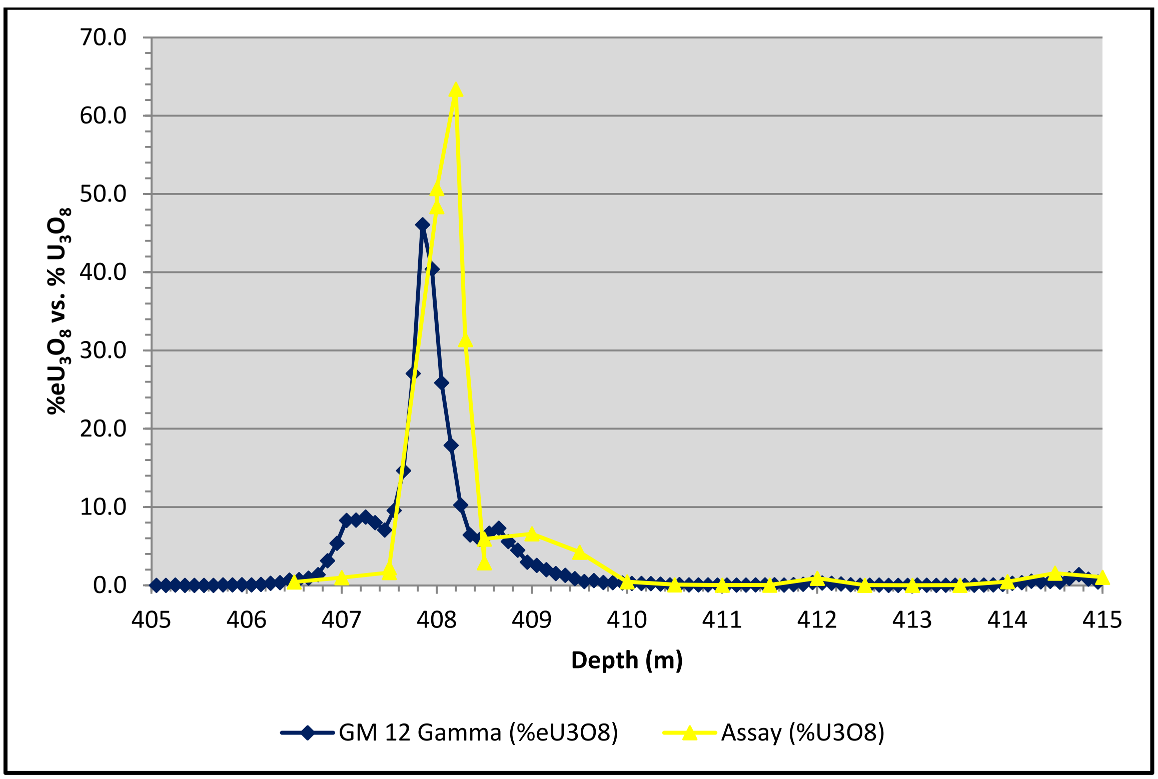


Figure 12-2. WR-334 Radiometric vs Assay % U_3O_8 Values

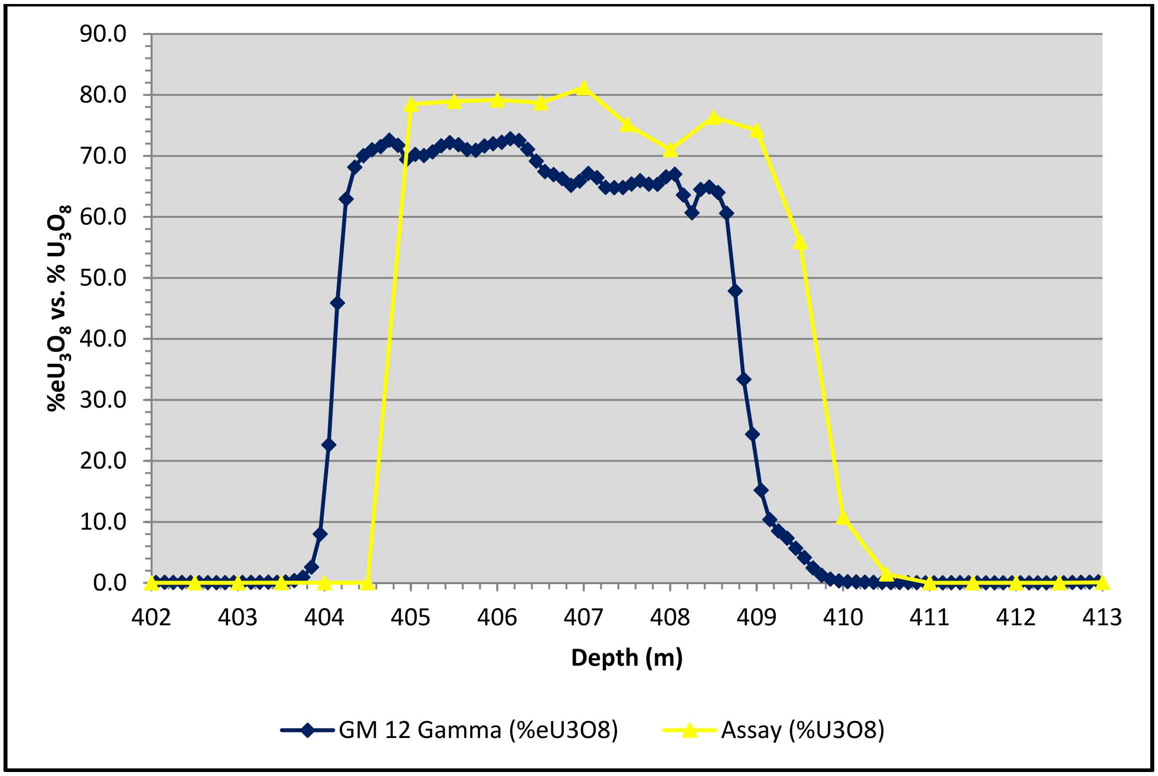


Figure 12-3. WR-273 Radiometric vs Assay % U_3O_8 Values

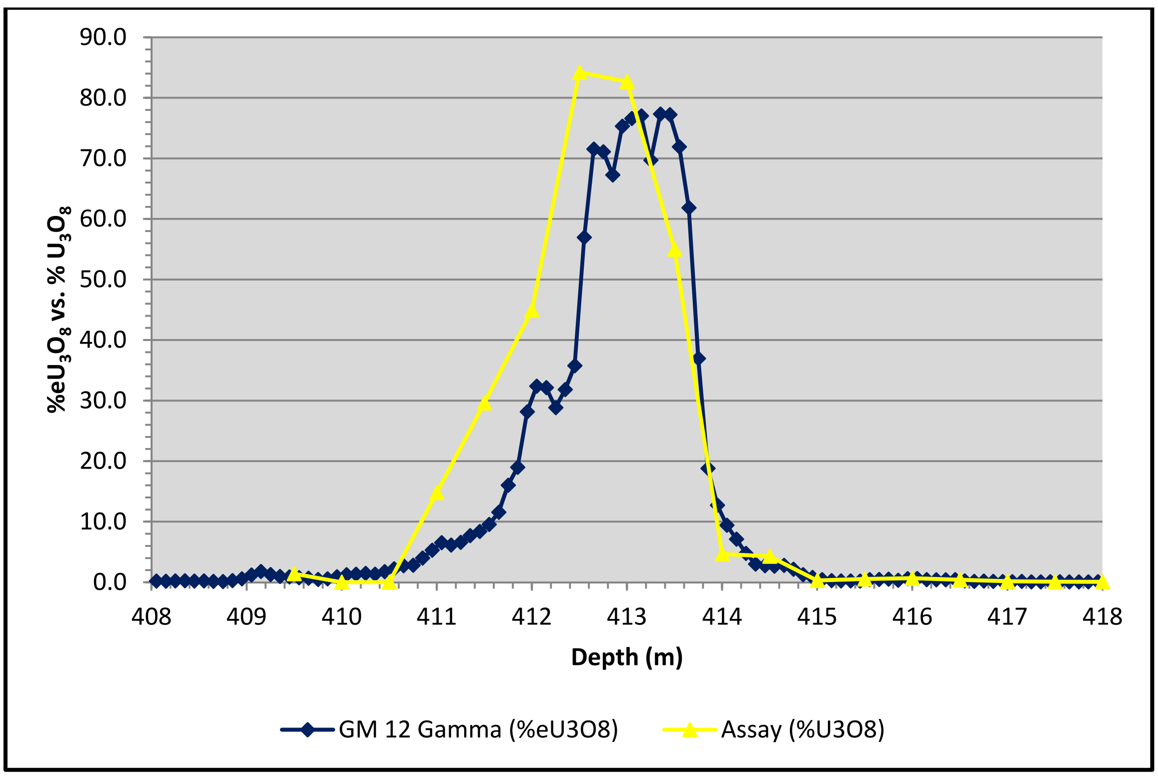


Figure 12-4. WR-435 Radiometric vs Assay % U_3O_8 Values

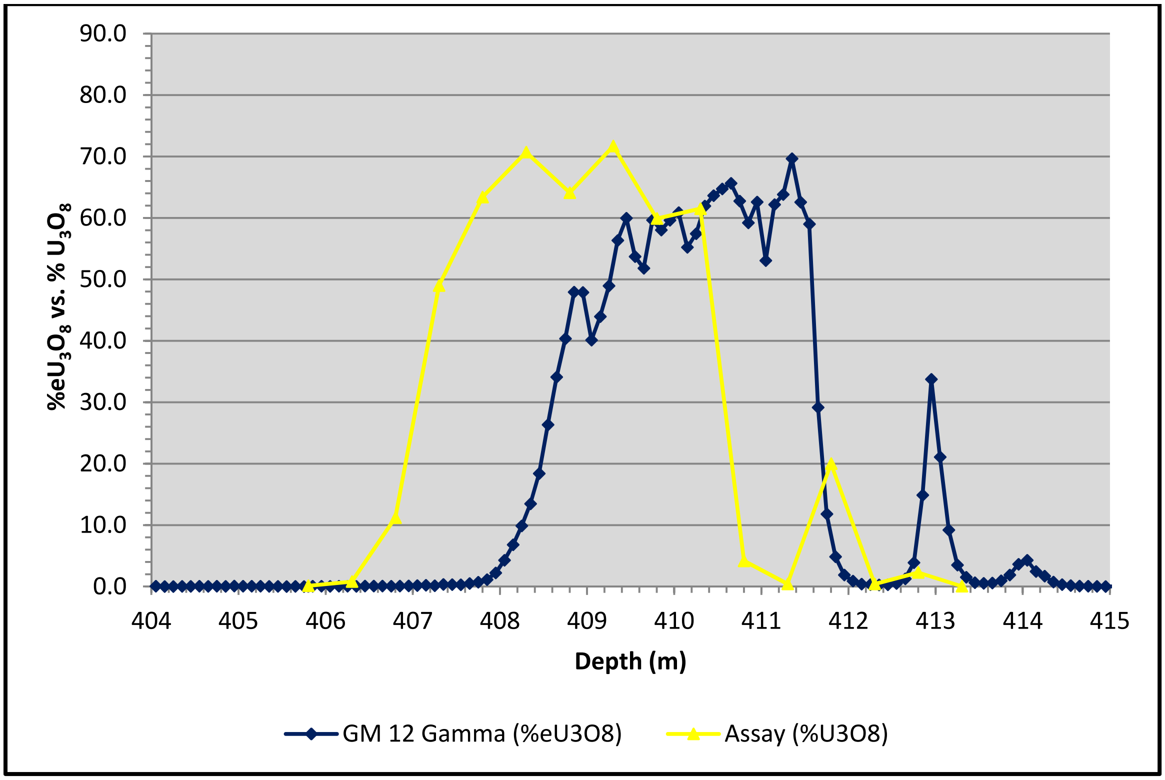


Figure 12-5. WR-548 Radiometric vs Assay % U_3O_8 Values

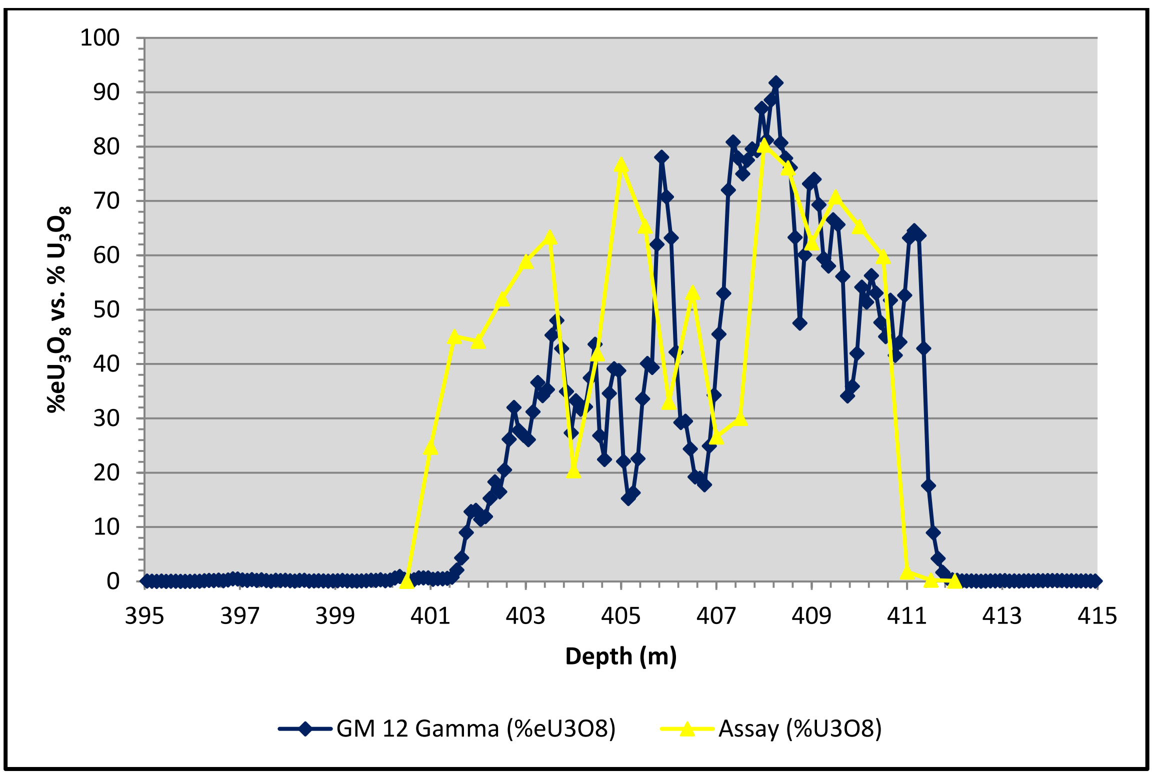


Figure 12-6. WR-525 Radiometric vs Assay % U_3O_8 Values

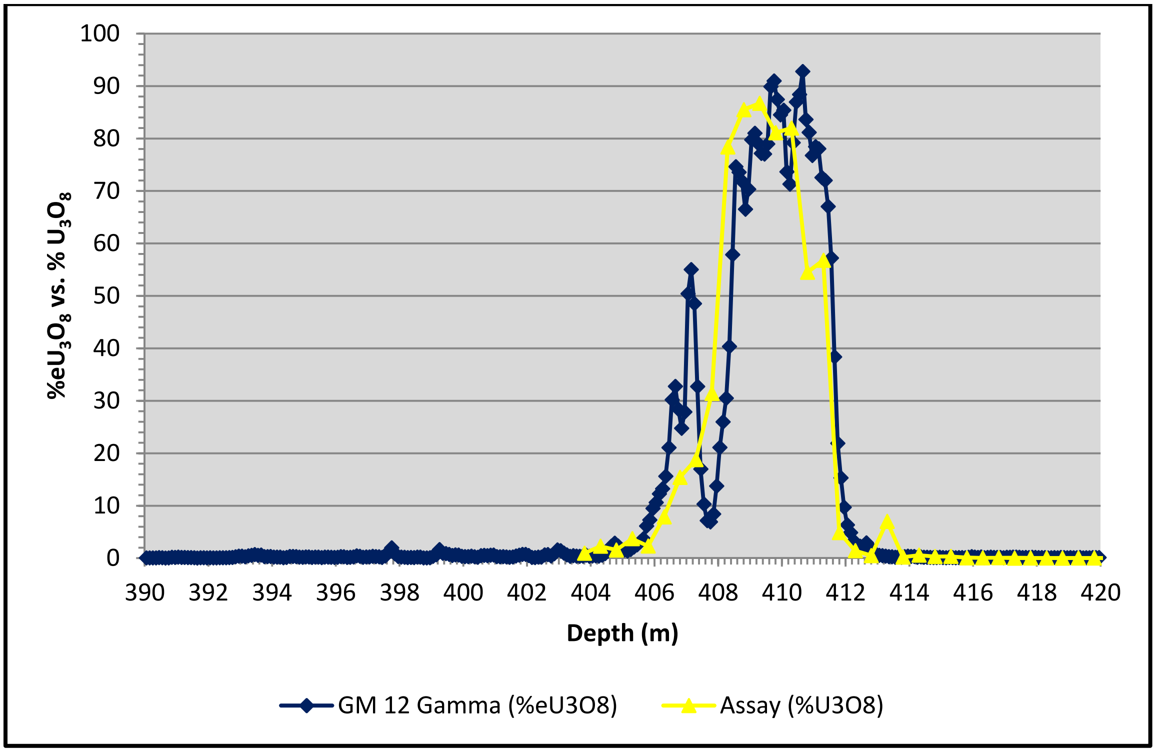


Figure 12-7. WR-401 Radiometric vs Assay % U_3O_8 Values

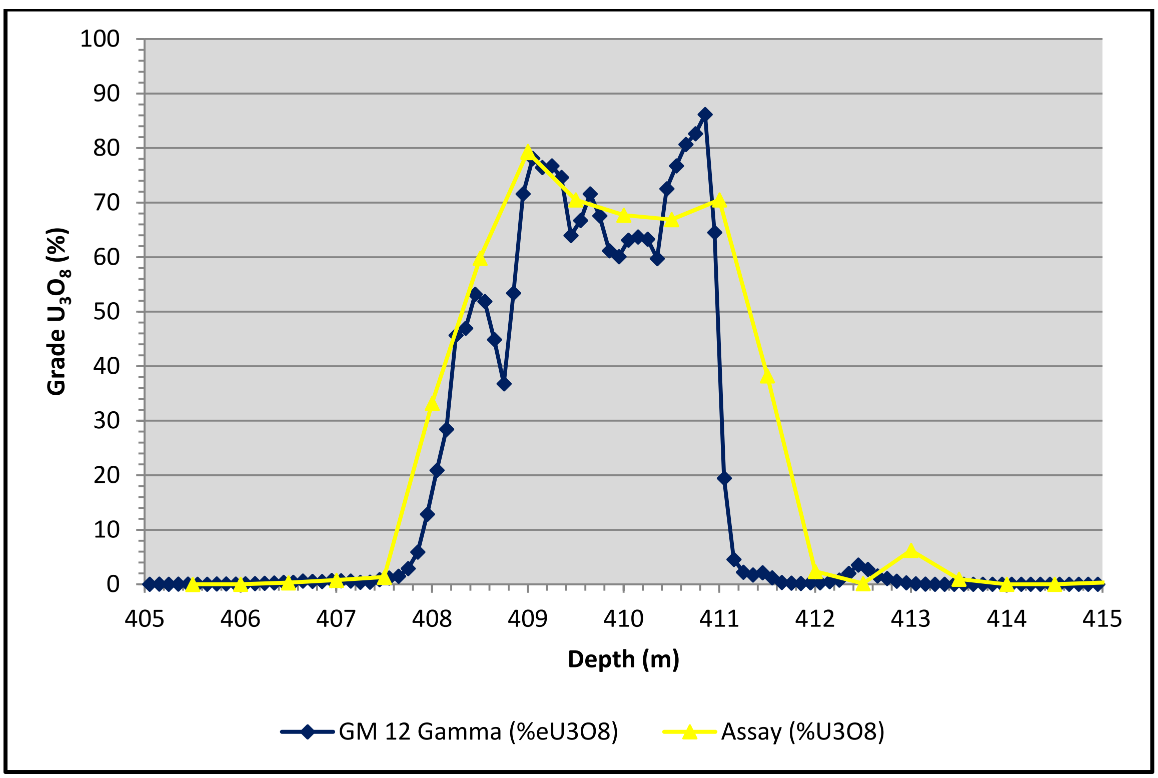


Figure 12-8. WR-306 Radiometric vs Assay % U_3O_8 Values

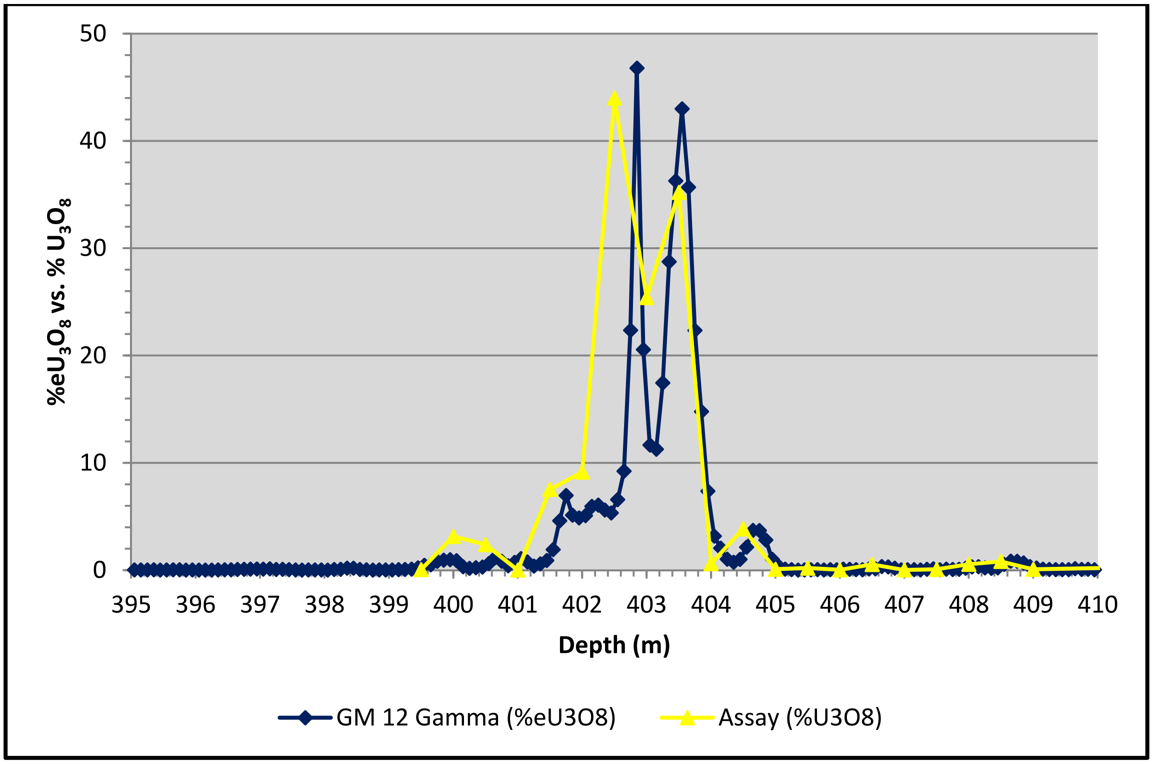


Figure 12-9. WR-539 Radiometric vs Assay % U_3O_8 Values

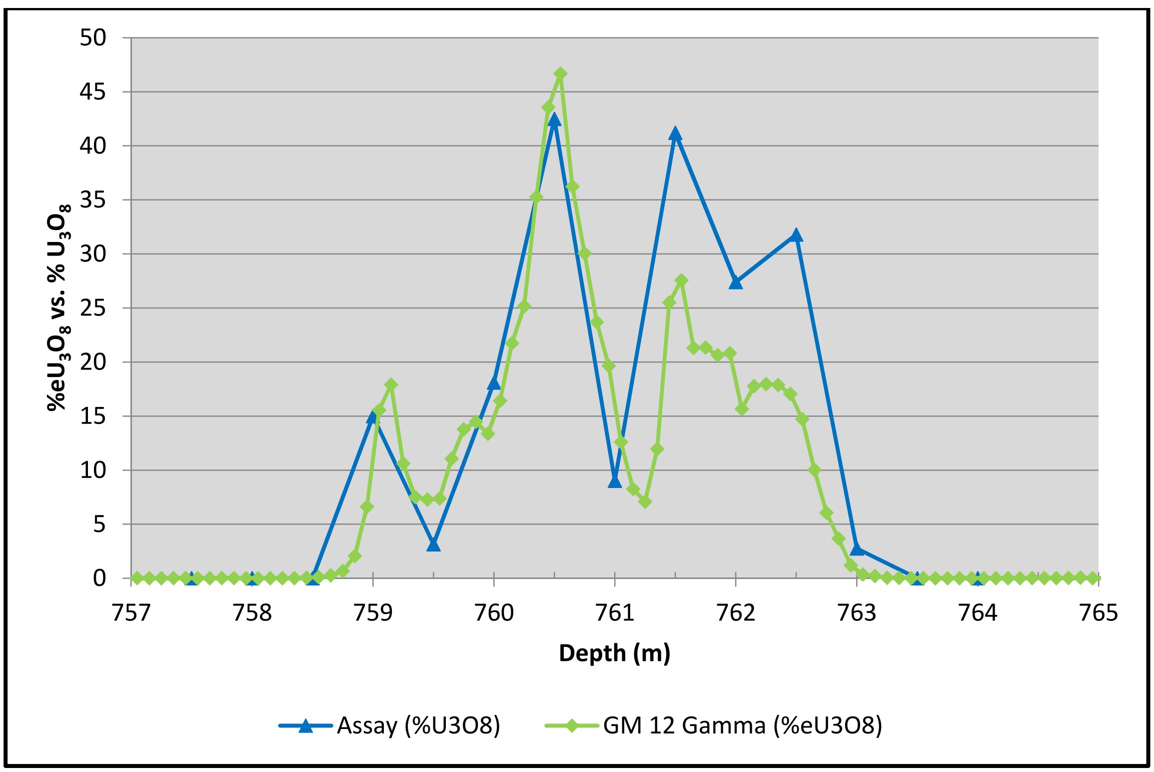


Figure 12-10. WR-560 Radiometric vs Assay % U_3O_8 Values

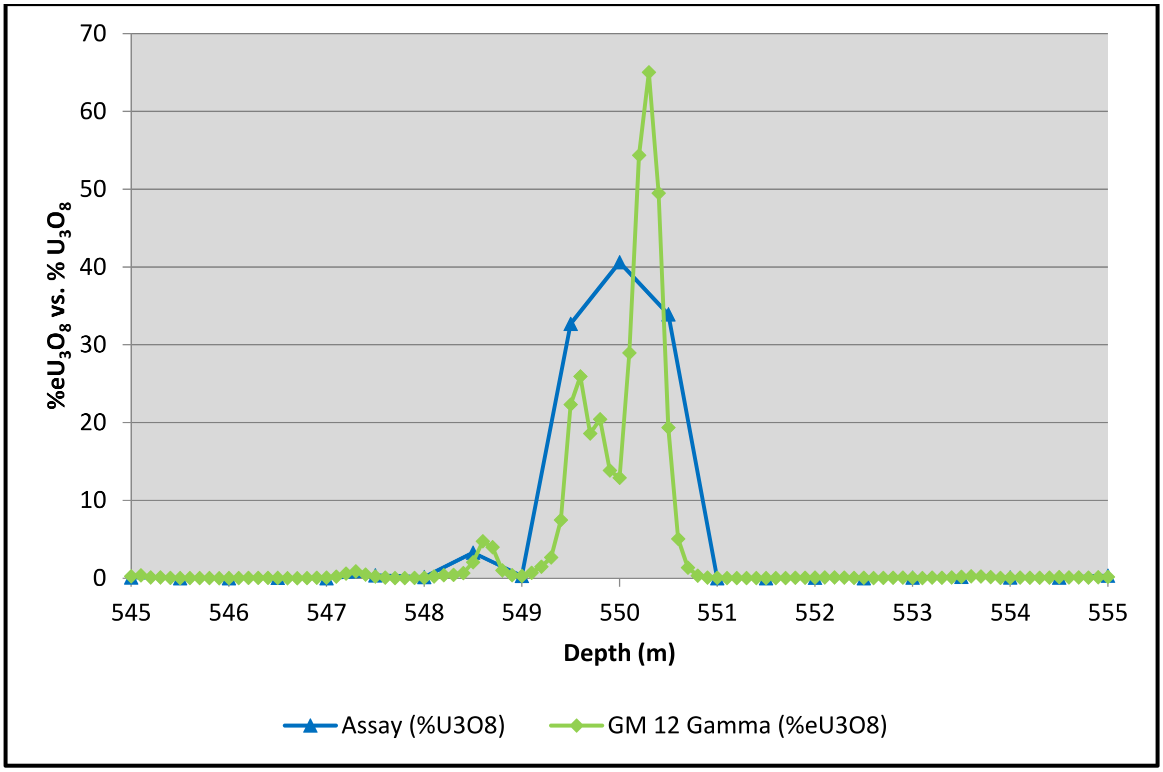


Figure 12-11. WR-573D1 Radiometric vs Assay % U_3O_8 Values

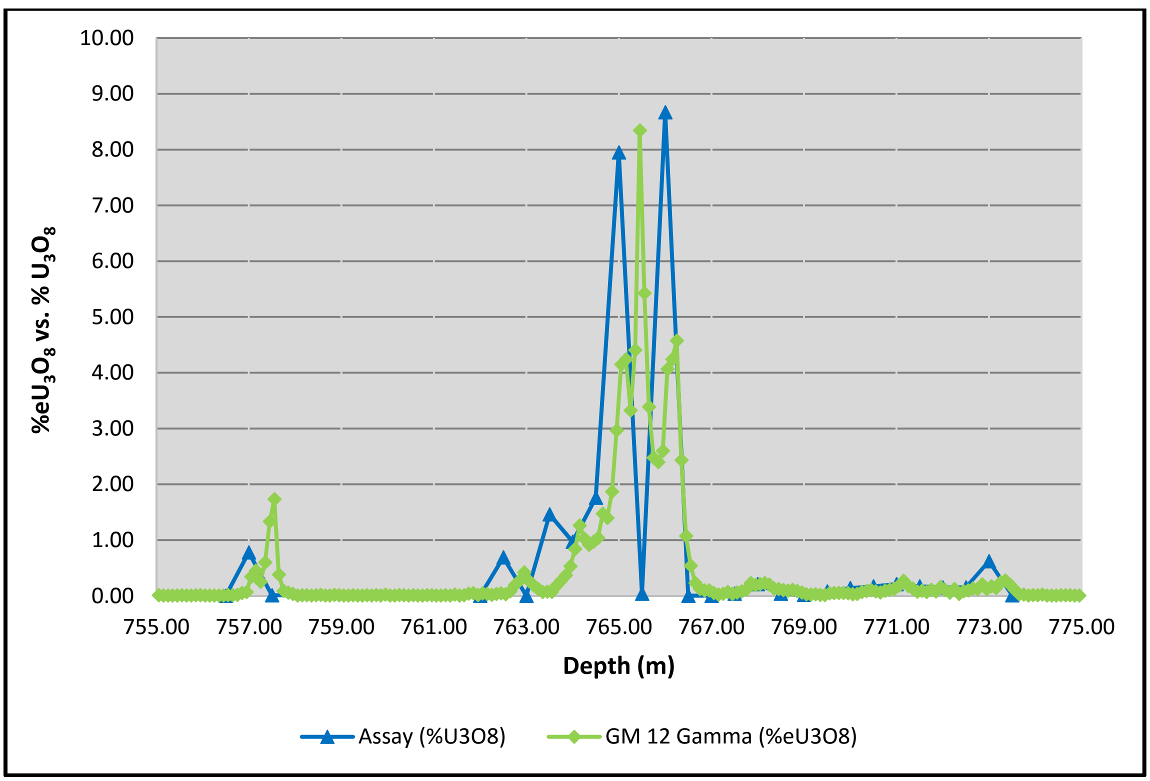


Figure 12-12. WR-582 Radiometric vs Assay % U_3O_8 Values

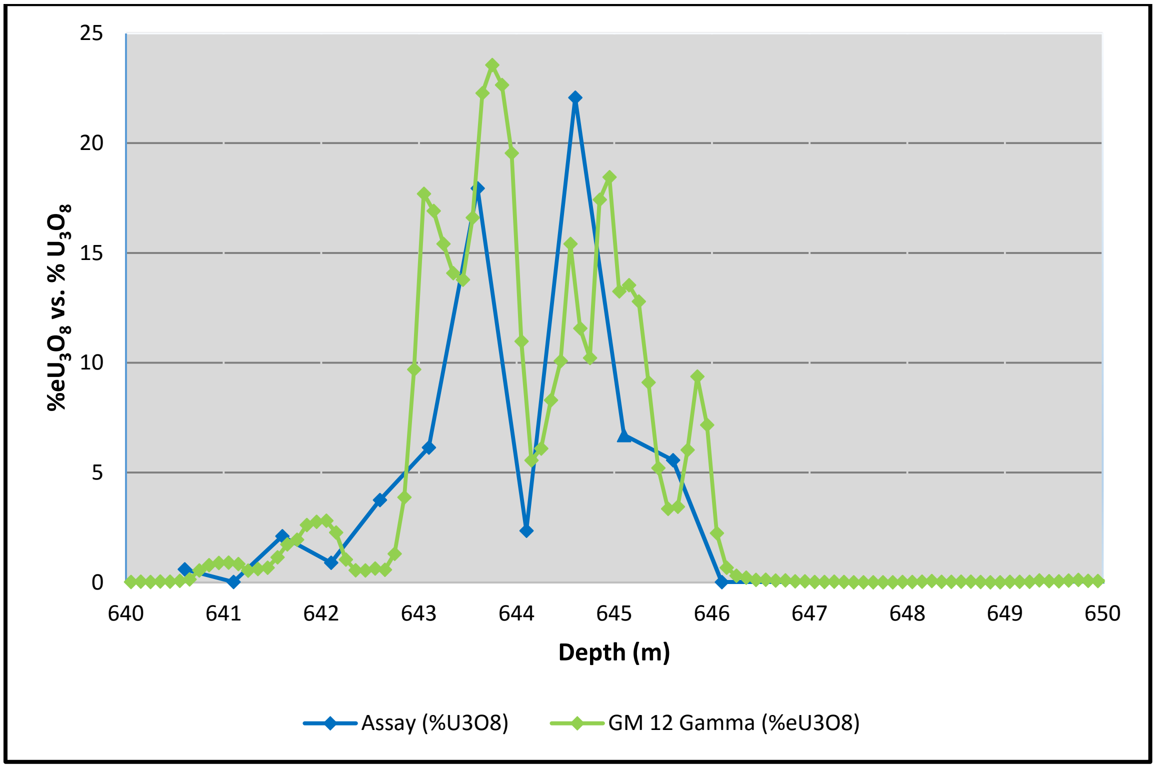


Figure 12-13. WR-584B Radiometric vs Assay % U_3O_8 Values

13 Mineral Processing and Metallurgical Testing

This section provides a description of metallurgical test methods and results, analysis of the results, and comments on the amenability of the Phoenix and Gryphon deposits for mineral processing. The Phoenix deposit mine production is to be processed on site at Wheeler River, whereas the Gryphon deposit mine production is to be processed at Denison's 22.5% owned McClean Lake mill.

13.1 Phoenix Deposit Metallurgical Testing

Uranium in situ recovery (ISR) operations utilize lixiviate solutions consisting of groundwater, a complexing agent, and, typically, an oxidant to effectively extract uranium from the host mineral. To estimate the amount of uranium expected to be recovered from ISR mining operations, agitation and/or column leach tests may be conducted on representative core samples extracted from the deposit. The average recovery from test results may then be used to determine the likely recoverable uranium resource.

Typically, Canadian high-grade uranium deposits have been mined using subsurface techniques with recovered minerals being transported to conventional milling circuits for resource recovery. ISR mining, employing carbonate/bicarbonate or sulfuric lixiviates with or without the addition of oxidants, has become an accepted, if not the predominant, means of uranium recovery in the US (primarily alkaline leach) and in eastern Europe and Australia (typically acid leach). Denison's Wheeler River project appeared to be suitable, from a geological and hydrological perspective, for ISR uranium mining. Accordingly, leach tests were conducted to support the operation, which included batch leach tests and "bottle roll" or "agitation leach" tests with both alkaline and acidic based lixiviates and a follow-up column leach test with acid lixiviate. These tests indicated that recoveries of greater than 90% of contained uranium could be extracted in 120 or more pore volumes of introduced acidic/oxidant lixiviates. Alkaline/oxidant tests did not leach acceptable quantities of the uranium resource.

Denison and Woodard & Curran (formerly TREC) contracted RDE in December 2016 to provide technical support in the setup and evaluation of agitation leach and associated test work on core recovered from the Wheeler River project. RDE subcontracted Inter-Mountain Laboratories Inc. (IML) for conducting the bottle roll and column leach tests since both entities have developed a business relationship performing similar tests in the past. Additionally, RDE relied on the services provided by TetraTech for conducting physical testing for permeability and porosity determination.

IML is a multi-disciplinary company offering services and products related to environmental measurement. The main lab campus is located at 1673 and 1633 Terra Avenue, Sheridan, WY, USA. The scientists working at IML are trained in methods and quality assurance protocols approved by EPA, A2LA, OSHA, NIOSH, and multitude state agencies. IML is a US licensed NRC facility.

Tetra Tech is a leading provider of consulting and engineering services. Tetra Tech's fully automated geotechnical laboratory in the United States includes most geotechnical tests (strength, permeability, compressibility, classification, etc) required for designing infrastructure and resource projects. The laboratory has been inspected and qualified by the U.S. Army Corps of Engineers for geotechnical testing and the Geosynthetic Accreditation Institute Laboratory Accreditation Program for geosynthetics testing and has a United States Department of Agriculture soil permit to import foreign soils for testing purposes. The tests were conducted at 618 S, 25th Street, Billings, Montana, USA.

13.1.1 Sample Selection

In advance of selecting samples for metallurgical testing, Denison completed a statistical analysis of the Phoenix deposit based on available information prior to 2017. Based on this data, Denison established an initial variability assessment of key elemental grade ranges within the deposit (Table 13-1). The Denison sample selection report titled Wheeler River Project Phoenix Sample Selection Methodology dated July 2017 is summarized below.

Table 13-1. Phoenix Deposit Grade Variability

	Min	Avg.	Max	90% Conf. Level
U_3O_8 (%)	0.800	18.48	86.70	38.52
Arsenic (%)	Trace	0.02	1.87	0.041
Molybdenum (%)	Trace	0.03	2.12	0.054
Al_2O_3 (%)	0.06	17.2	36.9	29.4
Fe_2O_3 (%)	0.03	5.4	54.4	13.6
MgO (%)	Trace	4.11	13.80	8.84
CaO (%)	0.01	0.07	20.00	1.42
Cadmium (%)	Trace	0.001	0.285	Trace
Selenium (%)	Trace	0.004	0.228	0.007

Source: Denison - "Phoenix Deposit Metallurgical Sample Selection Methodology"

Denison conducted a drilling program for the sole purpose of recovering material to support metallurgical testing. As a result, holes WR-419, WR-405, WR-287, and WR-525 were identified as prime candidates, offering a wide range of mineral composition. Table 13-2 summarizes assay results and mineral characteristics measured during previous exploration programs.

Table 13-2. Reported Mineral Characteristics within the Phoenix Geological Model

Hole No.	Density (g/cc)	Grade (% U_3O_8)	Al_2O_3 (%) (Note 2)	Fe_2O_3 (%)	Thickness (m)
WR-405	2.51	12.69	13.68	4.60	7.00
WR-419	2.48	9.23	15.86	16.03	6.00
WR-287	2.85	21.73	14.59	12.29	10.33
WR-525	3.22	28.53	8.13	11.97	10.67
Average (Note 1)	**2.76**	**18.04**	**13.06**	**11.22**	**8.50**

Notes:
1. Weighted average based on the geological model.
2. Assuming that the clay carrying mineral is aluminum.

The key parameters assessed for variability include iron oxide, uranium grades, and clay content (Al_2O_3). Other elements, such as arsenic, were at such low values without significant variability that it did not warrant assessment for potential environmental or processing challenges.

Existing drilling in the Phoenix deposit was completed for exploration and mineral resource estimation purposes. Due to the age and outdoor storage of the core, as well as splitting of the core for assaying purposes, the existing core was not deemed suitable for supporting new metallurgical testing. As a result, new drill holes were executed for sampling. The sample retrieval process was designed to twin the original hole intercepts to obtain as similar core samples as possible. Drilling followed the existing boreholes until ~30-50 m above the mineralized zone when the hole was wedged. From that point through the mineral zone, fresh undisturbed core was retrieved, which would be 0-1 m away from the original hole.

The new holes were identified as WR-419D1, WR-405D1, WR-287D1, and WR-525D1 and D2. Figure 13-1 depicts the location of all twinned holes within the Phoenix deposit. Drill hole WR-525 was wedged twice (D1 and D2) providing fresh core material with similar characteristics.

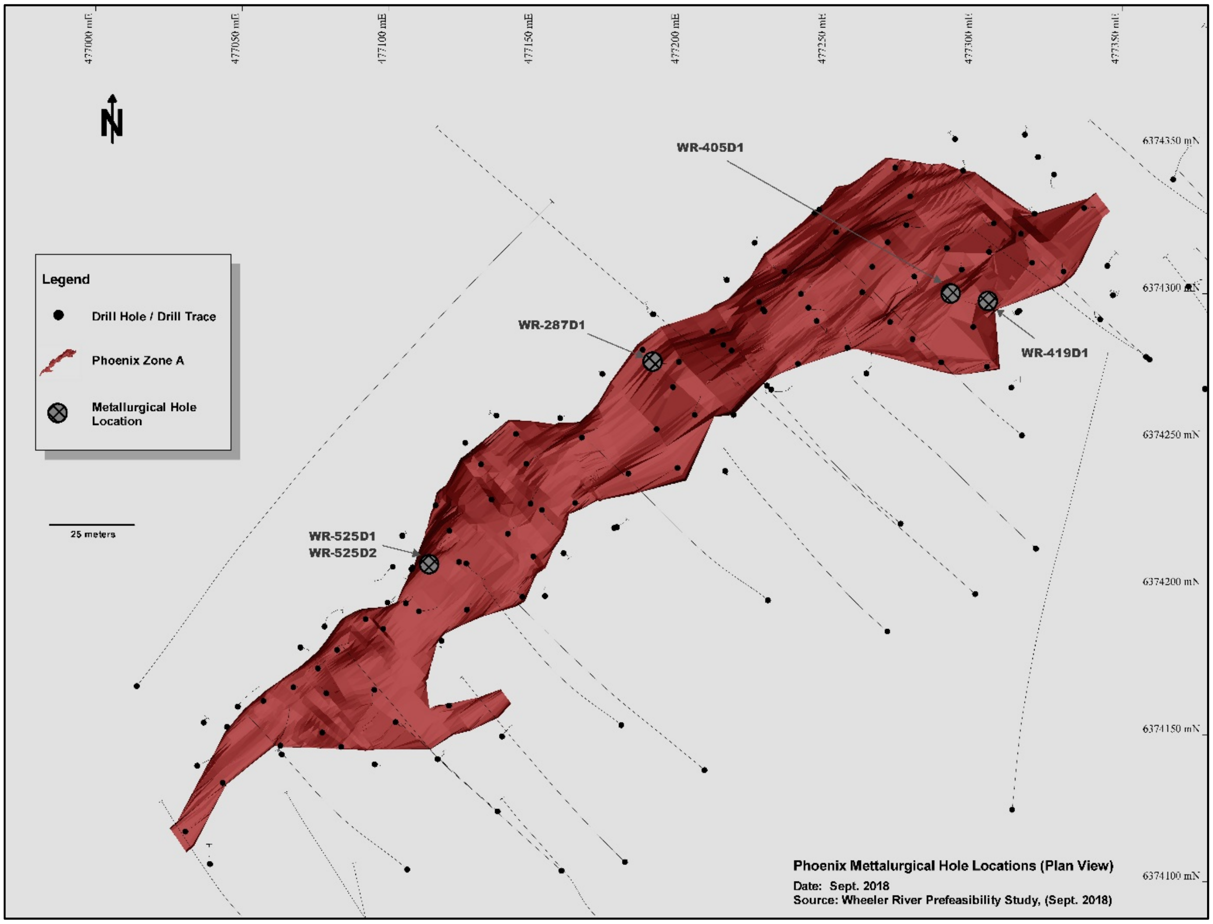


Figure 13-1. Phoenix Metallurgical Hole Locations

Radioactivity measurement was used as an indicator to group each half meter intercepts into the Low, Medium, and High-Grade sub groups. From experience, Denison could correlate radioactivity intensity measurement with actual uranium grade content. This technique was used as a primary sorting tool. Ultimately, each individual sample bag was assayed in preparation of all sub groups targeting uranium content of ~15% U_3O_8 for the Low-Grade, 20% for the Medium-Grade, and 35% for the High-Grade.

All core boxes were sealed and secured at the exploration site until the intercepts were logged, tagged, and preserved. An AMERIVACS AVN-20 industrial sealer complete with a nitrogen purge was used to seal all the bags and preserve the samples. All half a meter intercepts were geologically characterized, logged, and numbered. The same numbers were referenced for the various different tests whenever the material was used for conducting metallurgical testing.

High and low intercepts were mixed together to produce Low-Grade and High-Grade sub groups. This practice also ensures the incorporation of clay and other different minerals in each sub group mixes. Finally, 50.5 kg of high-grade material was preserved and originally stored at the Wheeler River exploration camp, and ultimately used for completing the metallurgical testing program. In total, around 220 kg of material containing a wide range of uranium and clay content was tested in many different facilities.

Three Phoenix composites were prepared as summarized in Table 13-3. QEMSCAN mineralogy analysis was completed on each of the composites, as summarized in Table 13-4. The samples tested are considered to be reasonably representative of the commercial deposits for a PFS based on the deposit variability data.

Table 13-3. SRC – Phoenix Composite Sample Assay Results

Sample		Low Grade	Medium Grade	High Grade
U_3O_8	%	10.94	17.10	37.15
Al_2O_3		7.68	20.6	18.1
Fe_2O_3		5.23	6.66	7.91
CaO		0.59	0.82	1.53
MgO		1.26	5.08	4.545
K_2O		0.58	1.31	0.785
TiO_2		0.3	0.93	0.69
Mo	ppm	186	151	192
As		245	334	438
Ni		374	549	516.5
Pb		8,730	13,100	24,250
Co		98	96	133.5
Cu		9,670	5,900	2,115
V		498	1,020	742
Zn		471	1,230	236

Source: SRC - "Denison Uranium Ore Metallurgical Testing Part One: Phoenix Acid Leaching and Settling"

Table 13-4. SRC – Phoenix Mineralogy

Mineral	Ideal Formula	Phoenix Low Grade	Phoenix Medium Grade	Phoenix High Grade
Pyrite	FeS_2	7.98	7.30	4.02
Pyrrhotite	$Fe_{1-x}S$ (x = 0 to 0.17)	0.04	0.06	0.03
Galena	PbS	3.55	2.61	1.24
Sphalerite	(Zn,Fe)S	0.03	0.36	0.10
Chalcopyrite	$CuFeS_2$	0.40	2.39	3.61
Quartz	SiO_2	0.10	10.59	58.68
Feldspar	$KAlSi_3O_8$	0.00	0.05	0.07
Muscovite/Illite	$KAl_2(Si_3Al)O_{10}(OH; F)_2$	4.90	9.39	4.64
Chlorite	$(Fe,Mg)_2(Al; Fe^{3+})_3Si_3AlO_{10}(OH)_8$	30.67	41.93	9.68
Titanite	$CaTiSiO_5$	0.00	0.00	0.00
Zircon	$ZrSiO_4$	0.01	0.04	0.05
'Kaolinite' (clay)	$Al_2Si_2O_5(OH)_4$	2.82	2.30	2.52
Calcite	$CaCO_3$	0.00	0.00	0.04
Apatite	$Ca_5(PO_4)_3(F,Cl,OH)$	0.40	0.01	0.13

Mineral	Ideal Formula	Phoenix Low Grade	Phoenix Medium Grade	Phoenix High Grade
Rutile/Anatase	TiO_2	0.82	1.46	0.51
Fe-oxy-hydroxides	$FeO(OH)\cdot nH_2O$	5.36	2.16	3.99
Corundum	Cr_2O_3	0.02	0.01	0.00
APS Minerals	$CaAl_3(PO_4)(PO_3OH)(OH)_6$	0.16	0.24	0.27
Uraninite	UO_2	42.72	19.10	10.39
Gersdorffite	NiAsS	0.00	0.01	0.00
Nickeline	NiAs	0.00	0.01	0.00

Source: SRC - "Phoenix and Gryphon Uranium Ore Metallurgical Testing Part Three: QEMSCAN Mineralogy Analysis"

13.1.2 Core Samples, Physical Testing Results

Table 13-5 presents results of physical core testing provided by Tetra Tech. Only core located peripheral to the high-grade deposit (excepting sample number S066910) was provided due to the lack of physical integrity (competence) of the target, high uranium grade resource.

Table 13-5. Permeability Test Report Summary

Sample ID	Units	SO66905	S066906	S066907	S066908	S066909	S066910
Hole ID	-	WR-419D1	WR-419D1	WR-525D2	WR-405D1	WR-405D1	WR-525D2
Interval Start	m	396.05	408.26	398.31	389.2	408.15	410.45
Interval End	m	396.18	408.39	398.38	389.3	408.25	410.55
Lithology	-	SDST	BSMT	SDST	SDST	BSMT	BSMT
Estimated U	%	0.58	0.22	0.06	0.06	0.08	51.72
Flow Rate	cc/sec	4.34E-05	7.26E-05	1.27E-01	1.81E-04	6.96E-06	6.77E-02
Permeability	cm/sec	7.8E-08	1.41E-07	1.23E-04	2.40E-07	9.37E-09	1.16E-04
	millidarcy (md)	0.08	0.14	128	0.25	0.01	121
Specific Gravity	-	2.67	2.71	2.65	2.65	2.67	6.61
Porosity	Decimal %	0.1329	0.064	0.1699	0.1197	0.1802	0.2593

Source: TREC Phoenix Deposit ISL Evaluation Post Testing Report, November 6, 2017

13.1.3 Acid Agitation Leach Test Results

Complete results, including supportive data tables and charts, are located in the appendices of the report entitled “Agitation and Column Leach Testing of Wheeler River Ores in Support of In-situ Uranium Recovery” dated September 5, 2018.

Sulfuric acid/hydrogen peroxide agitation (bottle roll) leach testing of the Phoenix sample was initiated in August 2017 from the composited core samples. Following the initial 30 pore volume (PV) staged test, U resource recovery totaled ~31%. Comparisons to usual roll front U deposits, which typically are depleted after this leach period, were not possible considering the average ~30% U concentration in the tested sample, as lixiviate leach agents at typical ISR concentrations were aggressively consumed.

Several additional periods of sulfuric/peroxide leach at various lixiviate concentrations were tested, finally resulting in a ~90% U resource recovery at 120 PVs. Table 13-6 represents results of the various lixiviate concentrations used to mobilize the highly concentrated uranium resource during the agitation leach test. Complete results are available in the report entitled "ISR Testing of Denison Wheeler Uranium Deposit" dated November 5, 2017.

Table 13-6. Agitation Leach Data Analysis

Leach Phase	Lixiviate	PV Tot	Vol Rec (L)	U Rec (g)	U Conc. (g/L U)	U_3O_8 (g/L)	% U Rec (g)	% U Rec/PV	Acid Consumption kg H_2SO_4/kg U
Pre-conditioning	25 g/L H_2SO_4 2.5 g/L H_2O_2	5	0.157	8.90	56.6	66.7	6.9%	1.4	0.34
Sulfuric Leach	5 g/L H_2SO_4 2.5 g/L H_2O_2	15	0.910	17.4	19.1	22.6	11.8%	0.79	0.23
Sulfuric Leach	7.5 g/L H_2SO_4 2.5 g/L H_2O_2	60	3.72	57.7	15.5	18.3	39.2%	0.65	0.16
Sulfuric Leach	10 g/L H_2SO_4 2.5 g/L H_2O_2	40	2.44	46.8	19.2	22.6	31.8%	0.80	0.57
Totals/Avg		**120**	**7.23**	**130.8**	**18.1**	**21.3**	**89.7**	**0.75**	**0.32**

Initial ISR testing of Phoenix high-grade uranium samples using the agitation leach technique demonstrated 90% recovery of the resource. The test results indicated that, by using acid leach lixiviates, average recovery grades of 18.08 g/L U could potentially be maintained over 120 pore volumes (PV). Physical testing of the sample composite and the selected competent core sections demonstrated that porosity and permeability within the high-grade zone could be favorable for ISR operations.

Agitation leach or "bottle roll" tests are limited; however, in identifying potential geochemical difficulties which could be encountered in the ISR process, such as gas-locking or gypsum formation. The agitation leach test did provide data relative to:

- Pore volumes of leach required for efficient resource recovery;
- Necessary lixiviate acid concentrations;
- Demonstrated that additional Fe will probably not be needed for U oxidation (namely Fe^{2+}/Fe^{3+} couple);
- Established potential for direct uranium peroxide precipitation from leach solutions; and
- Established expected acid consumption in kg H_2SO_4/kg U.

13.1.4 Acid Column Leach Testing

Based on the results of the agitation leach test, it was decided that further testing of material collected from the deposit, including acidic lixiviate column leach tests, would assist in determining if problems such as gas-locking or gypsum formation would be process concerns. The benefits of performing column leach testing as compared to relatively simple agitation leach testing include:

- Recycle of recovered leach fluids to refortified injection, which provides information on the extent of chemical constituent concentration buildup during extended leaching periods;
- Evaluation of potential formation gas locking or plugging with precipitates (gypsum);
- Determination of the effect of "lost" physical material (uranium in high-grade >5%) on formation permeability and porosity as leaching progresses;
- Establishing sulfuric acid and oxidant requirements for most efficient leach; and

- Collecting adequate recovery fluids for surface process evaluation, including direct peroxide precipitation of $UO_4{\cdot}2H_2O$ prior to lixiviate refortification and reinjection.

Approximately 3 kg of the high-grade Wheeler River core was used for performing a column test. Table 13-7 summarizes the material which was composited into a single column charge.

Table 13-7. Sample Inventory for ISR Column Leach Testing

Original Sample Purpose	Sample I.D.	WR Hole No.	Depth Int. (m)	Lithology	Est. U%	Mass (g)	Mass U (g)
Porosity/Perm.	S066906	419D1	408.2-408.4	BSMT	0.22	320	0.61
Porosity/Perm.	S066907	525D2	398.3-398.4	SDST	0.06	323	0.17
Porosity/Perm.	S066908	405D1	389.2-389.3	SDST	0.06	270	0.14
Porosity/Perm.	S066909	405D1	408.2-408.3	BSMT	0.08	299	0.21
Porosity/Perm.	S066910	525D1	410.4-410.5	BSMT	51.72	843	375
Leach Testing	S066911	525D1	398.9-399	SDST	0.06	282	0.17
Leach Testing Composite Sample	S066912-S066916	525D1 525D2	403.1-412.4 @ Select Ints.	SDST & BSMT	29.4	1,090	276
Leach Testing Total Composite Sample	S066906-S066916	405D1 419D1 525D1 525D2	403.1-412.4 @ Select Ints.	SDST & BSMT	19.03 (wet)	3,427 (wet)	652.3

Source: RDE "Agitation and Column Leach Testing of Wheeler River Project Ores to Support In-Situ Uranium Recovery", dated September 5, 2018

A composite sample charge utilizing the core material described above was generated and delivered to the column test. The composite column leach test head sample was characterized prior to testing and the results are presented in the first column of Table 13-8.

Table 13-8. Column Leach Test Feed Ore Characteristics

Parameter	Units	Results	
		Heads (Assayed)	Heads (Calculated) (Note 1)
CO3	%	<0.1	
CEC	meq/100g	14.9	
Al	mg/kg	29,100	29,600
As	mg/kg	382	139
Ba	mg/kg	97.4	81
B	mg/kg	1,800	86
Cd	mg/kg	24.2	3
Ca	mg/kg	7,290	3,624
Cr	mg/kg	227	23
Cu	mg/kg	6,640	2,424
Fe	mg/kg	41,600	38,520
Pb	mg/kg	16,800	3,832
Mg	mg/kg	3,620	2,352
Mn	mg/kg	489	252
Mo	mg/kg	332	74
Ni	mg/kg	183	96
P	mg/kg	469	188
K	mg/kg	130	697
Se	mg/kg	926	54

Parameter	Units	Results	
		Heads (Assayed)	Heads (Calculated) (Note 1)
Na	mg/kg	261	383
Sr	mg/kg	0.21	44
V	mg/kg	370	196
Zn	mg/kg	625	355
S	mg/kg	33,600	28,970
U	mg/kg	242,000	235,040
^{226}Radium (Note 2)	pCi/g	55,700 ± 25.9	
	Bq/g	2,061 ± 1.0	
^{230}Thorium (Note 2)	pCi/g	66,600 ± 1,590	
	Bq/g	2,464 ± 58.8	
^{232}Thorium (Note 2)	pCi/g	993 ± 197	
	Bq/g	36.7 ± 7.3	
^{210}Lead (Note 2)	pCi/g	20,000 ± 34.8	
	Bq/g	740 ± 1.3	
^{210}Polonium (Note 2)	pCi/g	170 ± 6.3	
	Bq/g	6.3 ± 0.2	

Notes:

1. Final solids residues U content assays X mass + U extracted during leaching
2. Radiochemistry on heads assays only

The second column depicts a second method for estimating parameter feed concentration. The post-testing leach test solids residues were recovered and assayed for most parameters listed above. For some elements, such as Al, Fe, Na, and U, the two different methods of calculating parameter feed concentration provide similar results. For others, such as B, Ca, Se, Pb, and Sr, the spread is significant. It is difficult to prepare representative solids composite samples when assaying for trace elements. For the main elements of concern, such as uranium and iron, the assay result correlation is satisfactory.

13.1.5 Column Leach Data Analysis

In situ leach processes typically involve injection of a lixiviant that is chemically suitable to oxidize complex uranium, within a suitably mineralized zone. Uranium bearing solutions are recovered through wells completed in the mineralized zone at some distance from the injection wells. The solubilized uranium is then removed to a surface plant and the barren leach solution is refortified with the lixiviate chemicals and returned to the injection wells. Most uranium ISR operations are conducted on low-grade deposits (0.03-0.2% U) and the uranium extraction is performed using ion exchange media. The Phoenix deposit is very high-grade (grade for column test was 24.2% U) and resultant solutions should contain 10 to 50 g/L U as $[UO_2(SO_4)_3]^{4-}$. The uranium grades observed in the agitation leach tests suggested that direct uranium precipitation from solutions should be possible with the resultant barren solution returned to the wellfield leach circuit.

During February 2018, a 5.1 cm (2 inch) diameter column was filled with composited core detailed in Tables 13-7 and 13-8 above. Flow of deionized water (based on the known low TDS of the mineralized zone groundwater) to the column was initiated February 23, 2018 to simulate baseline aquifer conditions prior to initiating ISR mining. Mineral zone pre-conditioning flow was started on February 24, 2018 using ~30 g/L sulfuric acid. Sodium chlorate (5 g/L) was added as an oxidant, as preliminary testing conducted prior to the agitation leach tests (2017) indicated that the addition of an oxidant would maximize uranium leach rates and efficiency. Hydrogen peroxide was successfully used on the agitation leach test, however, concerns about the potential of precipitating uranium peroxide in the column suggested the use of chlorate as an alternate oxidant. Twenty pore volumes (PV) of the pre-conditioning lixiviate were recovered from the saturated up flow column operation over a 5-day period (4 PV/day).

The column leach test was conducted in five mining phases to provide for the direct precipitation and observe the effect of recycling post-precipitate barren solution refortified with sulfuric acid and chlorate to desired concentrations. A total of 137 PV of uranium bearing solutions were recovered from the column over a two-month period (68 days at an average 2 PV/day). An additional "pre-restoration" leach was conducted over a 10-day period, recovering 16 PV from injection of a 10 g/L sulfuric acid solution with no additional oxidant. The leach phase resulted in ~90% uranium resource recovery at average solution grades of ~8,400 mg/L U.

Table 13-9 summarizes the data from the leach portion of the column test.

Table 13-9. Column Leach Test Conditions and Results

Test Condition	**Result**
Sample Source (see tables above)	405D1, 419D1, 525D1, 525D2
Column Size (overall)	2 inch/5.1 cm I.D. x 32 inch/81.3 cm H
Column Charge Size (initial)	2 inch/5.1 cm I.D. x 27 inch/68.6 cm H
Column Charge Size (final)	2 inch/5.1 cm I.D. x 24 inch/61.0 cm H
Specific Gravity Calc	1.95 g/cm^3
Porosity	~36%
Pore Volume	~500 mL
Mass of Charge (dry)	3.13 kg
Sample % Moisture	<1.0%
Feed Solution Flow Rate	~2-4 Pore Volumes/Day
	~0.7-1.4 mL/min
Pre-Conditioning Phase Lixiviate [H_2SO_4 g/L] [$HClO_3$ g/L]: Duration	29.9 g/L, 5 g/L: PV 0-20
Mining Phase I Lixiviate [H_2SO_4 g/L] [$HClO_3$ g/L]: Duration	22.5 g/L, 2.5 g/L: PV 20-46
Mining Phase II Lixiviate [H_2SO_4 g/L] [$HClO_3$ g/L]: Duration	15.4 g/L, 2.5 g/L: PV 47-77
Mining Phase III Lixiviate [H_2SO_4 g/L] [$HClO_3$ g/L]: Duration	15.3 g/L, 2.5 g/L: PV 77-107
Mining Phase IV Lixiviate [H_2SO_4 g/L] [$HClO_3$ g/L]: Duration	15.0 g/L, 2.5 g/L: PV 107-137
Pre-Restoration Phase Lixiviate [H_2SO_4 g/L]: Duration	10.2 g/L, No Ox.: PV 138-154
Feed Lixiviate ORP (mv)	+810-850 mv
Feed Lixiviate pH (standard units; s.u.)	1.00-1.35
Sample Grade (calculated/assayed)	235,000 mg/kg (24.2%)
Available U (calculated/assayed)	735.55 g/757.5 g
U Recovered by Leach Flow (calculated/assayed)	659.8 g/681.8 g U
U Recovered in Post Leach Flow	14.5 g U
U in Tailings	61.2 g U
U mg/kg Remaining in Tailings (calc)	32,100 (3.21%)
% U Recovered (calculated/assayed)	89.7%/90.0%
% U_3O_8 Recovered in Post Leaching	2.0%/1.9%
% Uranium Overall Recovery (calculated/assayed)	91.7%/91.9%
Pore Volumes Recovered (leach and pre-rest)	154
% U Recovered per Pore Volume (leach)	0.59%

Test Condition	Result
Peak U Solution Grade mg/L	27,400 mg/L U
Average U Solution Grade mg/L	8,411 mg/L U
H_2SO_4 Consumption kg acid/kg U	0.53 kg/kg
Tailings Mass (dry)	1.905 kg
Mass Lost to Leach	1.225 kg
% Mass Lost to Leach	39.0%

Lixiviant chemistries used to achieve the leach recovery are summarized in Table 13-10 below.

Table 13-10. Chemistry of Column Leach Phases Injection Lixiviant

Sample ID/ Parameter	Units	Wellfield Conditioning	Mining Phase I	Mining Phase II	Mining Phase III	Mining Phase IV	Pre-restoration Phase
Lixiviant	H_2SO_4 $NaClO_3$	30 g/L 5 g/L	22.5 g/L 2.5 g/L	15.4 g/L 2.5 g/L	15.3 g/L 2.5 g/L	15 g/L 2.5 g/L	10.2 g/L 0 g/L
pH	Note 1.	1.00	1.29	1.32	1.28	1.26	1.35
Elec Cond	umho/cm	138,000	69,600	71,100	69,700	69,000	75,800
ORP	mV	+505	+850	+842	+812	+845	+845
Free Acid	g/L	29.9	22.5	15.4	15.3	15	10.2
Al	mg/L	N/A	56.8	84.5	118	176	0.84
As	mg/L	N/A	3.61	3.2	6.0	5.43	<0.01
Ba	mg/L	N/A	<0.1	<0.1	<0.1	<0.1	<0.1
B	mg/L	N/A	1.6	6.6	2.0	1.4	2.0
Cd	mg/L	N/A	0.081	0.097	0.093	0.147	0.093
Ca	mg/L	N/A	229	240	256	273	465
Cr	mg/L	N/A	0.4	0.52	0.47	0.52	0.1
Cu	mg/L	N/A	54.7	51.7	57.8	74.9	0.6
Fe	mg/L	N/A	366	551	574	691	2.3
Pb	mg/L	N/A	5.43	3.66	2.84	3.71	0.14
Mg	mg/L	N/A	36.0	26.7	34.7	29.3	61.1
Mn	mg/L	N/A	12.6	18.4	18.2	22.3	12.8
Mo	mg/L	N/A	3.18	4.03	3.76	3.32	0.81
Ni	mg/L	N/A	6.65	6.0	5.55	5.84	5.43
P	mg/L	N/A	8.3	8.0	8.9	8.5	<0.1
K	mg/L	N/A	5.2	10.4	5.4	5.0	27.2
Se	mg/L	N/A	0.571	<0.025	<0.025	0.381	1.93
Na	mg/L	N/A	7,720	11,500	11,000	12,700	26,100
Sr	mg/L	N/A	1.28	0.88	0.53	0.38	N/A
V	mg/L	N/A	11.3	9.35	10.3	8.3	0.06
Zn	mg/L	N/A	5.56	10.0	10.6	19.4	4.2
SiO_2	mg/L	N/A	133	140	162	185	13.6
SO_4	mg/L	N/A	28,560	44,100	26,880	31,200	57,000
U	mg/L	N/A	3,530	1,130	296	472	13.4
U	mg	N/A	52,950	18,080	4,736	7,552	134
Volume	mL	10,000	15,000	16,000	16,000	16,000	10,000

Note 1: All pH measurements reported throughout section 13 are expressed in standard units
N/A: Not available

Figures 13-2 and 13-3 below illustrate the leach recovery grade and percent uranium recovery with increasing pore volumes.

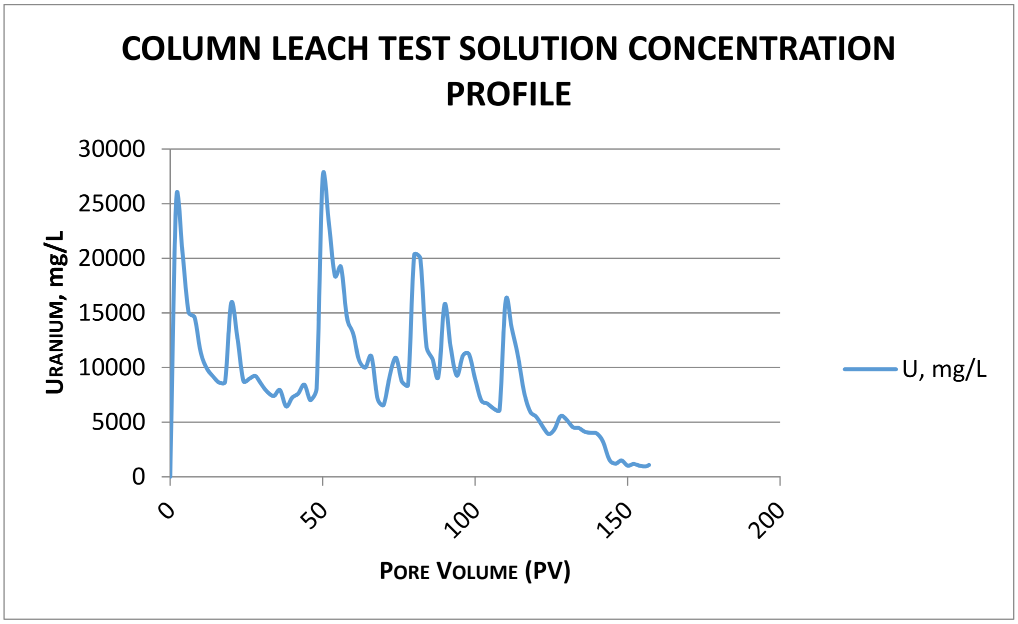


Figure 13-2. WR-525D1 Leach Recovery Grade

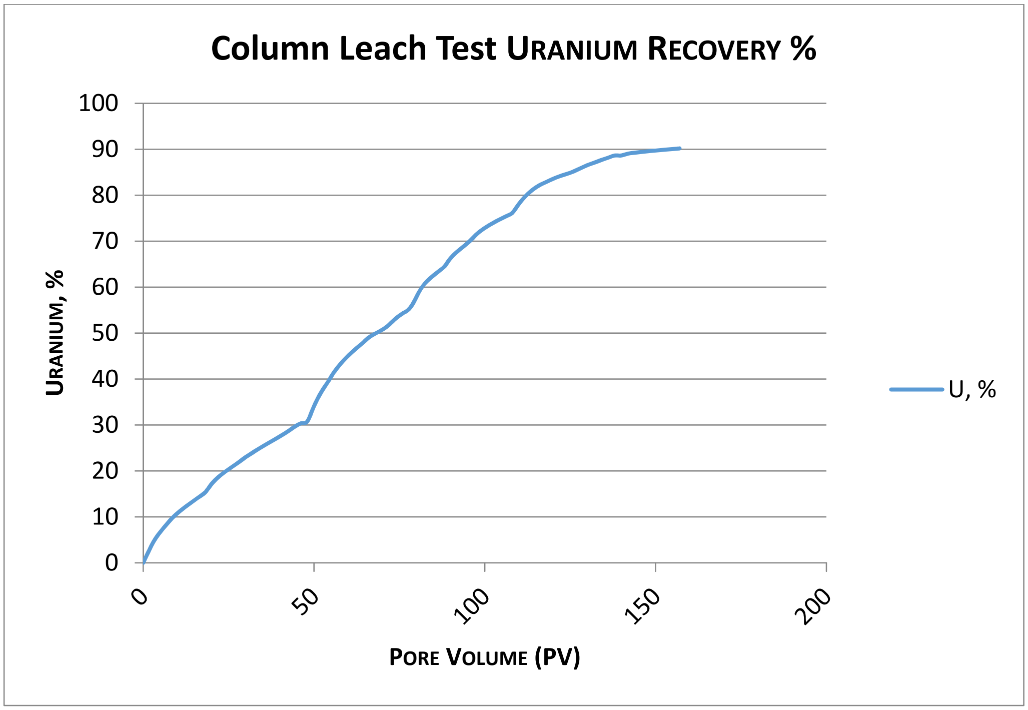


Figure 13-3. WR-525D1 Uranium Recovery Percent

13.1.6 Technical Discussion

Agitation and column leach tests conducted on the Phoenix samples from August 2017 to May 2018 successfully demonstrated that sulfuric/oxidant lixiviates can leach the uranium resource effectively under in situ mining conditions (see Tables 13.6 and 13.9). In both acidic leach tests performed, over 90% of available uranium was recovered, typically in 120-140 pore volumes.

Acid leaching of uranium deposits is described by the following reactions.

Sulfuric acid ionizes in solution to form sulfate, bisulfate, and hydrogen ions. Reaction with hexavalent uranium, which dissolves as the UO_2^{2+} cation, produces uranyl sulfate anionic complexes as follows:

$$UO_3 + 2H^+ = UO_2^{2+} + H_2O$$
$$UO_2^{2+} + 2SO_4^{2-} = [UO_2(SO_4)^2]^{2-}$$
$$[UO_2(SO_4)^2]^{2-} + SO_4^{2-} = [UO_2(SO_4)^3]^{4-}$$

It requires 2-3 moles of sulfuric acid (98 grams/mole) to solubilize 1 mole of uranium (238 grams/mole). Therefore, the sulfuric consumption to merely solubilize oxidized uranium is 0.41 kg H_2SO_4 per kg U.

Acid consumption during the column leach test was 0.53 kg H_2SO_4/kg U. Acid consumption is a function of the gangue constituents present in the deposit. Calcite, dolomite, magnesite, and siderite react readily with acids at low acid concentrations. Sulfides, phosphates, molybdates, vanadates, and oxides consume additional acid.

Typical reactions include:

$$CaCO_3 + H_2SO_4 = CaSO_4 + CO_{2(g)} + H_2O$$
$$(Ca,Mg)(CO_3)_2^{2+} + 2H_2SO_4 = CaSO_4 + MgSO_4 + 2H_2O + 2CO_{2(g)}$$
$$2PO_4^{3-} + 3H_2SO_4 = 2H_3PO_4 + 3SO_4^{2-}$$

Although some dissolution of complex aluminosilicates, such as hydrated clays or other acid soluble silicates, may also occur, the column leach tests did not indicate that significant reductions in these constituents were observed from heads to tails sample analysis (Table 13-11).

Table 13-11. Column Testing Dissolution Rates Results

Sample ID/ Parameter	Units	Head Concentration* (assayed/calculated)[1]	Final Residues (assayed)	Calculated Dissolution Rate** (%)[2]
Mass	kg	3.13	1.905	-
Al	mg/kg	29,100/29,600	43,800	9.9/10.1
As	mg/kg	382/139	60	73.6/26.8
Ba	mg/kg	97/81	133	0/16.9
B	mg/kg	1,800/86	64	54.7/2.6
Cd	mg/kg	24/863	0.8	84.0/10.8
Ca	mg/kg	7,290/3,620	648	89.1/44.3
Cr	mg/kg	227/23	25	33.7/3.4
Cu	mg/kg	6,640/2,620	2,100	47.3/17.3
Fe	mg/kg	41,600/38,520	45,300	28.4/26.3
Pb	mg/kg	16,800/3,830	5,970	5.2/1.2
Mg	mg/kg	3,620/2,350	1,560	59.6/38.8
Mn	mg/kg	489/252	22	94.8/48.9
Mo	mg/kg	332/74	30	75.5/16.7
Ni	mg/kg	183/96	35	77.9/41.1
P	mg/kg	469/188	108	65.1/26.2
K	mg/kg	130/697	1,080	5.7/30.7
Se	mg/kg	926/54	38	57.5/3.4
V	mg/kg	370/196	76	76.3/40.3
Zn	mg/kg	625/355	104	82.2/46.7
U	mg/kg	242,000/235,040	32,100	91.9/91.7
^{226}Radium	pCi/g	55,700 ± 25.9	91,000 ± 42.3	0.57
	Bq/g	2,061 ± 1.0	3,370 ± 2	0.57
^{230}Thorium	pCi/g	66,600 ± 1,590	30,300 ± 720	72.3
	Bq/g	2,464 ± 58.8	1,120 ± 27	72.3
^{232}Thorium	pCi/g	993 ± 197	31 ± 6	98.1
	Bq/g	36.7 ± 7.3	1.1 ± 0.2	98.1
^{210}Lead	pCi/g	20,000 ± 34.8	24,300 ± 42	26.1
	Bq/g	740 ± 1.3	900 ± 2	26.1
^{210}Polonium	pCi/g	170 ± 6.3	N/A	N/A
	Bq/g	6.3 ± 02		

Notes
1. *Calculated head concentration is based on solids residue content (mass X assays) + quantity extracted during leach test (volume X assays) divided by original head sample weight (3.13 kg).
2. **Dissolution rate is calculated by multiplying concentrations times mass $(t_0 - t_{final})/t_0$ X 100.

Tailings analyses were conducted following groundwater restoration/formation reclamation testing completion.

13.1.7 Precipitation of $UO_4 \cdot 2H_2O$ Directly from Recovered Pregnant Leach Solution

Uranium bearing solution recovered from active ISR wellfield extraction wells (or conventional mill primary extraction circuits) are typically introduced to either liquid/solid (ion exchange resin) or liquid/liquid (solvent extraction) ion exchange circuits to concentrate and purify the contained uranium. High uranium concentration eluates recovered from these processes may range from 10 to >50 g/L U and are usually delivered to precipitation circuits for uranium peroxide or diuranate generation.

Recovered solution from the agitation leach test already ranged from ~10 to ~50 g/L U and, as such, was considered potentially suitable for direct uranium peroxide precipitation. Tests were conducted on three batches of agitation leach recovery fluids to confirm that acceptable yellowcake products could be generated by direct precipitation from uranium bearing solutions. Tables 13-12 and 13-13 present information on the precipitation and product testing. In general, the precipitation testing indicated that by using ~0.5 g H_2O_2/g U on pH 2.0 pregnant lixiviant resulted in >96% uranium precipitation efficiency. In addition, the $UO_4.2H_2O$ product generated met converter specifications with respect to all regulated contaminants.

Table 13-12. Agitation Leach Test Direct U Precipitation Results

Test Solution	Units	PV 6-15	PV 41-50	PV 111-120
Uranium Conc.	mg/L	31,700	13,500	12,800
Volume	mL	200	200	200
Uranium Mass	mg/L	6,340	2,700	2,560
H_2O_2 Conc.	kg/kg U	0.54	0.69	0.57
H_2O_2 Addition pH		2.0	2.0	2.0
Final pH		1.75	1.75	1.75
Supernate [U]	mg/L	409	66.5	491
Precipitation Eff.	%	98.7	99.5	96.2

Table 13-13. Agitation Leach Test Solution Direct Precipitation Yellowcake Product Analysis

Sample ID	PV 6-15 Cake				Limits	
Sample Date	January 26, 2018				Standard (without penalty)	Max Limit (without reject)
	Dry @ 105C		Calcine @ 1000 F			
Units	mg/kg	% by wt.	mg/kg	% by wt.	%	%
Arsenic	115	0.01	<0.6	<0.01	0.01	0.04
Barium	<0.08	<0.01	<0.08	<0.01	0.01	0.04
Boron	<2	<0.005	319	<0.005	0.005	0.1
Cadmium	<0.2	<0.01	<0.2	<0.01	0.01	0.04
Calcium	1,740	0.17	1,930	0.19	0.05	1.0
Chromium	<0.2	<0.01	<0.2	<0.01	0.01	0.04
Iron	71	<0.15	2,040	0.2	0.15	1.0
Lead	465	0.047	752	0.075	N/A	N/A
Molybdenum	<0.6	<0.10	<0.6	<0.10	0.10	0.30
Phosphorus	57.3	<0.10	64.6	<0.10	0.10	0.70
Potassium	80	<0.20	170	<0.20	0.20	3.0
Selenium	<1.3	<0.01	<1.3	<0.01	0.01	0.04
Silicon	2,020	<0.50	7,520	<0.50	0.50	2.0
Silver	503	0.05	160	0.02	N/A	N/A
Sodium	3,620	<1.0	9,230	<1.0	1.0	7.5
Thorium	<100	<0.01	<100	<0.01	0.10	2.5
Titanium	136	0.01	256	0.03	0.01	0.05
Uranium	605,000	60.5	80,800	80.8	N/A	65% min
Vanadium	1,510	0.15	563	0.06	0.06	0.30
Zirconium	<1	<0.01	<100	<0.01	0.01	0.10

Based on the data generated on the agitation leach solutions relative to direct product precipitation from uranium bearing solutions, each leach batch (1 through 5) was composited (minus volumes lost to sampling and analysis) and a standard peroxide precipitation was conducted. Testing of solutions produced during the agitation leach process indicated that precipitation efficiencies exceeding 95% could be expected using 0.5 to 0.75 kg H_2O_2/kg U with final precipitation pH of 2.0-2.5. Table 13-14 depicts the performance of the uranium direct precipitation steps conducted during the column leach test.

Table 13-14. Column Leach Direct Precipitation Results and BLS Generation (Tests 1-4)

Precipitation Sequence	**#1**	**#2**	**#3**	**#4**
Date(s) Performed	**March 2-3, 2018**	**March 12-14, 2018**	**March 27-28, 2018**	**April 16-17, 2018**
Sample Container Nos.	1 to 9	10 to 24	25 to 39	40 to 54
Recovered Column Volume (mL)	9,333	14,863	15,700	15,783
Volume to H_2O_2 Precipitation	8,044	12,707	13,354	13,601
Post Precipitation Supernate Volume (mL)	6,632	10,975	12,162	12,032
Yellowcake Wash Volume (mL)	2,918	2,163	N/A	N/A
Volume Recovered for Barren Leach Solution Make-up (mL)	9,550	13,138	12,162	12,032
Volume Deionized Water Diluent (mL)	5,450	2,862	3,838	3,968
Barren Leach Solution (BLS) Volume (mL)	15,000	16,000	16,000	16,000
Dilution Percentage to Create BLS (%)	36.3	17.9	24.0	24.8
Solution Uranium Concentration (mg/L)	13,770	8,821	13,228	12,100
Precipitation Vessel Solution U Mass (g U)	111	112	177	165
Hydrogen Peroxide (H_2O_2) Mass (g)	30	45	180	192
Mass H_2O_2/Mass U Ratio	0.27	0.40	1.02	1.17
Final pH Following H_2O_2 and NaOH Addition	1.83	2.51	2.75	2.80
Supernate Soluble Uranium (mg/L U)	7,240	1,380	443	652
Uranium Peroxide [$UO_4.2H_2O$] Precipitation Efficiency (%)	**47.42**	**84.36**	**96.65**	**94.61**
BLS Feed Solution U Concentration After Dilution and Acid Adjustment (mg/L)	3,530	1,130	296	472

Interferences or contaminants present in the column solutions (but absent in the agitation leach solutions) prevented efficient precipitation of uranium peroxide in some trials. The initial uranium peroxide precipitation of the uranium solution recovered from the ore pre-conditioning phase was only 47% efficient resulting in reintroduction of a significant amount of soluble uranium to the column via injection of barren leach solution (BLS). By using increasing amounts of peroxide exceeding 1 kg H_2O_2/kg U and higher final pH ~2.75, efficiencies exceeding 80% were achieved for leach batches 2 through 4.

Precipitation test work on #5 solution was conducted to analyze the effects of altering peroxide addition rates and reaction times on uranium precipitation efficiency. Of twelve tests performed, the peak PPT efficiency was 71.7%. Table 13-15 examines variables and results of the expanded precipitation test work.

Table 13-15. Column Leach #5 Direct Precipitation Test Work Results

Test No.	**DW-1**	**DW-2**	**DW-3**	**DW-4**	**DW-5**	**DW-6**	**DW-7**	**DW-8**	**DW-9**	**DW-10**	**DW-11**	**DW-12**	**Mean**
Solution Volume (mL)	950	950	950	950	950	950	950	950	950	950	950	950	N/A
Solution Mass (g)	993	991	992	992	994	994	993	993	989	994	996	994	N/A
Solution Specific Gravity (g/cm^3)	1.045	1.043	1.044	1.044	1.046	1.046	1.045	1.045	1.041	1.046	1.048	1.046	1.045
Solution U Concentration (mg/L)	7,510	7,510	7,510	7,510	7,510	7,510	7,510	7,510	7,510	7,510	7,510	7,510	7,510
Solution U Mass (g)	7.13	7.13	7.13	7.13	7.13	7.13	7.13	7.13	7.13	7.13	7.13	7.13	85.6
H_2O_2 Addition (g/g U)	0.43	0.52	0.60	0.69	0.77	0.87	0.95	1.04	0.52	0.69	0.87	1.04	N/A
H_2O_2 Volume @ 30% (mL)	10.3	12.4	14.3	16.5	18.4	20.6	22.5	24.7	12.4	16.5	20.6	24.7	N/A
Test Start Time	13:37	13:54	14:00	14:20	14:55	15:10	15:23	15:35	16:18	16:27	16:37	16:47	N/A
pH Initial	1.36	1.39	1.38	1.37	1.38	1.41	1.4	1.41	1.48	1.46	1.46	1.46	N/A
pH Post H_2O_2	1.34	1.37	1.36	1.35	1.36	1.39	1.38	1.39	1.46	1.44	1.44	1.44	N/A
pH Final	2.77	2.74	2.74	2.74	2.60	2.61	2.60	2.62	2.77	2.82	2.87	2.88	N/A
Mass 10N NaOH Addition (g)	26.6	26.4	27.0	27.1	25.9	26.2	26.2	26.6	26.7	25.0	25.4	25.5	N/A
Reaction Duration (h)	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	6.0	6.0	6.0	6.0	N/A
Test Direct Precip. End Time	13:50	14:06	14:10	14:27	15:03	15:15	15:30	15:40	16:24	16:33	16:43	16:53	N/A
Time Coagulant Addition	14:40	14:56	15:00	15:17	15:53	16:05	16:20	16:30	N/A	N/A	N/A	N/A	N/A
Time Flocculant Addition	14:45	15:01	15:05	15:22	15:58	16:10	16:25	16:35	NA	NA	NA	NA	NA
Test Complete End Time	14:50	15:06	15:10	15:27	16:03	16:15	16:30	16:40	22:30	22:30	22:30	22:30	NA
Supernate U Conc. (mg/L)	3,340	3,140	2,580	2,370	2,730	2,050	2,330	2,300	5,170	6,110	4,920	3,870	3,408
Supernate U Volume (mL)	974	976	978	980	981	984	986	988	976	979	983	987	11,771
Supernate U Mass (g)	3.25	3.06	2.52	2.32	2.68	2.02	2.30	2.27	5.04	5.98	4.84	3.82	40.1
PPT Efficiency (%)	**54.42**	**57.06**	**64.64**	**67.44**	**62.45**	**71.73**	**67.81**	**68.15**	**29.29**	**16.16**	**32.19**	**46.44**	**53.15**

The supernatant from the individual direct precipitation tests (DW-1 through DW-12) were composited and tested in an attempt to recover residual uranium using further hydrogen peroxide additions. Although uranium precipitation efficiencies exceeding 90% were achieved at final pH of 6-7, the products generated were unacceptable with respect to iron and arsenic concentrations. The higher pH products were likely a mixture of uranium peroxide and sodium di-uranate. Table 13-16 indicates the precipitation efficiency of the process.

Table 13-16. Column Leach Test – Uranium Precipitation Test Results

Test I.D.	**S5.1-1**	**S5.1-2**	**S5.1-3**	**S5.1-4**	**S5.1-5**	**S5.1-6**	**S5.1-7**	**S5.1-8**	**S5.1-9**	**S5.1-10**
Supernate #5 Volume (mL)	10,850	10,850	10,850	10,850	10,850	10,850	10,850	10,850	10,850	10,850
Supernate #5 U Conc. (mg/L)	3,408	3,408	3,408	3,408	3,408	3,408	3,408	3,408	3,408	3,408
Supernate #5 U Mass (g)	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0
pH Initial	2.77	2.61	3.02	3.55	4.21	4.51	5.00	5.60	6.00	6.60
pH Post H_2O_2 Addition	2.44	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
pH Final	2.61	3.02	3.55	4.21	4.51	5.00	5.60	6.00	6.60	7.00
Supernate 5.1 U Conc. (mg/L)	2,210	1,320	1,320	1,750	1,720	1,570	789	316	39.2	6.9
Supernate 5.1 U Mass (g)	24.0	14.3	14.3	19.0	18.7	17.0	8.6	3.4	0.4	0.1
PPT Efficiency (%)	**35.15**	**61.27**	**61.27**	**48.65**	**49.53**	**53.93**	**76.85**	**90.73**	**98.85**	**99.80**

An observation noted during the direct uranium peroxide precipitation processes was that immediate co-precipitation of reddish solids accompanied by off gassing occurred following hydrogen peroxide addition and subsequent NaOH based pH adjustments (to achieve final pH >2.0). It was also observed that the concentrations of certain metallic species were constantly increasing in both initial uranium bearing solutions and BLS solutions, notably aluminum and iron. Postulating that iron and other metals were potentially interfering with and impeding the uranium precipitation process, tests to remove these contaminants prior to uranium recovery were attempted. It appeared that simple pH adjustment of the solution prior to hydrogen peroxide addition could co-precipitate certain metal hydroxides, which could then be removed from solution via gravity settling, filtration, or centrifugation.

Details of the resulting test work is shown in Tables 13-17 and 13-18. By adjusting solution pH to 3.5 to 4.25, iron was reduced from ~1,400 mg/L to <100 mg/L. Arsenic was completely removed from the solution precipitation feed solution above pH 3.5. Uranium precipitation efficiency increased to >99% at all tested post iron removal pHs using 0.3 to 1.0 kg H_2O_2/kg U. Uranium peroxide products generated from two tests (pH 2.75 and pH 3.5) indicated that all converter specifications were met, typically without penalty. A product analysis summary for all tests conducted is shown in Table 13-19.

A second test of the metal hydroxide removal process was performed on June 19, 2018 to ascertain the mass and anticipated volume and chemical composition (including radionuclides) of the generated "sludge". Table 13-18 presents the results of that testing. The metal hydroxide precipitate generated in the follow-up test was separated easily by centrifugation. Solids were very dense and represented 1.5% mass/mass fluid wet and ~0.15% on a dry weight basis. Full analysis of the precipitated solids is ongoing (as noted in the above table).

Uranium products generated through the direct precipitation method (including the initial metal hydroxide precipitation process) and low temperature drying (400°F) are presented in Table 13-19.

Table 13-17. Uranium Precipitation Process Testing

Sample ID	Metal Hydroxide (Me-OH) Precipitation (PPT) Test Work							$UO_4 \cdot 2H_2O$ Precipitation Supernate Analysis					
	FEED	pH 2.75	pH 3.0	pH 3.25	pH 3.5	pH 4.0	pH 4.25	S-1	S-2	S-3	S-4	S-5	S-6
pH @ H_2O_2	N/A	N/A	N/A	N/A	N/A	N/A	N/A	2.72	4.10	4.02	4.01	4.09	4.36
pH Post H_2O_2	N/A	N/A	N/A	N/A	N/A	N/A	N/A	2.05	2.15	2.13	2.1	2.04	2.02
H_2O_2 Addition	N/A	N/A	N/A	N/A	N/A	N/A	N/A	1.0	0.3	0.5	0.7	0.9	1.1
Adj pH	N/A	N/A	N/A	N/A	N/A	N/A	N/A	2.76	3.13	2.95	3.11	3.04	3.43
Al, mg/L	214	247	262	219	197	186	167	183	130	133	152	156	115
As, mg/L	7.47	8.66	8.58	1.57	<0.01	<0.01	<0.01	6.96	<0.01	<0.01	<0.01	<0.01	<0.01
B, mg/L	4.2	4.9	5.3	4.0	4.0	6.5	4.8	2.0	1.3	4.2	1.7	2.2	1.5
Cd, mg/L	0.203	0.219	0.231	0.177	0.169	0.171	0.179	0.137	0.094	0.093	0.111	0.121	0.104
Ca, mg/L	368	408	423	392	378	391	406	338	309	300	336	361	336
Cr, mg/L	1.20	1.39	1.56	0.88	0.41	0.28	0.28	1.15	0.23	0.22	0.21	0.26	0.14
Cu, mg/L	92.8	104	107	96.5	92.2	92.7	92.5	81.4	70.0	69.8	77.9	82.1	73.2
Fe, mg/L	1,390	1,620	1,680	444	55.7	22.0	13.2	1,210	15.5	9.82	10.0	17.2	6.30
Pb, mg/L	12.3	13.4	14.1	12.3	10.9	10.4	10.1	3.69	1.57	2.11	1.96	1.66	0.68
Mg, mg/L	71.7	79.6	82.8	77.0	74.4	75.3	77.5	37.9	35.0	34.0	37.9	40.8	38.1
Mn, mg/L	42.2	49.6	52.5	43.6	39.7	42.4	44.2	36.2	28.4	27.5	32.7	35.0	30.3
Mo, mg/L	7.54	8.31	8.74	2.15	1.36	2.65	3.07	7.65	1.45	1.24	1.56	2.74	2.90
Ni, mg/L	8.86	9.76	10.2	9.24	8.95	9.28	9.52	8.18	7.31	7.17	7.98	8.61	7.93
P, mg/L	12.8	13.9	13.8	2.8	<0.1	<0.1	<0.1	12.8	0.2	0.3	<0.1	<0.1	0.3
K, mg/L	11.9	16.3	16.9	13.8	13.3	15.4	15.3	9.6	8.4	8.8	9.6	10.6	9.3
Na, mg/L	11,500	18,400	20,100	18,400	17,700	19,200	20,900	15,600	14,700	13,800	16,600	18,300	16,200
Sr, mg/L	0.97	1.01	1.10	1.01	0.97	0.99	1.06	0.84	0.77	0.75	0.85	0.91	0.85
V, mg/L	17.6	19.2	20.1	17.7	17.7	18.2	18.8	14.4	11.0	12.4	15.9	17.0	17.1
Zn, mg/L	32.6	37.8	40.3	34.9	32.5	35.0	37.1	28.7	24.8	23.4	28.1	31.1	26.9
SiO_2, mg/L	189	208	217	175	172	164	158	176	133	137	150	149	129
SO_4, mg/L	33,300	37,200	39,900	35,100	33,300	35,100	37,500	35,700	30,900	29,190	34,500	38,100	32,700
U, mg/L	14,200	16,900	17,700	15,300	14,200	15,300	15,800	33.8	22.8	22.3	24.7	30.9	80.4
Avg solution U	-	-	-	-	-	-	**15,629**	-	-	-	-	-	-
% U PPT EFF	-	-	-	-	-	-	-	**99.78**	**99.85**	**99.86**	**99.84**	**99.80**	**99.49**

Table 13-18. Metal Concentration Reduction in Solution Prior to Direct U PPT Testing

Sample ID/ Parameter	UNITS	Fe PPT FEED (CALC)	Fe PPT FEED COMP	Fe PPT SUPERNATE	TEST 6 COMP	TEST 6 SUPERNATE COMP	TEST 6 CAKE WASH	TEST 6 Fe CAKE SOLIDS (mg/kg)
pH	-	2.225	1.96	4.02	2.19	3.88	4.22	N/A
Elec Cond	umho/cm	35,138	36,200	33,900	N/A	41,700	1,738	N/A
ORP	mV	+543	+532	+366	N/A	+560	+400	N/A
Free Acid	g/L	4.21	N/A	N/A	N/A	N/A	N/A	N/A
Al	mg/L	222	183	149	214	124	1.25	15,200
As	mg/L	2.03	1.01	0.06	7.47	<0.01	0.33	757
Ba	mg/L	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	5.9
B	mg/L	1.4	1.9	1.8	4.2	2.4	0.9	35.4
Cd	mg/L	0.249	0.184	0.157	0.203	0.093	<0.001	8.9
Ca	mg/L	381	334	328	368	313	3.4	1,000
Cr	mg/L	0.35	0.4	0.07	1.20	0.13	0.01	128
Cl	mg/L	N/A	964	985	N/A	902	22.0	N/A
Cu	mg/L	71.8	63.1	59.3	92.8	77.9	1.46	1,110
Fe	mg/L	728	632	11.1	1,390	2.68	<0.05	435,000
Pb	mg/L	7.25	6.27	4.66	12.3	0.63	0.04	709
Mg	mg/L	102	79.2	77.4	71.7	45.2	<0.2	394
Mn	mg/L	14.4	12.6	12.4	42.2	20.9	0.4	42.3
Mo	mg/L	1.30	1.65	0.588	7.54	1.09	0.371	413
Ni	mg/L	4.6	4.09	3.99	8.86	7.62	0.16	39.2
P	mg/L	4.2	0.6	<0.1	12.8	0.7	0.3	1,140
K	mg/L	8.2	2.5	2.7	11.9	7.0	<0.2	130
Se	mg/L	0.047	<0.025	<0.025	<0.025	<0.025	<0.025	40.3
Na	mg/L	9,948	8,960	10,900	11,500	13,400	223	22,900
Sr	mg/L	0.55	0.46	0.46	0.97	0.86	<0.02	7.7
V	mg/L	4.0	4.04	1.72	17.6	8.47	0.077	897
Zn	mg/L	28.8	25.4	25.3	32.6	16.0	0.02	91.9
SiO_2	mg/L	136	124	110	189	143	3.7	3,410
SO_4	mg/L	24,368	22,020	21,450	33,300	21,030	468	144,900
U	mg/L	3,515	2,770	2,680	14,200	39.7	10.3	22,200
^{226}Radium	pCi/L	N/A	3,400 ± 14.2	6,030 ± 18.9	9,230 ± 72	4,580 ± 16	N/A	1,280 ± 5.6
	Bq/L	N/A	126 ± 0.5	223 ± 0.7	342 ± 2.7	169 ± 0.6	N/A	47.4 ± 0.2

Sample ID/ Parameter	UNITS	Fe PPT FEED (CALC)	Fe PPT FEED COMP	Fe PPT SUPERNATE	TEST 6 COMP	TEST 6 SUPERNATE COMP	TEST 6 CAKE WASH	TEST 6 Fe CAKE SOLIDS (mg/kg)
^{228}Radium	pCi/L	N/A	700,000 ± 28,700	93,100 ± 6,300	4,870,000 ± 236,000	86,300 ± 7,040	N/A	20,600 ± 1,120
	Bq/L	N/A	25,925 ± 1,063	3,448 ± 233	180,370 ± 8,740	3,196 ± 261	N/A	763 ± 41.5
^{230}Thorium	pCi/L	N/A	847 ± 28.9	168 ± 12.8	5,430 ± 226	52.8 ± 7.0	N/A	963 ± 13
	Bq/L	N/A	31.4 ± 1.0	6.2 ± 0.5	201 ± 8.4	2.0 ± 0.3	N/A	35.7 ± 0.5
^{232}Thorium	pCi/L	N/A	20,000 ± 51.0	13,100 ± 41	31,600 ± 205	4,320 ± 23.9	N/A	2.7 ± 0.5
	Bq/L	N/A	741 ± 1.9	485 ± 1.5	1,170 ± 7.6	160 ± 0.9	N/A	0.1 ± 0.02
^{210}Polonium	pCi/L	N/A	3,400 ± 14.2	6,030 ± 18.9	9,230 ± 72	4,580 ± 16	N/A	N/A
	Bq/L	N/A	126 ± 0.5	223 ± 0.7	342 ± 2.7	169 ± 0.6	N/A	N/A
^{210}Lead	pCi/L	N/A	700,000 ± 28,700	93,100 ± 6,300	4,870,000 ± 236,000	86,300 ± 7,040	N/A	N/A
	Bq/L	N/A	25,925 ± 1,063	3,448 ± 233	180,370 ± 8,740	3,196 ± 261	N/A	N/A

Table 13-19. Product Analysis Summary

SAMPLE ID	DWP 1	DWP 2	DWP 3	DWP 4	DWP 5	DWP 6	PPT S-1	PPT S-3	PPT 2,4,5,6 COMPOSITE	CONVERTER SPECIFICATIONS	
										LIMITS w/o PENALTY	MAX LIMIT w/o REJECTION
PARAMETER/ UNITS	%	%	%	%	%	%	%	%	%	%	%
Arsenic	<0.01	<0.01	<0.01	0.01	0.02	0.12	<0.01	<0.01	<0.01	0.01	0.04
Barium	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.01	0.04
Boron	0.017	0.013	0.011	0.016	0.014	0.009	0.019	0.017	0.015	0.005	0.1
Cadmium	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.01	0.04
Calcium	0.23	0.28	0.27	0.27	0.23	0.69	0.17	0.26	0.68	0.05	1.0
Chromium	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.01	0.04
Iron	<0.15	0.51	0.69	0.79	1.97	11.2	0.67	<0.15	<0.15	0.15	1.0
Lead	<0.01	<0.01	<0.01	<0.01	0.02	<0.01	<0.01	<0.01	N/A	N/A	N/A
Manganese	<0.02	<0.02	<0.02	<0.02	<0.02	0.24	<0.02	<0.02	<0.02	0.02	0.50
Molybdenum	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	0.1	0.30
Phosphorus	<0.1	<0.1	<0.1	<0.1	<0.1	0.1	<0.1	<0.1	<0.1	0.1	0.70
Potassium	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	0.2	3.0
Selenium	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.01	0.04
Silicon	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	0.5	2.0
Silver	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	N/A	N/A	N/A
Sodium	<1	5.0	1.1	1.6	<1	7.8	<1	<1	<1	1.0	7.5
^{232}Thorium	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	0.10	2.5
Titanium	0.02	0.02	0.03	0.03	0.03	0.03	0.03	0.01	0.02	0.01	0.05
Uranium	76.3	73.4	80.3	68.5	69.7	29.3	79.0	78.9	80.0	N/A	65% min
Vanadium	<0.06	<0.06	<0.06	<0.06	0.07	0.11	<0.06	<0.06	<0.06	0.06	0.30
Zirconium	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	<0.01	0.01	0.50

13.1.8 ISR Groundwater Restoration Results and Discussion

Following termination of the leach test at PV 156, low pH leached core was retained in the test column in preparation for groundwater restoration testing. The testing protocol was developed to approximate actual field conditions. Groundwater restoration flow was initiated at RPV 1 on June 4, 2018 and continued through RPV 108 ending on July 23, 2018. The test results indicate that, following neutralization of excess acid remaining in the ore, restoration to near ore zone pre-mining baseline conditions could be achieved. Testing included groundwater sweeps, followed by sodium bicarbonate neutralization in conjunction with simulated reverse osmosis lixiviant circulation. A groundwater stability period (column soak without circulation) was incorporated to ensure that long-term restoration to acceptable water quality goals would be maintained. Complete testing results are reported in R&D Enterprises report "Denison Wheeler River Project Groundwater Chemical Restoration Following Acidic Insitu Uranium Recovery Operations", dated September 9, 2018.

Details of the groundwater restoration processes followed during the test are depicted below.

Phase 1 (RPV 1-84) injected simulated upper formation groundwater quality at the bottom of the column. The initial phase of restoration displaces the low pH, high TDS fluids with simulated groundwater. The fluid recovered at the top of the column is collected and sent for analysis. Table 13-20 presents a summary of the groundwater quality exiting the column following initiation of groundwater sweeps.

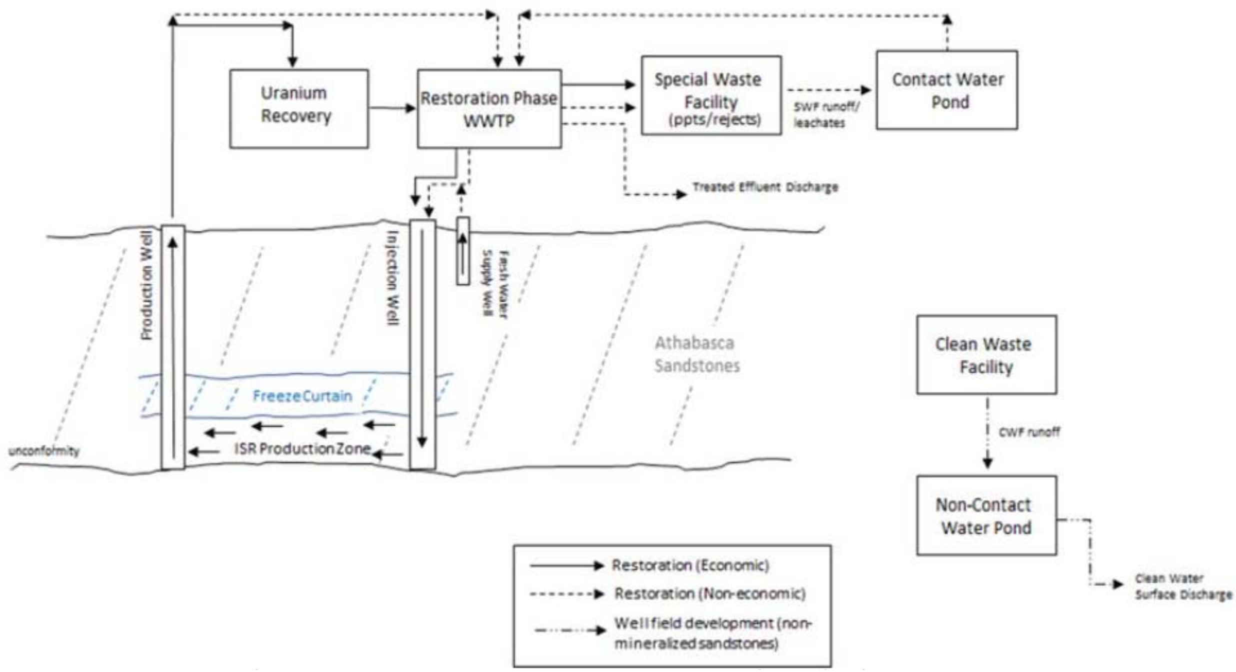
Table 13-20. RPV 1-10 Chemistry Highlight

	pH	Elec Cond	Al	As	Fe	Mo	Ni	Na	Sr	S	SO_4 CALC	U	U	Vol Rec
PV	**Leachate**	**umho /cm**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg**	**ml**
RPV 1-2	2.08	40,400	315	0.61	939	1.85	5.72	12,300	0.62	10,400	31,200	4,860	5,030.1	1,035
RPV 3-4	2.56	17,410	200	0.02	797	0.204	3.47	5,320	0.61	4,810	14,430	4,080	4,131.4	1,012.6
RPV 5-6	2.71	11,740	94.8	<0.01	422	0.107	1.82	2,470	0.44	2,380	7,140	1,960	1,975.7	1,008.0
RPV 7-8	2.87	7,510	50.3	<0.01	255	<0.005	1	1,330	0.42	1,390	4,170	1,110	1,108.7	998.8
RPV 9-10	2.94	5,940	33.9	<0.01	185	<0.005	0.83	947	0.41	1,040	3,120	762	765.1	1,004.1
													13,011	**5,058.5**
														U (mg/L)
														2,572

Source: RDE– Groundwater Chemical Restoration Following Acidic In-Situ Uranium Recovery Operations dated September 9, 2018

The first Restoration Pore Volumes (RPV) chemistry is mostly residual lixiviant from the active mining leaching test. Similar to the leach testing conditions, each pore volume (PV) during restoration activities represents 500 mL of solution. In total, 13,011 mg of uranium was recovered representing a uranium average of 2.57 g/L. In an industrial setting, solution exiting the ground resulting from groundwater sweeps would be directed to the ISR recovery plant prior to treatment and recirculation to wellfield. Immediate forced neutralization with basic solutions was avoided to prevent formation plugging with calcium sulfate or other co-precipitates.

Phase 2 (RPV 84-108) circulated simulated ore zone0 water quality fortified with 1 g/L Bicarbonate [from $NaHCO_3$]. The test simulated the operation of a Reverse Osmosis (RO) water treatment step where solution exiting the column would be treated prior to being re-introduced. The closed loop circulation system reduces the need for pumping a large volume of groundwater and avoids generating effluent that would otherwise be disposed of in the environment. Reverse osmosis is expected to provide an 80% permeate, 20% brine split on recovered solutions. The generated “permeate” was adjusted to ~1 g/L bicarbonate using $NaHCO_3$. The use of sodium bicarbonate allows for neutralization of residual acid, forces ion exchange of hydrogen off clays, and reduces the potential for column plugging with precipitates. Phase 2 was terminated when uranium and trace metal concentrations were approaching pre-mining levels. It is anticipated to evaporate the 20% brine split in a crystallizer. The salts would be stored in tote bags and disposed of in the Gryphon mine workings while the distilled water re-injected supporting ground remediation efforts. Figure 13-4 presents an overview of the anticipated ground water restoration process. Solution evaporation system (crystallizer) and RO being integral components of the “Restoration Phase” Waste Water Treatment Plant (WWTP).



Source: SRK – Wheeler River Project Phoenix In-Situ Recovery Wastewater Treatment Design Criteria dated August 2018

Figure 13-4. Restoration Phase WWTP Schematic

Phase 3 (RPV 108-114) re-established injection of simulated groundwater quality at the bottom of the column following the circulation of an alkaline base neutralizing agent. The objective of this phase is to displace the bicarbonate and to ensure ground water stability once the circulation of fluid is halted. Termination of “restoration stability” is determined by the return of the water quality to near pre-mining conditions as guided by ground water samples from WR-525-352-395 [3/29/17]. Table 13-21 presents results of the acidic ISR column leach restoration and groundwater stability studies.

Table 13-21. Acidic ISR Column Leach Restoration and Groundwater Stability Study Results

Phoenix Deposit Ore Zone Water Quality				Leach Data	Restoration Phase Data			
Parameter	Reporting Units	Mean	Max	H_2SO_4 Leach End of Mining	Groundwater Sweep		Neutralization	Groundwater Stability
				PV 154-156	RPV 30-32	RPV 76-84	RPV 82-108	Post Rest.
Inorganics								
Alkalinity (as $CaCO_3$)	mg/l	32.7	78.1	<5	<5	<5	588	165
Calcium	mg/l	11.2	28	542	109	28	13	5
Magnesium	mg/l	6.1	18	87.4	3.7	<1	<1	<1
Potassium	mg/l	6.2	14	11.7	<1	<1	<1	2
Sodium	mg/l	14.9	38	22500	221	36	235	87
Bicarbonate	mg/l	70.7	136	<5	<5	<5	686	199
Chloride	mg/l	29.4	142	N/A	37	15	2	12
Fluoride	mg/l	0.37	0.69	N/A	N/A	0.5	1.2	0.8
Sulfate	mg/l	5.9	26.2	60600	860	174	117	100
Metals								
Aluminum	mg/l	0.4	1.2	92.2	5.6	0.27	1.32	4.4
Arsenic	mg/l	0.422	2.3	2.23	<0.010	0.10	0.04	0.06
Barium	mg/l	0.09	0.15	<0.5	<0.05	<0.05	0.05	0.04
Boron	mg/l	0.47	0.75	2.7	1.8	0.2	0.3	<0.1
Cadmium	mg/l	0.004	0.016	0.125	<0.001	0.002	<0.001	<0.001
Chromium	mg/l	N/A	N/A	0.22	0.04	<0.01	<0.01	<0.01
Copper	mg/l	0.01	0.04	39.4	2.23	0.04	<0.01	<0.01
Iron	mg/l	1.1	2.2	279	54.1	6.13	0.44	1.23
Lead	mg/l	0.02	0.10	7.84	3.08	2.13	0.36	0.39
Manganese	mg/l	0.23	0.77	17.0	0.68	0.07	0.02	0.05
Molybdenum	mg/l	0.07	0.36	1.85	0.05	0.03	0.05	<0.005
Nickel	mg/l	0.04	0.18	7.06	0.20	0.03	<0.01	<0.01
Selenium	mg/l	0.004	0.01	0.586	<0.025	<0.025	<0.025	0.026
Uranium	mg/l	0.006	0.01	1080	105	3.5	4.1	0.50
Vanadium	mg/l	N/A	N/A	1.7	0.09	<0.01	0.007	0.03
Zinc	mg/l	0.16	0.39	13.6	1.48	0.08	<0.01	<0.01
Other								
EC	µmhos/cm	154	268	77800	1874	472	1330	613
pH	s.u.	6.48	7.32	1.76	3.87	5.8	8.5	8.3
Silica as SiO_2	mg/l	15.2	19.9	139	71.9	43.7	43.8	44.4
TDS	mg/l	186	540	N/A	N/A	360	760	350
Radiological								
Lead-210	Bq/L	5.2	17	3167+/-9.8	530+/-1.3	301+/-0.7	40+/-0.3	22+/-0.2
Polonium-210	Bq/L	2.1	7.0	43+/-2.6	6.3+/-0.5	14.9+/-0.3	1.9+/-0.1	2.7+/-0.1
Thorium-230	Bq/L	0.6	1.3	14960+/-3310	105+/-9.6	0.2+/-0.03	1.36+/-0.14	3.2+/-0.4
Thorium-232	Bq/L	N/A	N/A	<1	0.2+/-0.04	<0.01	<0.01	<0.01
Radium-226	Bq/L	1.8	4.3	345+/-1.4	65.8+/-0.3	389+/-0.7	262+/-0.5	129+/-0.4

Source: SRK – Wheeler River Project Phoenix In-Situ Recovery Wastewater Treatment Design Criteria dated August 2018

The laboratory groundwater restoration study supports the conclusion that the groundwater can be restored to a pre-mining use category following acidic ISR mining. The results of the groundwater restoration studies using core samples previously leached with acid lixiviates were compared with the mean groundwater baseline for the Denison Wheeler River Phoenix deposit. Greater than 60% of the tested parameters achieved target restoration values [TRV] where ~40% exceeded the maximum concentration limited as reported for the Phoenix ore zone water quality. Major species exceeding the baseline maximum limit includes only those that were injected in leach zone during ISR mining or restoration activities [Alkalinity-HCO_3, SO_4, and Na]. pH and Electric Conductivity (EC) remained elevated over baseline values due to the final neutralization phase occurring immediately before the stability period. Al, Pb and U also were elevated above the baseline maximum values. Results obtained during the test work are not inconsistent with those observed on pilot and commercial scale alkaline ISR groundwater restoration operations.

13.1.9 ISR Process Design Criteria

Based on the results from the column leach test, key process design criteria (PDC) were defined for the processing of Phoenix ore. These are presented in Table 13-22. PDC developed from test work is primarily associated with leaching the ore in-situ as simulated during the test program. The laboratory test work discussed previously resulted in solution concentrations averaging approximately 18 g/L in the agitation (bottle roll) test and approximately 12 g/L in the column leach test. As a column leach test most effectively simulates in situ conditions, but does not re-create them perfectly, the average concentration from the column leach test (approximately 12 g/L) was discounted, and the costs associated with the ISR operation were developed based on a solution concentration of 10 g/L. Other assumptions were made for estimating hydrogen peroxide consumption for replacing the barium chloride as oxidant as tested. In part due to the simplicity of the uranium recovery process and considering the inclusion of equipment redundancy, it is assumed the plant operating 360 days per year. It was determined early on ferric sulphate addition is not required considering the high natural content of iron minerals in the Phoenix deposit. The expected performance of the ISR plant includes measured losses incurred during the iron/radium removal and the high uranium precipitation efficiency. As such, the expected overall plant uranium recovery rate is set at 98.5%.

Table 13-22. Phoenix PDC Values

Parameter	Units	Value
Uranium Production	U_3O_8 Mlbs/y	6
No. Plant Operating Days	Days/y	360
Wellfield Pregnant Solution Head Grade	g U_3O_8/L	10
Wellfield Barren Strip Solution Sulfuric Acid Concentration	g/L	15
Wellfield Barren Strip Solution Nominal Feed Rate	m^3/h	27.1
Wellfield Barren Strip Solution Design Feed Rate	m^3/h	45.9
Wellfield Pregnant Solution Sulfuric Acid Concentration	g/L	5
Wellfield Barren Strip Solution Hydrogen Peroxide Concentration	g/L	0.5
Wellfield Barren Strip Solution Ferric Sulphate Concentration	g/L	Not Required
Overall ISR Plant Uranium Recovery	%	98.5

Source: W & C – Process Calculations and Equipment Sizing Wheeler River Uranium ISR Project Revision G dated August 30, 2018

13.2 Gryphon Deposit Metallurgical Testing

This section summarizes the metallurgical test work that was completed on the Gryphon deposit, including the laboratory testing methods, results, and analysis. The test work program has considered the current operating conditions used at the McClean Lake mill, with the goal of utilizing similar operating conditions where possible.

In 2017, Denison undertook a metallurgical test work program at the Saskatchewan Research Council (SRC). The program was directly managed by Denison. Denison also completed a parallel test program at the Orano Service d'Études de Procédés et Analyses (SEPA) labs at L'Etablissement de Bessines located at 2, route de Lavaugrasse, CS30071, 87250 Bessines-sur-Gartempe, France. The analysis laboratory is ISO 17025 certified and its operation approved by the ministry. The objectives of the test work programs were to further develop the optimum processing conditions for each deposit and collect additional data to support engineering design.

SRC's Geoanalytical Laboratories (located at 125-15 Innovation Boulevard, Saskatoon, Saskatchewan, Canada) are recognized as Accredited Testing Laboratories by the Standards Council of Canada (SCC) under ISO 17025:2005. SRC is certified under ISO 9001:2008 for Quality Management Systems (QMS).

Hatch reviewed test work completed to date in support of Gryphon processing for the project. Hatch did not directly manage or supervise sample selection and collection, or the metallurgical test work program.

13.2.1 Sample Selection

Denison completed a statistical analysis of the Gryphon deposit based on available information prior to 2017. Based on this data, Denison established an initial variability assessment of key elemental grade ranges within the deposit, as presented in Table 13-23. Sample selection and compositing for the project was completed by Denison and summarized in a project report entitled "Gryphon Deposit Metallurgical Sample Selection", dated July 2017.

Table 13-23. Gryphon Deposit Grade Variability

	Min	Avg.	Max	90% Conf. Level
U_3O_8 (%)	0.200	2.37	42.5	2.9
Arsenic (%)	Trace	0.002	0.2	0.003
Molybdenum (%)	Trace	0.052	2	0.135
Al_2O_3 (%)	2.32	23.92	40	34.2
Fe_2O_3 (%)	0.09	1.09	19.3	2.11
MgO (%)	0.14	4.82	16.7	7.7
CaO (%)	0.03	0.36	11.5	0.38
Cadmium (%)	Trace	Trace	0.001	Trace
Selenium (%)	Trace	0.001	0.073	0.003

Source: Denison - "Gryphon Deposit Metallurgical Sample Selection Methodology"

Gryphon ore material is expected to be very low in arsenic and low in molybdenum, which are common impurity elements in uranium processing. Additionally, the main gangue associated with the mineralization is clay, which is important to assess as part of the metallurgical test program as it can influence acid consumption and uranium recovery as well as settling rheological performance.

Based on the target sample grades, Denison selected samples for each of the Gryphon composites from assay "reject" pulp samples that were previously crushed to – 6 mesh.

The samples were sent to SRC in Saskatoon. Denison provided guidance to SRC on the procedure for the compositing, blending, and assaying. Three Gryphon composites were prepared as summarized in Table 13-24. QEMSCAN mineralogy analysis was completed on each of the composites as summarized in Table 13-25.

The samples tested are considered to be reasonably representative of the commercial product for a PFS based on the deposit variability data provided by Denison. Additional variability testing on the extremes the deposit and by lithological groups is recommended to be completed during the feasibility study.

Table 13-24. SRC – Gryphon Composite Sample Assay Results

Sample		Low Grade	Medium Grade	High Grade
U_3O_8	%	1.58	3.14	6.43
Al_2O_3		23.30	23.50	24.30
Fe_2O_3		1.11	1.25	1.16
CaO		0.43	0.32	0.69
MgO		3.39	3.50	3.60
K_2O		3.00	2.64	3.02
TiO_2		0.82	0.68	0.77
Mo	ppm	778	1,010	1,115
As		29	39	57
Ni		81	94	112
Pb		1,280	2,730	5,495
Co		41	43	51
Cu		414	312	455
V		1,060	1,080	1,125
Zn		12	5	16

Source: Denison - "Denison Uranium Ore Metallurgical Testing Part Two: Gryphon Acid Leaching and Settling"

Table 13-25. SRC – Gryphon Mineralogy

Mineral	Ideal Formula	Gryphon Low Grade	Gryphon Medium Grade	Gryphon High Grade
Pyrite	FeS_2	0.31	0.67	0.33
Pyrrhotite	$Fe_{1-x}S$ (x = 0 to 0.17)	0.00	0.00	0.00
Galena	PbS	0.25	0.41	0.51
Sphalerite	(Zn,Fe)S	0.00	0.00	0.00
Chalcopyrite	$CuFeS_2$	0.16	0.14	0.12
Quartz	SiO_2	20.03	20.59	16.16
Feldspar	$KAlSi_3O_8$	0.35	0.41	0.65
Muscovite/Illite	$KAl_2(Si_3Al)O_{10}(OH; F)_2$	44.01	35.59	37.55
Chlorite	$(Fe,Mg)_2(Al; Fe^{3+})_3Si_3AlO_{10}(OH)_8$	16.73	20.57	20.40
Titanite	$CaTiSiO_5$	0.00	0.00	0.00
Zircon	$ZrSiO_4$	0.06	0.05	0.07
'Kaolinite' (clay)	$Al_2Si_2O_5(OH)_4$	12.85	15.91	14.16

Mineral	Ideal Formula	Gryphon Low Grade	Gryphon Medium Grade	Gryphon High Grade
Calcite	$CaCO_3$	0.17	0.07	0.38
Apatite	$Ca_5(PO_4)_3(F,Cl,OH)$	0.01	0.01	0.04
Rutile/Anatase	TiO_2	2.06	1.54	1.51
Fe-oxy-hydroxides	$FeO(OH)\cdot nH_2O$	0.65	0.92	0.96
Corundum	Cr_2O_3	0.00	0.02	0.10
APS Minerals	$CaAl_3(PO_4)(PO_3OH)(OH)_6$	0.10	0.13	0.13
Uraninite	UO_2	2.25	2.97	6.94
Gersdorffite	NiAsS	0.00	0.00	0.00
Nickeline	NiAs	0.00	0.00	0.00

Source: SRC - Report 13706-SC17 "Phoenix and Gryphon Uranium Ore Metallurgical Testing Part Three: QEMSCAN Mineralogy Analysis"

13.2.2 Metallurgical Testing

Denison has completed a number of metallurgical tests on the Gryphon deposit (Table 13-26). Test work has primarily been completed at the SRC lab in Saskatoon, Saskatchewan, with additional test work completed by SEPA in France.

In the most recent test work, the low-grade Gryphon sample was considered the primary sample as it most closely aligned with the life of mine average operating feed composition. All composite test results were considered as part of the metallurgical review, with the variability noted providing guidance on “design” maximum reagent requirements that could occur during the life of the project.

Table 13-26. Gryphon Metallurgical Test Work Summary

Test Type	Program	Tests	Key Variables
Comminution	SRC 2017	1	Bond Work Index
Leaching	SRC 2014	5	Initial scoping tests - oxidant, free acid, grind size, residence time
	SRC 2017	36	Composite variability tests - grind size, free acid, residence time on 3 composites
	SEPA 2017	6	Confirmatory leach testing and hydrogen evaluation. Two leach tests per composite with a focus on hydrogen evolution
Settling	SRC 2017	36	Settling rate on each leach product slurry
Filtration	SRC 2018	1	Single scoping test to evaluate filtration (in lieu of CCD) using a simulated comingled slurry
SX	SRC 2017	3	Continuous SX test on sample of solution from each composite
	SEPA 2017	5	Continuous SX test on sample of solution from 5 leaching tests
Neutralization	SRC 2017	3	Simulated neutralization test for each composite using leach discharge solids and raffinate form SX tests

Grinding

A single grinding test was completed at SRC on Gryphon material. During the test program, an additional sample (beyond those identified in the sample selection section) was selected by Denison and sent to SRC specifically to support the grinding test work. This was done as a coarser sample was needed for the SAG test work.

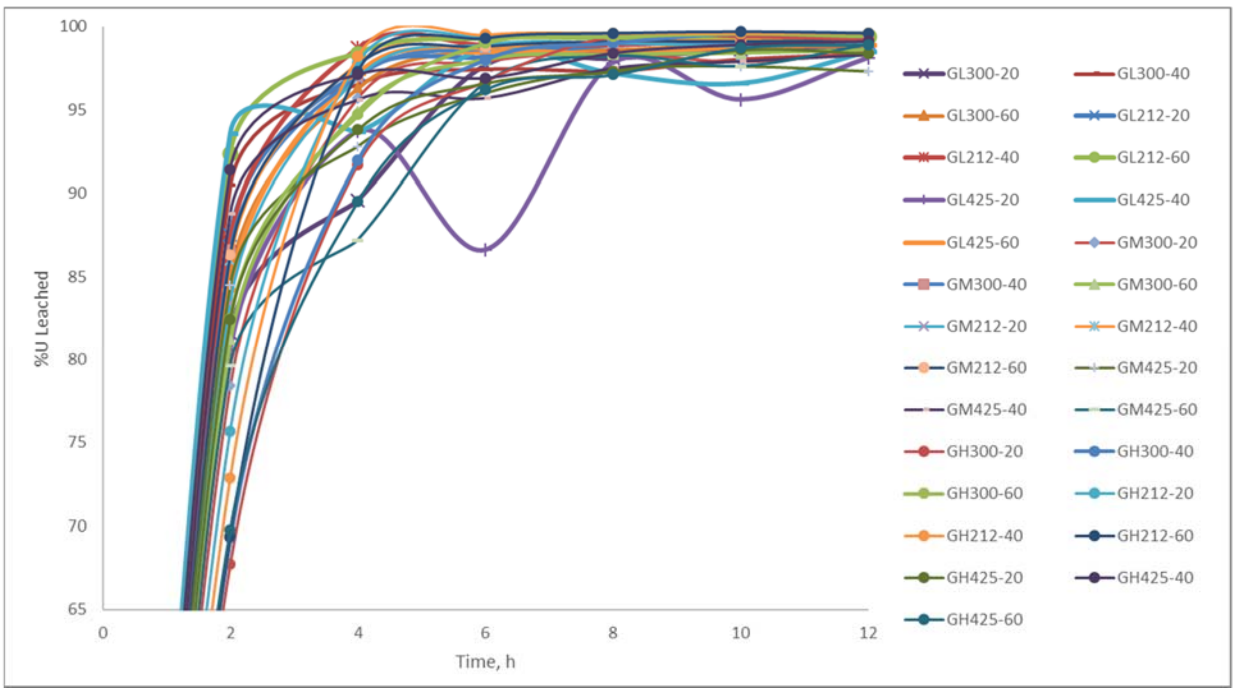
A SAG Design (W_{STD}) work index of 4.41 kWh/t and a standard Bond ball mill work index of 14.82 kWh/t were noted. Gryphon material can be described as soft from a SAG design standpoint, and moderate from a ball mill standpoint. This test is considered as indicative, however additional test work will be required to further establish the representativeness of the overall deposit material hardness.

Leaching

The original Gryphon leaching tests completed in 2014 arrived at optimal leaching conditions with a grind size of 300µm, 12 hours residence time, a target final free acid of 25 g/L, hydrogen peroxide added as an oxidant, and 1 g/L of ferric sulphate added to promote oxidation. These conditions resulted in a leaching recovery of 99.2% at 12 hours.

A leaching variability test program was completed at SRC in 2017, further investigating the key parameters of grind size P_{100} of (212 µm, 300 µm, and 425 µm), acid addition (varying residual free acid), and residence time (sub samples ranging from 0 to 12 hours). Testing was completed on all three Gryphon composites to assess metallurgical variability throughout the deposit.

All leaching tests utilized common operating conditions of 50% solids, atmospheric pressure, 50°C, and the use of 30%w/w hydrogen peroxide as the oxidant to maintain >450mV Oxidation Reduction Potential (ORP). In general, all Gryphon samples responded quickly to acid leaching, with over 85% recovery seen in under 4 hours of leaching in all high and medium grade tests (Figure 13-5). Finer grind sizes generally allowed for higher recovery at shorter leaching residence times.



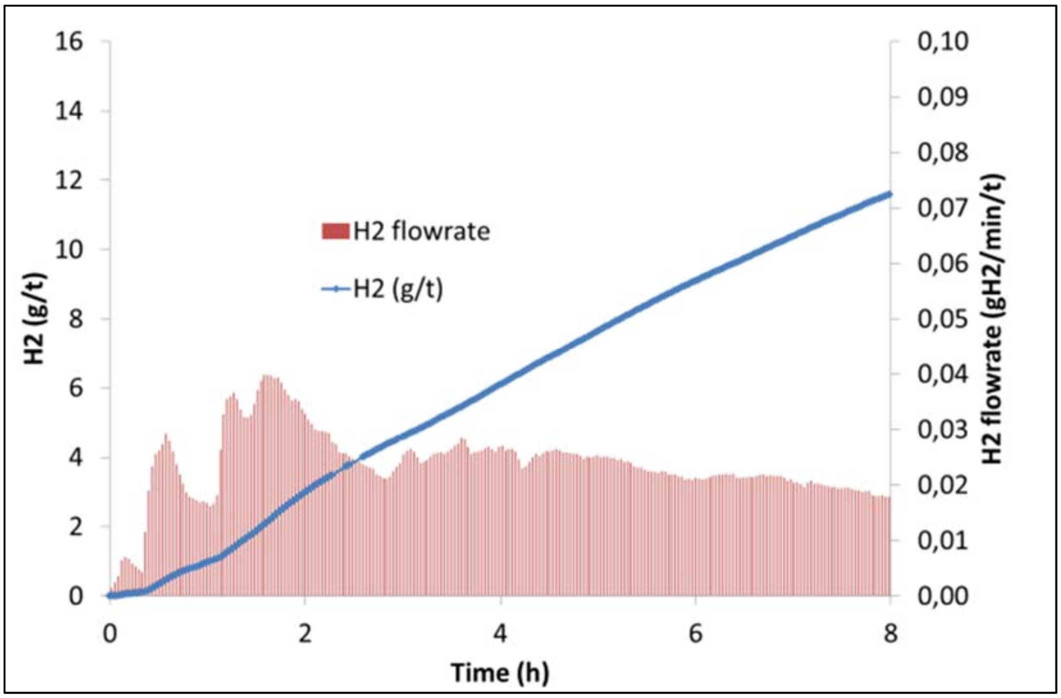
(Source: Hatch)

Figure 13-5. SRC – 2017 Gryphon Leaching Results – Uranium Recovery vs Time

Given the intent to utilize the existing McClean Lake mill, focus was placed on operating conditions that result in high recovery at an 8-hour residence time, and preferably with lower terminal acid additions to optimize operating costs. The low-grade Gryphon sample achieved a recovery of 99.0% with 8 hours residence time, fine grind (P_{100} of 212µm), and low terminal free acid.

In 2017, the SEPA lab in France completed additional leaching tests on Gryphon materials. SEPA was sent an independent set of core samples from site, which were composited, blended, and assayed. The intent of these tests was to further validate the SRC results and provide guidance on hydrogen evolution. Some ores in the Athabasca Basin have been known to release hydrogen during acid leaching, which presents a safety risk during processing. The McClean Lake mill secondary leach circuit was retrofitted, prior to processing Cigar Lake ores, with a number of novel safety layers of protection installed to mitigate the risk of hydrogen evolution during leaching. The initial leaching tests were not successful as ferric sulphate was not utilized to assist with oxidation. A single Gryphon low-grade test was repeated with ferric addition, and similar results to SRC were noted.

Hydrogen evolution testing demonstrated that Gryphon material evolve a small amount of hydrogen during acid leaching, with the high-grade material having the highest evolution rate (Figure 13-6). While the hydrogen evolution rate for Gryphon is quite low, it is recommended that the leaching circuit utilized for Gryphon employ hydrogen evolution design considerations.



Source: SEPA – "Denison Mines – Phoenix and Gryphon Study: Leaching, Hydrogen Evolution, and Solvent Extraction" dated June 20, 2018

Figure 13-6. SEPA – Leaching Test Work – Gryphon Low-Grade Hydrogen Evolution

Settling and Filtration

As part of the variability test work at SRC, the discharge slurry from each leaching test was subjected to settling test work. Preliminary flocculant screening was also completed. Low-grade Gryphon samples achieved terminal densities of between 40 and 44% solids.

Due to the expected high tonnage of Gryphon processing, a single pressure filtration test was completed to validate filtration as an alternative to the expansion of the counter current decantation (CCD) circuit. The test was completed on the slurry generated from a simulated comingled sample (the majority of the material was Gryphon, with Phoenix material used to simulate Cigar Lake ore).

Test work indicated that a filtration rate of 24 kg/m^2/h is achievable on the simulated comingled sample, and that washing of the filter cake is promising for aqueous uranium recovery, with the same uranium loss as CCD being achieved with minimal washing.

As only a single scoping filtration test was completed, additional filtration test work will be required in the next phase of work to validate the results.

Solvent Extraction

Continuous bench solvent extraction (SX) tests were completed on each composite to determine their performance though the SX process. The lab setup consisted of 4 stage extraction, 4 stage stripping, and 2 stage wash. Arsenic scrubbing was not tested due to the low amount of arsenic in the Gryphon sample. An organic solution of 6% Armeen 380, 3% Isodecanol, and 91% Diluent (CALUMET 400-500) was utilized. An Organic to Aqueous (O:A) ratio of 1:1 was utilized. The strip solution was 150 g/L ammonium sulphate solution, and the wash step utilized 10g/L sulphuric acid.

Each test was run for a minimum of 10 hours. Results indicated a high transfer of uranium through SX (>99.99%) and high rejection rate of all other impurities except for molybdenum, which typically is expected to transfer through uranium SX. No issues with the application of SX to Gryphon solution streams were observed.

Precipitation and Recovery

The pregnant strip solution from the SX tests were further subjected to yellowcake precipitation tests. The pregnant strip solution was adjusted to a pH of 7.0-7.5 with ammonium hydroxide. The yellowcake was then filtered, washed with di-ionized water, and calcined for two hours. Results are presented in Table 13-27.

It should be noted that a molybdenum removal unit operation was not tested on the Gryphon pregnant strip solution. As a result, a higher than expected amount of molybdenum reported to the calcined yellowcake. Molybdenum removal circuits are common in the uranium industry, and a circuit currently exists at the McClean Lake mill. Molybdenum's presence in the calcined yellowcake product is not considered to be a material risk. Other than for molybdenum, the calcined yellowcake produced from the Gryphon test work was high purity, with no other elements over the penalty limits specified under ASTM C967-13.

Table 13-27. Gryphon Calcined Yellowcake Assay vs Commercial Limits

Specifications	**ASTM C967-13 (Mass%, Uranium Basis)**		**Denison Mines U_3O_8 Sample (Mass%, Uranium Basis)**		
Component	**Limit without Penalty**	**Limit without Rejection**	**Gryphon Low Grade**	**Gryphon Medium Grade**	**Gryphon High Grade**
Uranium (U)	N/A	65% min.	87.0%	86.6%	79.6%
Calcium (Ca)	0.05%	1%	<0.01%	<0.01%	<0.01%
Chromium (Cr)	N/A	N/A	0.01%	0.02%	0.02%
Iron (Fe)	0.15%	1%	<0.008%	<0.008%	<0.009%
Lead (Pb)	N/A	N/A	0.002%	0.001%	0.000%
Magnesium (Mg)	0.02%	0.50%	<0.007%	<0.007%	<0.008%
Molybdenum (Mo)	0.10%	0.30%	0.56%	0.64%	0.20%
Phosphorus (PO_4)	0.10%	0.70%	<0.02%	<0.02%	<0.02%
Potassium (K)	0.20%	3%	0.02%	0.03%	0.02%
Silver (Ag)	N/A	N/A	0.007%	0.01%	0.007%
Sodium (Na)	1%	7.50%	<0.009%	<0.009%	<0.009%

Specifications	ASTM C967-13 (Mass%, Uranium Basis)		Denison Mines U_3O_8 Sample (Mass%, Uranium Basis)		
Component	Limit without Penalty	Limit without Rejection	Gryphon Low Grade	Gryphon Medium Grade	Gryphon High Grade
Thorium (Th)	0.10%	2.50%	<0.0002%	<0.0002%	<0.0003%
Titanium (Ti)	0.01%	0.05%	<0.007%	<0.007%	<0.008%
Vanadium (V)	0.06%	0.30%	0.001%	0.001%	0.001%
Zirconium (Zr)	0.01%	0.10%	0.02%	0.01%	0.002%

Source: SRC - "Denison Uranium Ore Metallurgical Testing Part Four: Uranium Solvent Extraction and Tailings Neutralization"

Tailings Neutralization

A neutralization test was completed at SRC for each Gryphon composite. The neutralization tests were designed to simulate the neutralization circuit currently used at the McClean Lake mill. The test included:

- Acidification of the slurry to pH 1.5;
- Addition of ferric sulphate for arsenic and molybdenum precipitation;
- Lime addition to pH 4;
- Addition of $BaCl_2$ and lime to pH 7.8; and
- Solid liquid separation of the neutralized product.

The results achieved are presented in Table 13-28. During test work, insufficient barium chloride was added for radium removal. As a result, the Ra^{226} levels exceed the regulatory limits in the test, however this is not considered to be a material risk as the regulatory limit should be achievable with the correct reagent addition dosage.

Table 13-28. SRC – Results of Neutralized Gryphon Filtrate Samples with CNSC and SK Limits

Element (mg/L)	CNSC*	SK**	Gryphon Low	Gryphon Medium	Gryphon High
Ag			<0.00005	<0.00005	<0.00005
Al			0.15	0.23	0.45
As (µg/L)	500	500	17	27	17
B			7.2	4.8	5
Ba			0.038	0.038	0.039
Be			<0.0001	<0.0001	<0.0001
Ca			555	551	546
Cd			0.00017	0.00022	0.00025
Co			0.011	0.012	0.01
Cr			<0.0005	<0.0005	<0.0005
Cu	0.3	0.3	0.0082	0.0089	0.0032
Fe			0.015	0.012	0.014
K			50	49	46
Mg			137	145	123
Mn			0.52	0.64	0.6
Mo			0.164	0.211	0.076
Na			32	36	86
Ni	0.5	0.5	0.014	0.018	0.016
Pb	0.2	0.2	0.0088	0.011	0.029
Sb			0.001	0.001	0.0014

Element (mg/L)	CNSC*	SK**	Gryphon Low	Gryphon Medium	Gryphon High
Se		0.6	0.019	0.024	0.011
Sn			<0.0001	<0.0001	<0.0001
SO_4^{2-}			1,950	2,010	2,030
Sr			2.36	2.12	1.84
Ti			0.0012	0.0016	0.0017
Tl			0.0037	0.005	0.0052
U (µg/L)		500	15	20	22
V		0.5	0.0029	0.0018	0.0017
Zn	0.5	0.5	0.0008	0.0012	0.001
TSS	25	25			
Ra^{228} (Bq/L)			<1	<0.9	<1
Ra^{226} (Bq/L)	0.37	0.37	14	12	19
Th^{230} (Bq/L)		3.7			
Pb^{210} (Bq/L)		1.85			
Min pH	6		6.11	6.42	7.5

Source: SRC - "Denison Uranium Ore Metallurgical Testing Part Four: Uranium Solvent Extraction and Tailings Neutralization"
*CNSC Effluent Discharge Limits for Cluff Lake, Key Lake and Rabbit Lake - Maximum Monthly Substance Unit Arithmetic Mean Concentration
**Saskatchewan Waste Water Quality Limits for Key Lake - Maximum Monthly Substance Unit Arithmetic Mean Concentration

13.2.3 Process Design Criteria

Based on the results of the metallurgical test program, key process design criteria (PDC) were established for the processing of Gryphon material. These are presented in Table 13-29. PDC developed from test work is primarily associated with leaching circuit operating conditions. Test work data was consulted in the development of PDC values for solid liquid separation and downstream uranium recovery circuits. However, due to the limited test work data, some assumptions and extrapolation have been used in generating the overall PDC for these circuits. These assumptions have been identified in the PFS report process design criteria. In general, Gryphon material respond well to conventional acid leaching with reasonable recoveries. Metallurgical test work conducted on Gryphon thus far has not identified any major risks, and the operating conditions align well to those currently employed at the McClean Lake mill.

It is important to note that the PDC and test work in this section refer to the standalone performance of Gryphon material. This study currently envisions co-milling of Cigar Lake and Gryphon feed in the McClean Lake mill. As a result, metallurgical results may vary during comingled processing and it is recommended that test work be completed on comingled Wheeler River and Cigar Lake material during the next phase of the project. Moreover, the PDC values utilized to support mill sizing (particularly reagent addition) consider the aggregate requirements of processing Wheeler River and Cigar Lake material comingled.

Table 13-29. Gryphon PDC Values

Parameter	Units	Value
Target Grind Size (P_{80})	µm	150
Leaching Residence Time	h	8
Leaching Temperature	°C	50
Leaching Pressure	kPa(g)	0
Leaching Acid Addition (98% H_2SO_4)	kg/t	60

Design Acid Addition (98% H_2SO_4)	kg/t	100
Hydrogen Peroxide Addition (22.5% H_2O_2)	kg/t	27
Ferric Sulphate Addition (45% $Fe_2(SO_4)_3$)	kg/t	25
Hydrogen Protection	Yes/No	Yes
Leaching Uranium Recovery (Gryphon ore)	%	99.0
Overall Mill Uranium Recovery (Comingled with Cigar Lake)	%	98.4

Note: Design acid addition is utilized for acid plant sizing
Source: Hatch – Wheeler Rivver – Processing Pre-feasibility, September 4, 2018

14 Mineral Resource Estimates

RPA has estimated mineral resources for the Phoenix and Gryphon deposits based on results of several surface diamond drilling campaigns from 2008 to 2017. The Phoenix deposit consists of Zone A and Zone B at the Athabasca unconformity, and Zone A basement mineralization which is immediately below the north part of Zone A. The Gryphon deposit consists of several stacked lenses in the basement and is located approximately three kilometers northwest of the Phoenix deposit.

Table 14-1 summarizes the mineral resource estimate, of which Denison's share is 63.3%. The effective date of the mineral resource estimate is January 30, 2018. The mineral resource estimate for Phoenix was reported in a previous NI 43-101 technical report (RPA, 2014) dated June 17, 2014 with an effective date of May 28, 2014, and there has been no change to the Phoenix mineral resource estimate since that time. Details of the estimation methodology follow below.

Table 14-1. RPA Mineral Resource Estimate – Wheeler River Project – January 30, 2018

Deposit	Category	Tonnes	Grade (% U_3O_8)	Million lbs U_3O_8 (100% Basis)	Million lbs U_3O_8 (Denison 63.3%)
Gryphon	Indicated	1,643,000	1.7	61.9	39.2
Phoenix	Indicated	166,000	19.1	70.2	44.4
Total Indicated		**1,809,000**	**3.3**	**132.1**	**83.6**
Gryphon	Inferred	73,000	1.2	1.9	1.2
Phoenix	Inferred	9,000	5.8	1.1	0.7
Total Inferred		**82,000**	**1.7**	**3.0**	**1.9**

Notes:

1. CIM definitions (2014) were followed for classification of mineral resources.
2. Mineral resources for the Gryphon deposit are estimated at an incremental cut-off grade of 0.2% U_3O_8 using a long-term uranium price of USD$50/lb and a USD$/CAD$ exchange rate of 0.75. The cut-off grade is based on incremental operating costs for low-grade material.
3. Mineral resources for the Phoenix deposit are reported above a cut-off grade of 0.8% U_3O_8. Mineral resources for the Phoenix deposit were last estimated in 2014 to reflect the expansion of the high-grade zone. As no new drilling has been completed at Phoenix since that time, the mineral resource estimates for the Phoenix deposit remain current.
4. High-grade mineralization was capped at 30% U_3O_8 and restricted at 20% U_3O_8 for the A1HG and capped at 20% U_3O_8 for the D1HG with no search restrictions.
5. Low-grade mineralization was capped at 20% U_3O_8 for the C1 domain with search restrictions applied to U_3O_8 grades greater than or equal to 10.0% U_3O_8.
6. Low-grade mineralization was capped at 15% U_3O_8 for the B1, B2, E1, and E2 domains with search restrictions applied to U_3O_8 grades greater than or equal to 10.0% U_3O_8 for the B1 domain and 5.0% U_3O_8 for the E2 domain.
7. Low-grade mineralization was capped at 10% U_3O_8 for the A1-A4, B3-B7, C4-C5, and D2-D4 domains with no search restrictions.
8. Low-grade mineralization was capped at 5% U_3O_8 for the D1 domain with no search restrictions.
9. Bulk density is derived from grade using a formula based on 196 measurements from Phoenix and 279 measurements from Gryphon.
10. A minimum mining width of 2 m was used.
11. Numbers may not add due to rounding.
12. Mineral resources are inclusive of mineral reserves.
13. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

RPA is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the mineral resource estimate.

14.1 Drill Hole Database

The property drill hole database includes drilling results from 1985 to 2017, which comprise 641 diamond drill holes totalling 328,357 m, of which 243 drill holes totalling 113,676 m have delineated the Phoenix deposit and 214 drill holes totalling 120,351 m have delineated the Gryphon deposit. Zone A at Phoenix is the northeastern lens and strikes N52°E and Zone B consists of two subzones, B1 and B2, which form the southwestern part of the Phoenix deposit. Zone A basement mineralization is within a narrow fracture zone that extends below the northern end of Zone A. The Gryphon deposit is a series of stacked basement mineralized lenses striking N20°E, referred to as the A (100 Series), B (200 Series), C (300 Series), D (400 Series), and E (500 Series) lenses.

Upon completion of the initial data processing, the borehole data as well as radiometric logging information was uploaded into VULCAN software. Table 14-2 lists details of the VULCAN database used for the resource estimate. Section 12 describes the verification steps made by RPA. In summary, no discrepancies were identified, and RPA is of the opinion that the drill hole database is valid and suitable to estimate mineral resources for the Phoenix and Gryphon deposits.

Table 14-2. Vulcan Database Records

Table Name	Number of Records	
	Gryphon	Phoenix
Collar	214	243
Survey	3,857	2,587
Stratigraphy	4,461	2,107
Assay Values	5,591	2,058
Radiometric Values (% eU_3O_8)	144,020	166,492
Block Model 1m Composites in Wireframes	1,975	703
A Deposit UC - Composites		471
B Deposit UC - Composites		92
A Deposit Basement - Composites		140

Drill holes at Phoenix were completed on northwest-southeast oriented sections spaced at approximately 25 m intervals along strike with a drill hole spacing of approximately 10 m along the sections. Earlier holes were drilled at steep angles to the northwest and later holes were collared vertically. Figure 14-1 shows Zones A and B with locations of drill holes. Figure 14-2 shows the location of the Zone A basement mineralization.

For Gryphon, drill holes were completed on northwest-southeast oriented sections spaced at approximately 50 m intervals along strike with a drill hole spacing of approximately 50 m along the sections. Figure 14-3 shows the locations of drill holes at Gryphon.

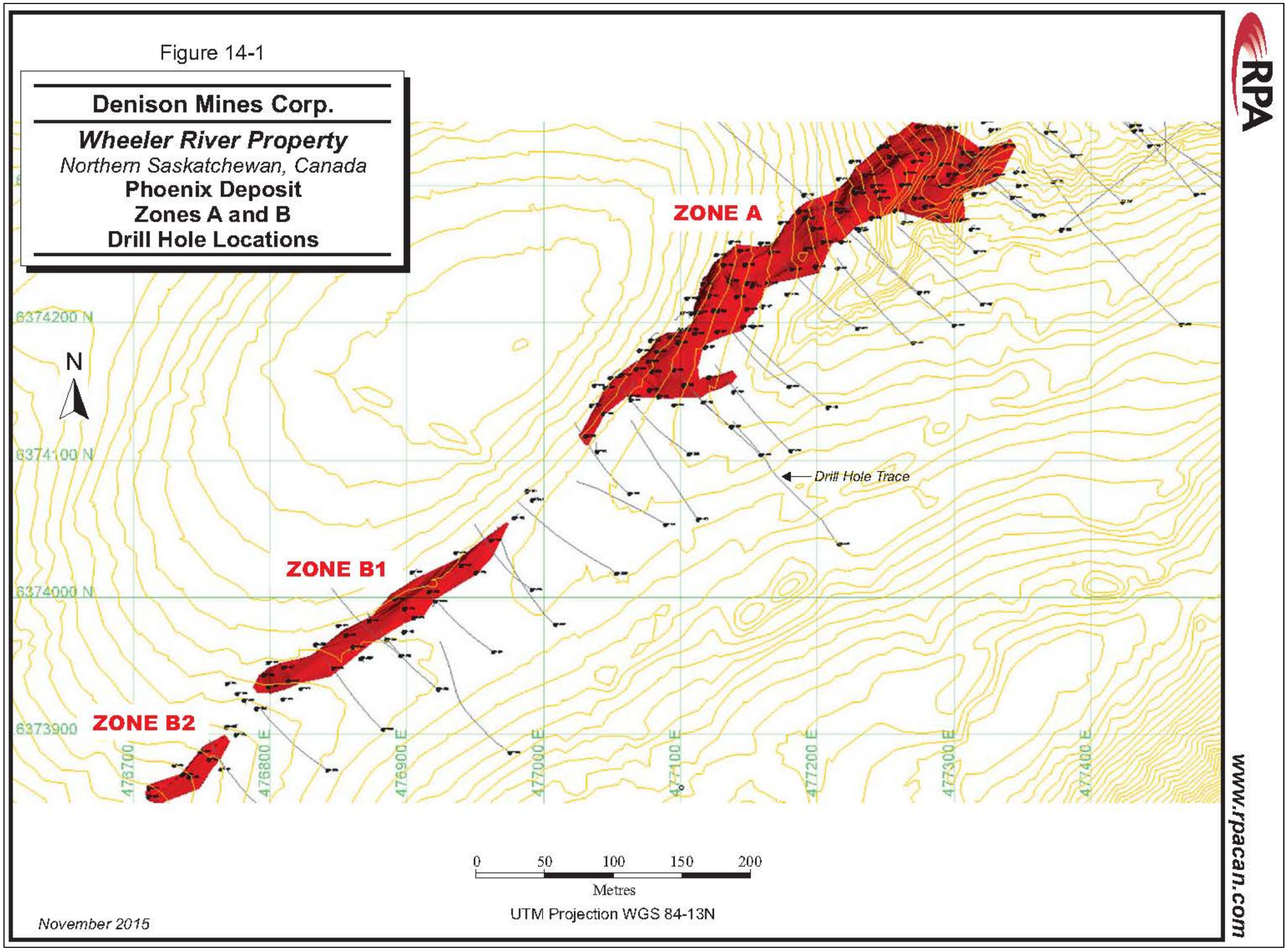


Figure 14-1. Phoenix Deposit Zones A and B Drill Hole Locations

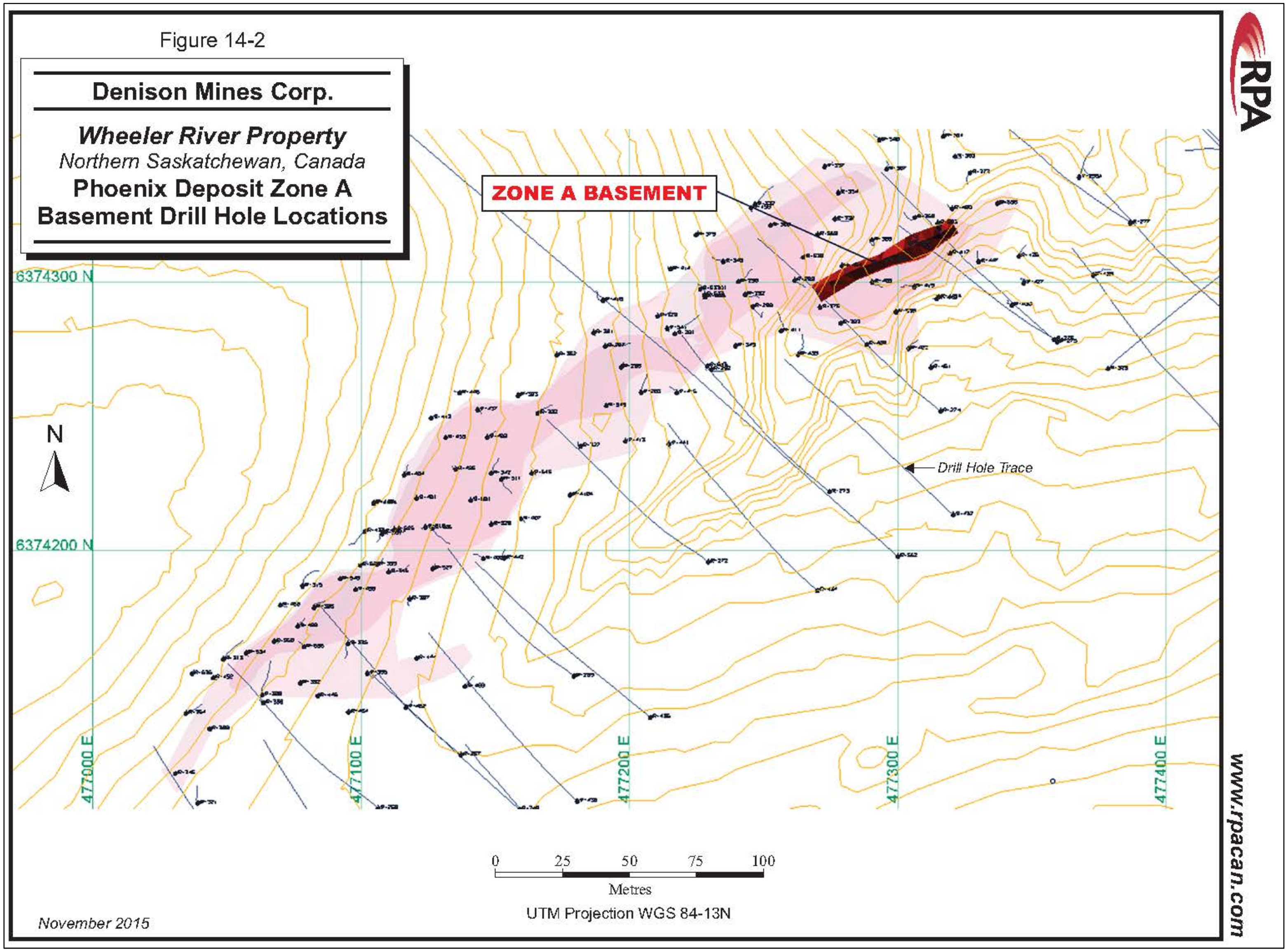


Figure 14-2. Phoenix Deposit Zone A Basement Drill Hole Locations

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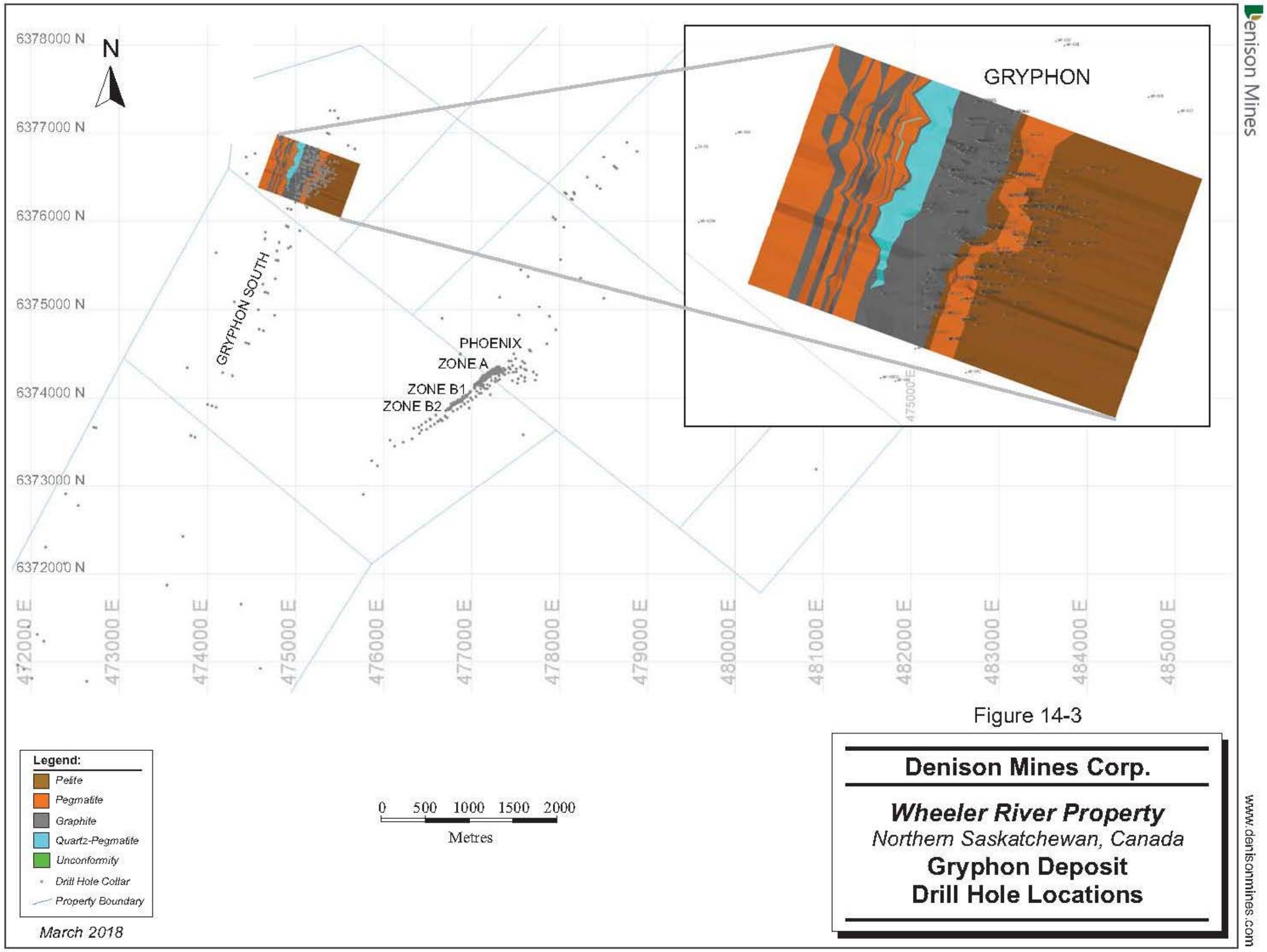

Figure 14-3. Gryphon Deposit Drill Hole Locations

14.2 Geologic Interpretation and 3D Solids

14.2.1 Phoenix Deposit

Denison has interpreted the geology, structure, and mineralized zones at Phoenix using data from 196 diamond drill holes that penetrate the basal unconformity of the Athabasca sandstone. Uranium mineralization occurs at the unconformity surface and in the adjacent sandstone above and in the adjacent graphitic pelitic gneiss basement rocks below the unconformity. Zones A and B both strike approximately N52°E and are essentially horizontal.

A regional fault, the WS Fault, is spatially associated with mineralization in the Phoenix deposit. The WS Fault trends north-easterly, parallel to the mineralization, and dips moderately to the southeast. It appears to be a steep angle reverse fault, displacing the unconformity in the order of 5 m or more upward on the southeast side. Uranium mineralization extends outward to the southeast from the WS Fault, suggesting that the primary controls on the Phoenix deposit are the intersection of the WS Fault with the unconformity and graphitic pelitic gneiss in the basement. Some uranium mineralization occurs on the northwest side of the WS Fault along the unconformity which is at lower elevation, however, it is limited in extent to the northwest. Other faults are present in the Phoenix deposit sub-parallel to the WS Fault but with lesser vertical displacements. Some cross faults with easterly or southeasterly trends are interpreted, with displacements in the order of 5 m or more.

The Zone A basement mineralization is restricted to a narrow (<3 m) fracture zone extending approximately 20 m below the northern end of Zone A. The fracture zone runs parallel to the strike of Zone A at approximately N52°E and dips at -65° to the southeast. The axis of the fracture is centred along drill holes WR-503, WR-403, and WR-506 and is interpreted as splay faulting associated with the WS Fault described previously.

Denison developed three-dimensional (3D) wireframe models, which were reviewed and accepted by RPA for the Phoenix deposit Zones A and B. The models represent grade envelopes using the geological interpretation described above as guidance. The wireframes consisted of a lower grade (LG) domain and a higher grade (HG) domain. For the LG wireframe, a threshold grade of 0.05% U_3O_8 was used as a guide. For Zone A, the threshold grade for inclusion in the HG domain was approximately 20% U_3O_8, although lower grades were incorporated in places to maintain continuity and to maintain a minimum thickness of 2 m. For Zone B, the minimum threshold for the HG domain was approximately 10% U_3O_8 over a minimum thickness of 2 m. Figures 14-4 to 14-6 are cross-sections of Zone A showing drill holes with 1-meter composite grades and the outlines of the HG and LG domains. Figure 14-7 shows the same for Zone B. Figure 14-8 is a longitudinal view of the Zone A basement domain.

The wireframe model developed for Zone A is approximately 380 m long, 36 m wide, and ranges in thickness from 2 m to 17 m with an average thickness of 5 m. The Zone B wireframe model measures approximately 290 m long, averages 19 m wide, and is approximately 3 m thick. The wireframes were used to assign domain codes to the blocks in the block model and for generating and coding composited assays.

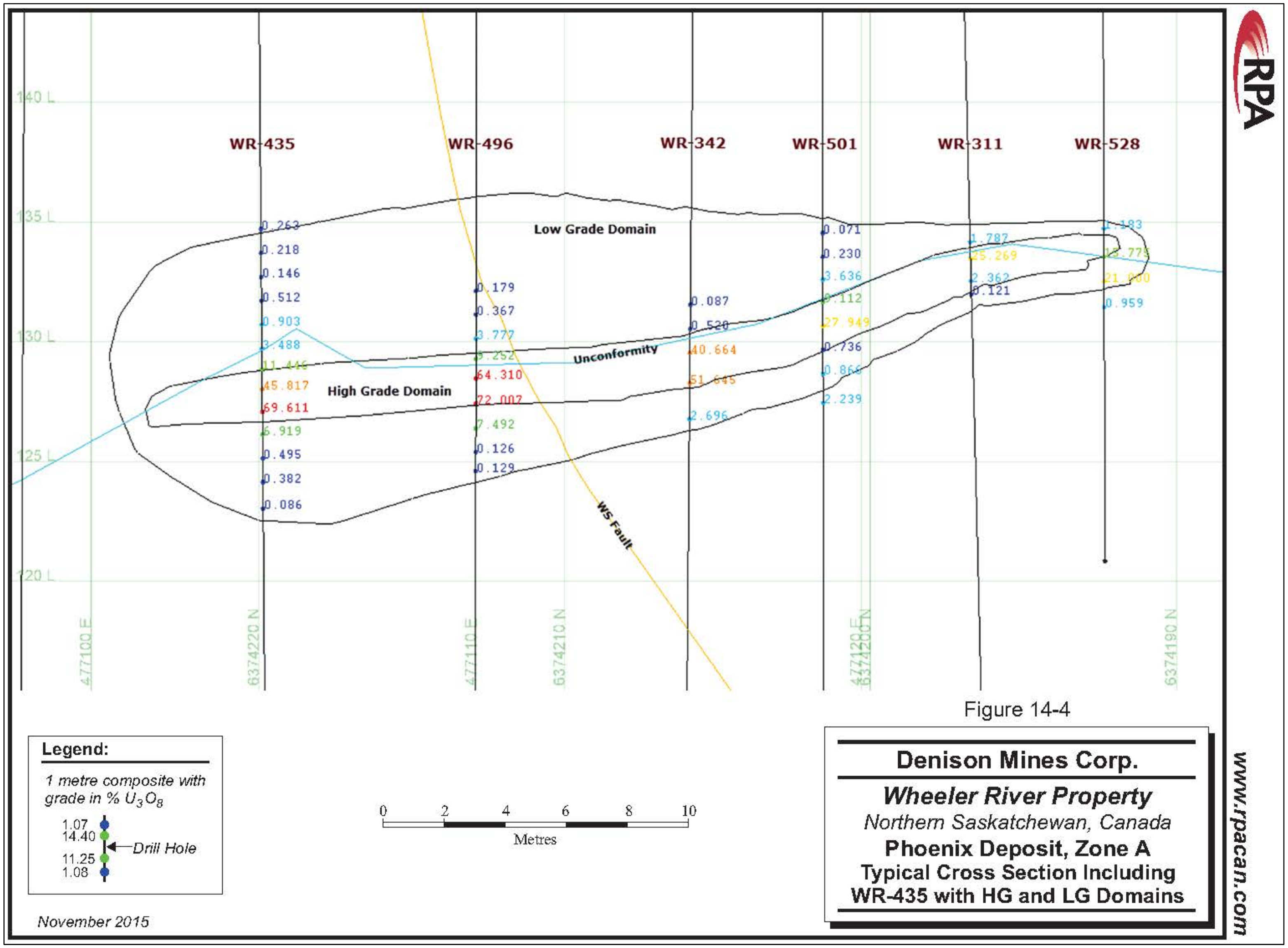

Figure 14-4. Phoenix Deposit Zone A Typical Cross-section Including WR-435 with HG and LG Domains

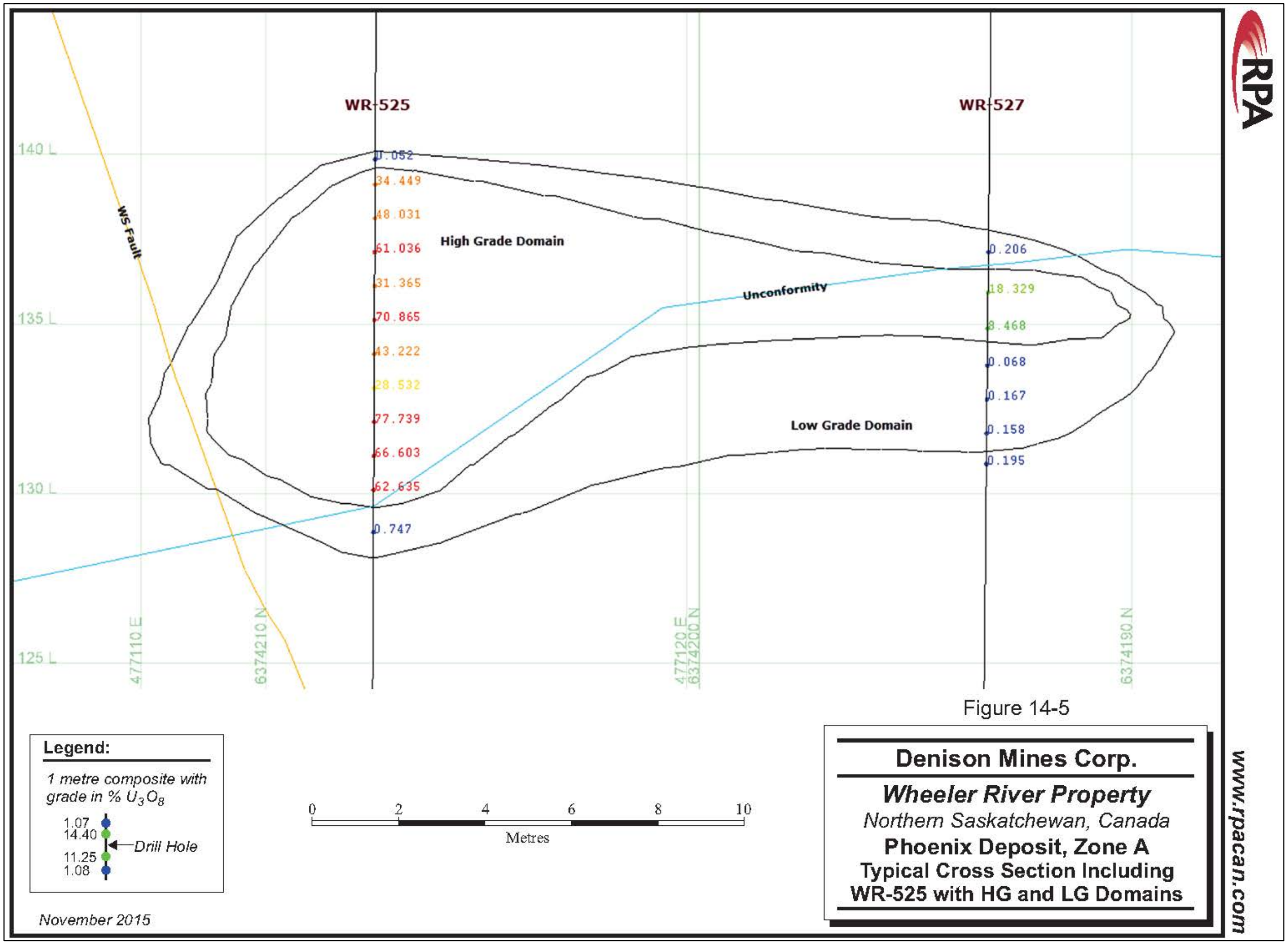

Figure 14-5. Phoenix Deposit Zone A Typical Cross-section Including WR-525 with HG and LG Domains

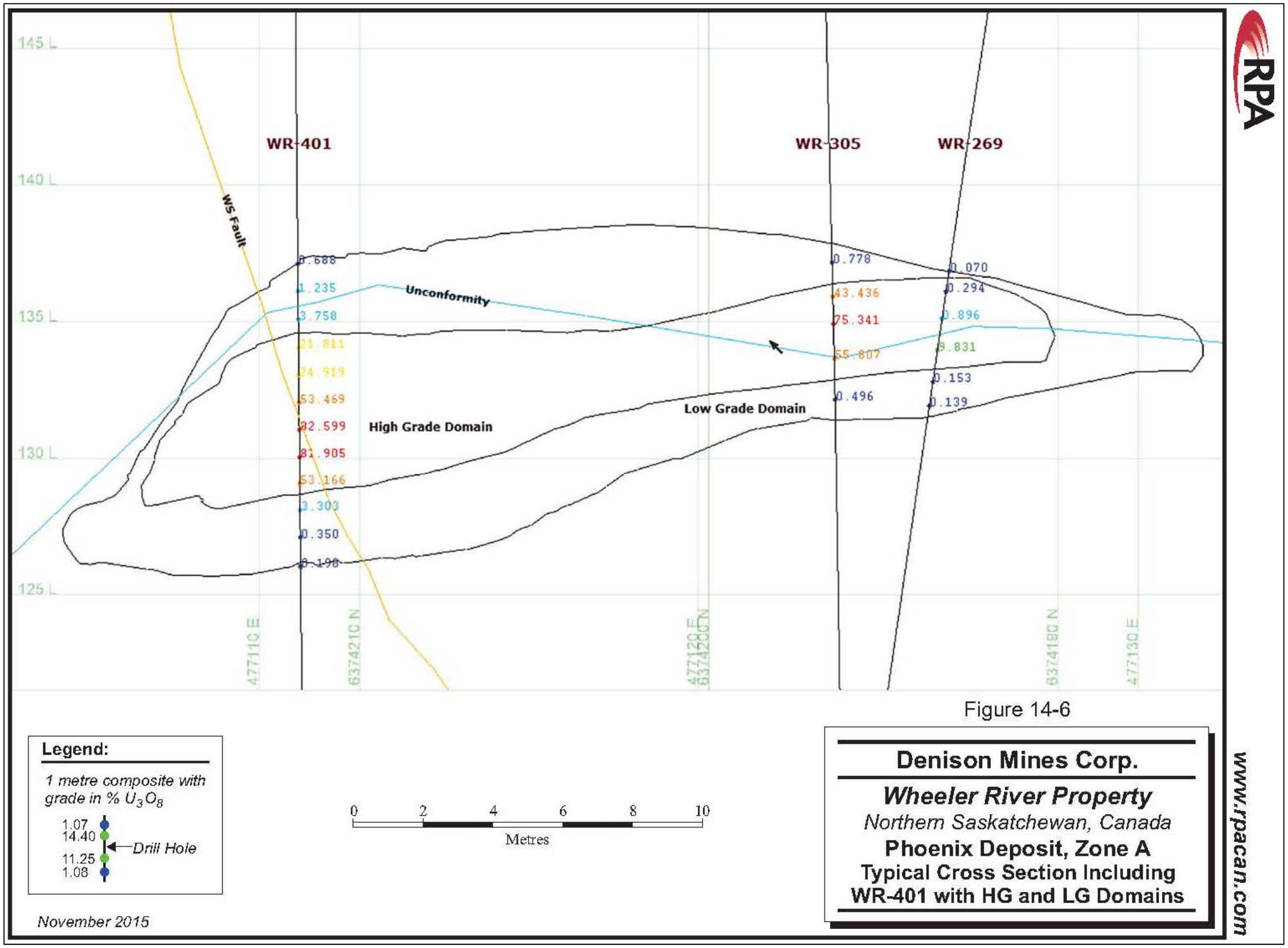

Figure 14-6. Phoenix Deposit Zone A Typical Cross-section Including WR-401 with HG and LG Domains

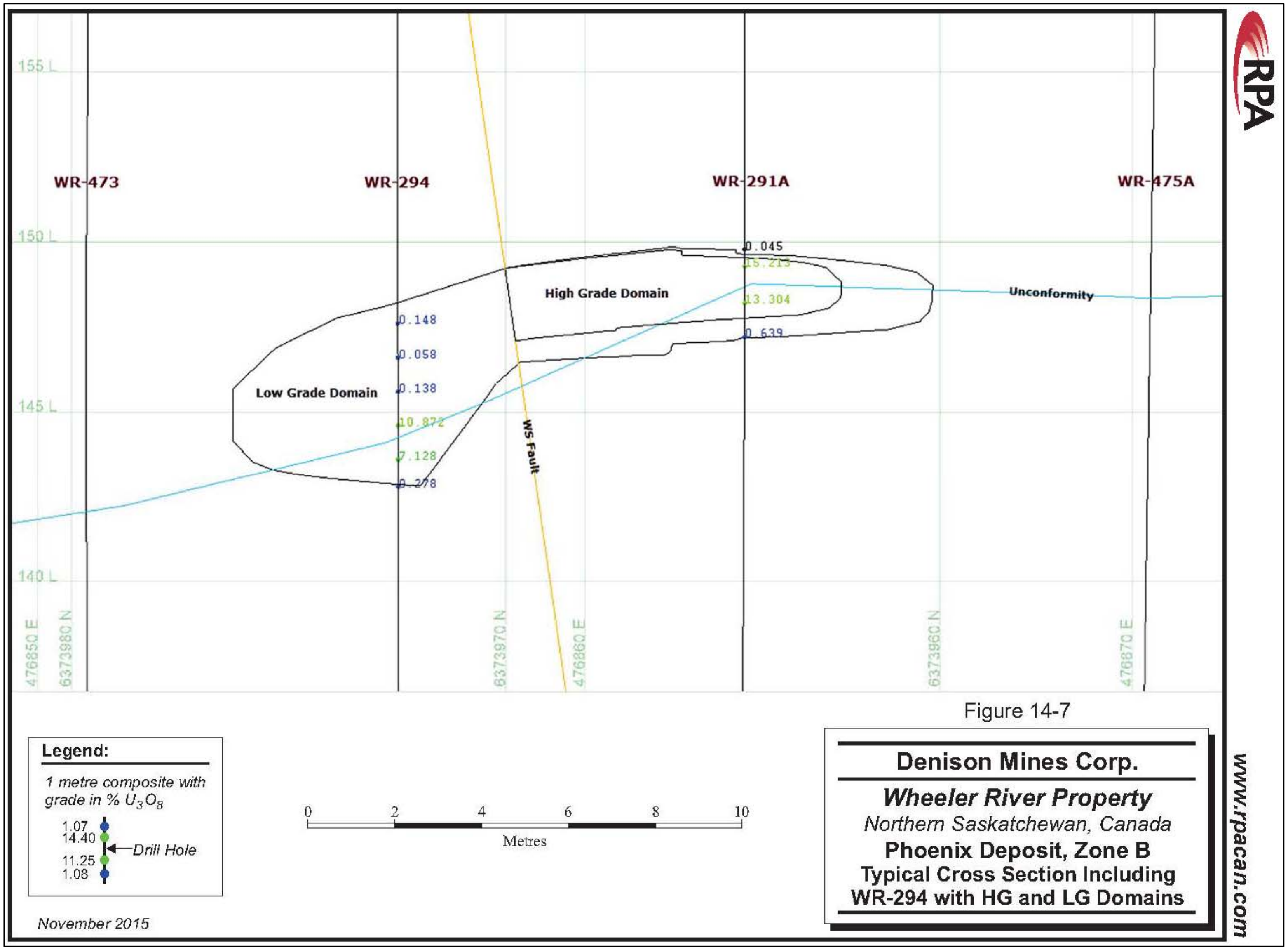

Figure 14-7. Phoenix Deposit Zone B Typical Cross-section Including WR-294 with HG and LG Domains

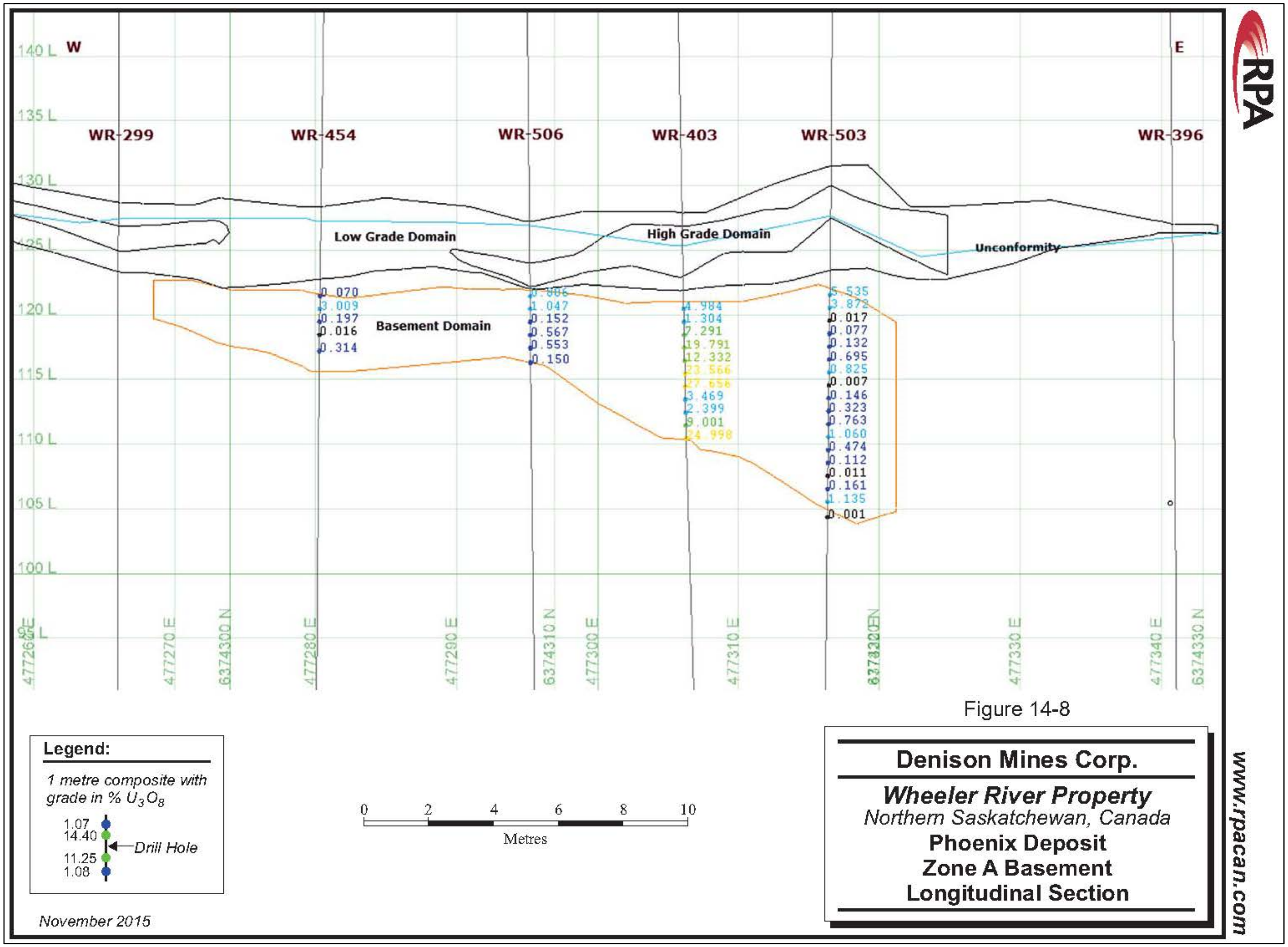

Figure 14-8. Phoenix Deposit Zone A Basement Longitudinal Section

14.2.2 Gryphon Deposit

Wireframe models of mineralized zones were used to constrain the block model grade interpolation process, based on a total of 210 holes. Previously, Gryphon wireframes were prepared by RPA using 3D polylines on northeast looking vertical sections spaced approximately 12.5 m apart. Polylines were "snapped" to assay intervals along the drill hole traces such that the sectional interpretations "wobbled" in 3D space. Polylines were joined together in 3D and the continuity was checked using a longitudinal section and level plans. Following the initial resource estimate in 2015, topographical surfaces, solids, and mineralized wireframes supporting the estimate were updated and remodelled in Gems software by Denison personnel and then audited for completeness and accuracy by RPA using Vulcan software. Extension distance for the mineralized wireframes was half-way to the next hole, or approximately 25 m vertically and horizontally past the last drill intercept. In some instances, it was necessary to reduce the extension distance to 12.5 m.

High-grade (HG) domain models were created using mineralized drill hole intercepts equal to or greater than two meters with a minimum grade of 4% U_3O_8. Lower grades were incorporated in places to maintain continuity and a minimum thickness of two meters. Other domain models were created using a lower threshold grade of 0.05% U_3O_8 and a minimum core length of two meters as a guide. RPA considers the selection of 0.05% U_3O_8 to be appropriate for construction of mineralized wireframe outlines, since this value reflects the lowest cut-off grade that is expected to be applied for reporting of the mineral resources in an underground operating scenario and is consistent with other known deposits in the Athabasca Basin. Sample intervals with assay results less than the nominated cut-off grade (internal dilution) were included within the mineralized wireframes if the core length was less than two meters or allowed for modelling of grade continuity.

In total, a series of 24 stacked lenses or domains of variable thicknesses that plunge 35° to 60° at 035° to 040° northeast, and dip 25° to 50° to the southeast were constructed, including four high-grade wireframes constrained within two enveloping wireframes (Table 14-3 and Figures 14-9 and 14-10).

The stacked lenses form a zone of mineralization measuring approximately 280 m long (along plunge) by 113 m wide (across plunge) and remain open both up and down plunge. Wireframes were assigned to zones as identified by Denison public disclosures.
The A1 domains together make up approximately 51% of the contained pounds of U_3O_8 in the mineral resource.

RPA conducted audits of the wireframes to ensure that the wireframes used in preparing the current resource estimate correspond to the reported mineralization. Quality control measures and the data verification procedures repeated in 2017 included the following:

- Check for overlapping wireframes to determine possible double counting;
- Check mineralization/wireframe extensions beyond last holes to see if they are reasonable and consistent;
- Check for reasonable compositing intervals;
- Check that composite intervals start and stop at wireframe boundaries; and
- Validate the solids for closure and consistent topology, and check that the triangles intersect properly (crossing). Any issues found were corrected with the appropriate Vulcan utility to ensure accurate volume and grade estimates.

RPA has accepted the mineralized wireframe domains prepared by Denison as appropriate for resource estimation.

Table 14-3. Summary of Gryphon Wireframe Models

Zone	Wireframe Name	Volume (m^3)	Tonnage	Block Model Code	Block Volume (m^3)	% Δ Volume
A1	GRYPHON_MINERALIZATION_GP_A1HG_11_15_CLIP.00t	61,186	146,845	1011	60,400	-1.28%
	GRYPHON_MINERALIZATION_GP_A1_11_15_2017_cut.00t	229,550	550,921	101	230,550	0.44%
A2	GRYPHON_MINERALIZATION_GP_A2_11_15_2017.00t	189,380	454,511	102	189,080	-0.16%
A3	GRYPHON_MINERALIZATION_GP_A3_11_15_2017.00t	22,130	53,112	103	22,440	1.40%
A4	GRYPHON_MINERALIZATION_GP_A4_11_15_2017.00t	7,690	18,456	104	7,560	-1.69%
B1	GRYPHON_MINERALIZATION_GP_B1_11_15_2017.00t	155,368	372,882	201	155,050	-0.20%
B2	GRYPHON_MINERALIZATION_GP_B2_11_15_2017.00t	88,674	212,818	202	88,860	0.21%
B3	GRYPHON_MINERALIZATION_GP_B3_11_15_2017.00t	31,643	75,943	203	31,270	-1.18%
B4	GRYPHON_MINERALIZATION_GP_B4_11_15_2017.00t	4,700	11,281	204	4,930	4.89%
B5	GRYPHON_MINERALIZATION_GP_B5_11_15_2017.00t	8,566	20,557	205	8,620	0.63%
B6	GRYPHON_MINERALIZATION_GP_B6_11_15_2017.00t	16,468	39,523	206	16,180	-1.75%
B7	GRYPHON_MINERALIZATION_GP_B7_11_15_2017.00t	7,499	17,997	207	7,430	-0.92%
C1	GP_C1_GRYPHON_2017_RESOURCE_C1C2C3_COMBI.00t	55,514	133,234	301	55,520	0.01%
C4	GRYPHON_MINERALIZATION_GP_C4_11_15_2017.00t	15,283	36,680	304	15,540	1.68%
C5	GRYPHON_MINERALIZATION_GP_C5_11_15_2017.00t	11,505	27,612	305	11,610	0.91%
D1	GRYPHON_MINERALIZATION_GP_D1HG_HW_12_14_17.00t	6,813	16,350	4011	6,950	2.02%
	GRYPHON_MINERALIZATION_GP_D1HG_MD_12_15_17.00t	4,284	10,282	4012	4,540	5.97%
	GRYPHON_MINERALIZATION_GP_D1HG_FW_12_14_17.00t	5,943	14,263	4013	6,090	2.48%
	GRYPHON_MINERALIZATION_GP_D1LG_12_15_17_cut.00t	154,890	371,737	401	154,500	-0.25%
D2	GRYPHON_MINERALIZATION_GP_D2_11_15_2017.00t	4,519	10,845	402	4,460	-1.30%
D3	GRYPHON_MINERALIZATION_GP_D3_11_15_2017.00t	7,779	18,670	403	7,490	-3.72%
D4	GRYPHON_MINERALIZATION_GP_D4_11_15_2017.00t	66,983	160,760	404	67,940	1.43%
E1	GP_E1_GRYPHON_2017_RESOURCE_2018-01-09.00t	15,818	37,963	501	15,330	-3.09%
E2	GP_E2_GRYPHON_2017_RESOURCE_2018-01-09.00t	65,517	157,240	502	65,480	-0.06%
Total		**1,237,702**	**2,970,484**		**1,237,820**	**0.01%**

Notes:

1. A-Series (A1 through A4): represent the mineralized zones on the hangingwall (Upper Zone) of the quartz-pegmatite assemblage along the G-fault.
2. B-Series (B1 through B7): represent the mineralized zones within the quartz-pegmatite assemblage along the G-fault.
3. C-Series (C1, C4, and C5): represent the mineralized zones along the footwall (Lower Zone) of the quartz-pegmatite assemblage along the G-fault.
4. D-Series (D1 through D4): represent the mineralized zones within the pegmatite-dominated footwall along a secondary fault zone ("Basal Fault") or within extensional relay faults which link to the G-fault.
5. E-Series (E1 and E2): represent the mineralized zones occur along the G-fault, up-dip and along strike to the northeast of the A and B series lenses, within the upper basement or at the sub-Athabasca unconformity.

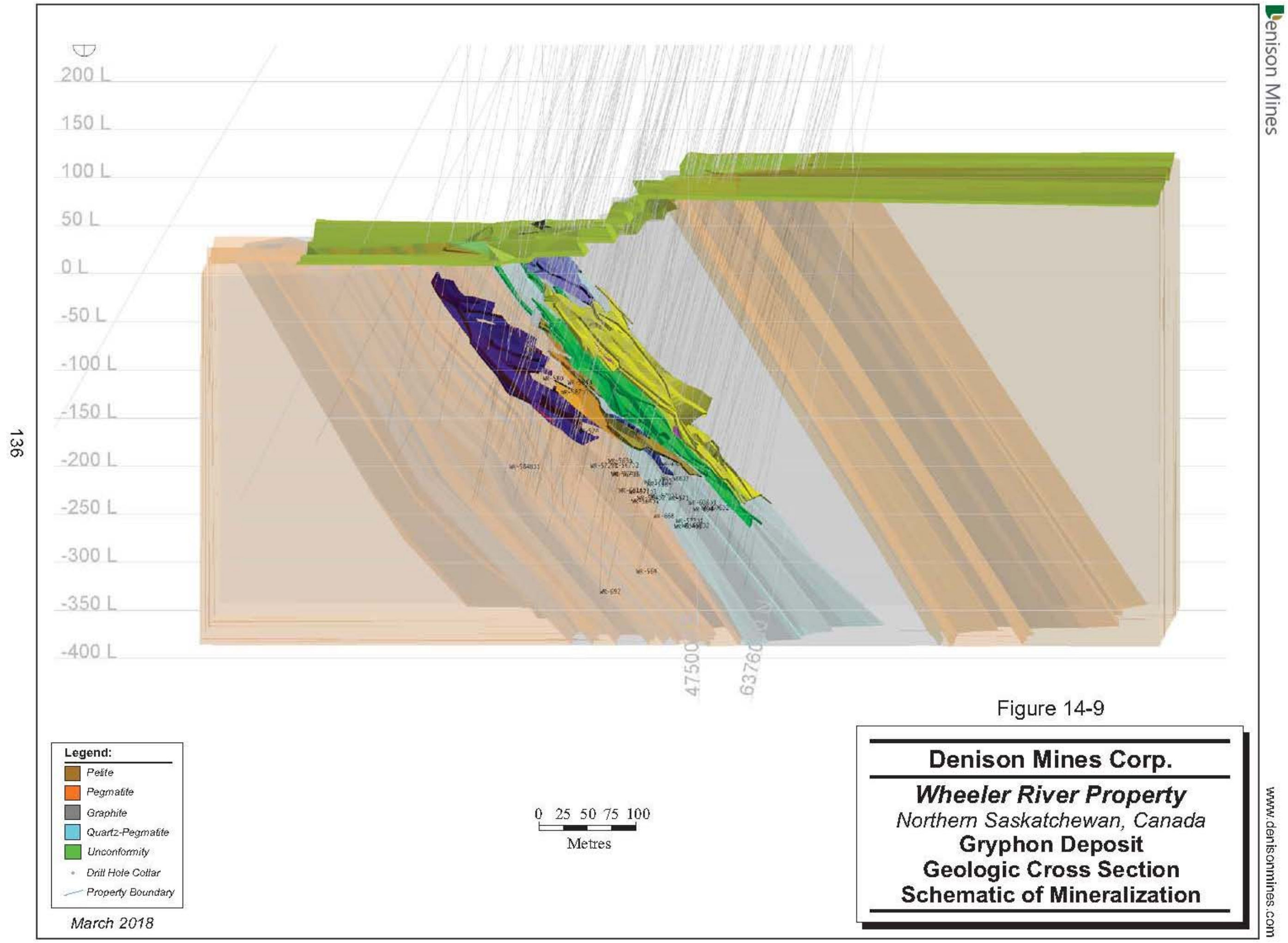

Figure 14-9. Gryphon Deposit Geologic Cross-section Schematic of Mineralization

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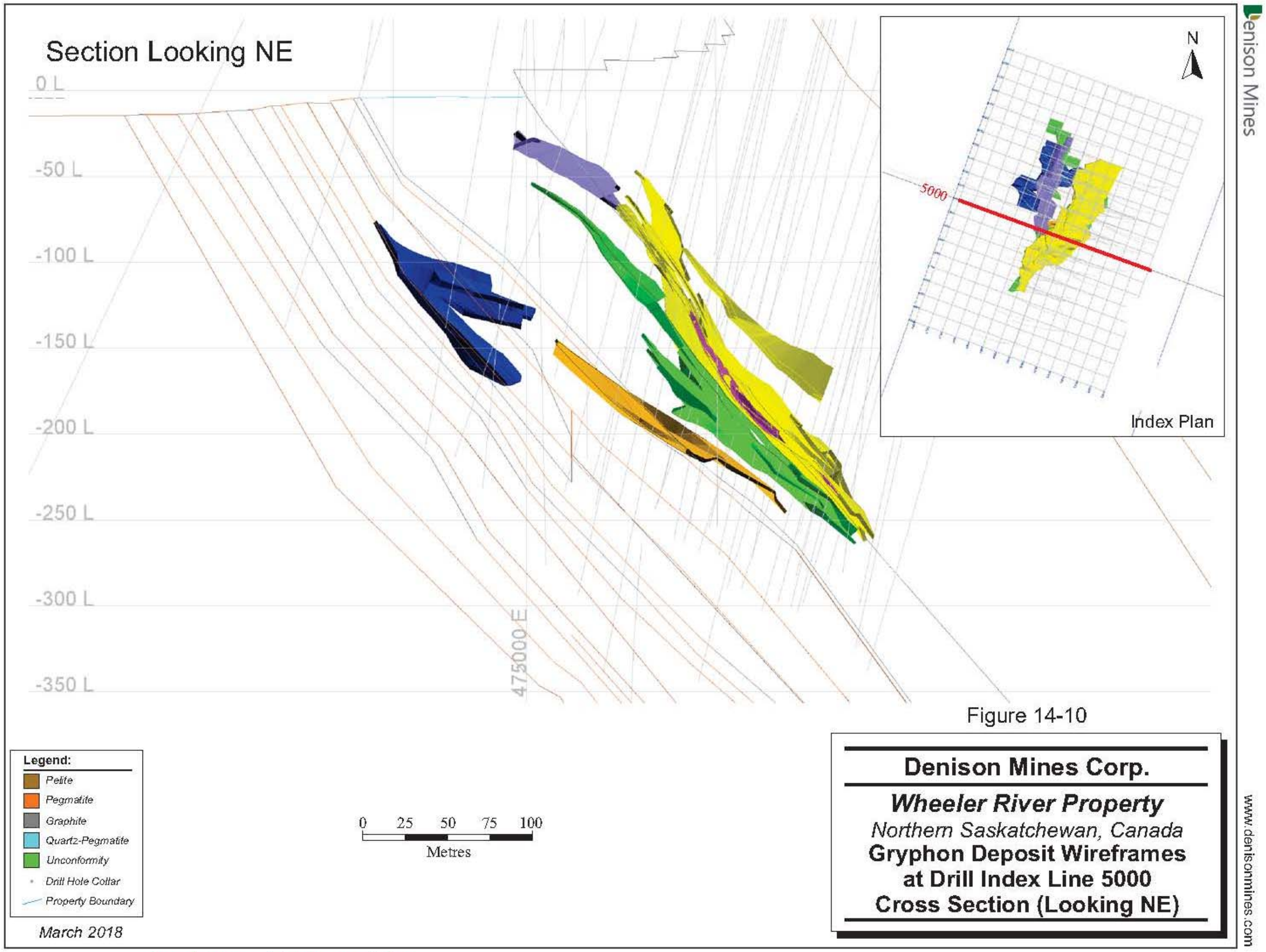


former 'Strat Legend' doesn't seem valid here

Figure 14-10. Gryphon Deposit Wireframes at Drill Index Line 5000 Cross-section (Looking NE)

14.3 Bulk Density

Bulk density is used to convert volume to tonnage and to weight the block grade estimates. In high-grade uranium deposits such as Gryphon, bulk density varies with grade due to the very high density of pitchblende/uraninite compared to host lithologies. Bulk density also varies with clay alteration and in situ rock porosity. For mineral resource estimates of high-grade uranium deposits, it is important to estimate bulk density values throughout the deposit and to weight grade values by density since small volumes of high-grade material contain large masses of uranium oxide.

Bulk density is determined by Denison with specific gravity (SG) measurements on drill core. SG is calculated as: weight in air/(weight in air – weight in water). Under all reasonable conditions, SG (a unitless ratio) is equivalent to density in t/m^3.

14.3.1 Phoenix Deposit

From 2012 to 2014, Denison completed a program of dry bulk density sampling from diamond drill core in order to establish the relationship between bulk density and grade for the Phoenix deposit Zones A and B. Dry bulk density samples were selected from the main mineralized zones to represent local major lithologic units, mineralization styles, and alteration types. Samples were collected from half split core, which had been previously retained in the core box after geochemical sampling. Samples were tagged and placed in sample bags on site, then shipped to the SRC in Saskatoon, Saskatchewan. In total, SRC has performed SG measurements on a total of 196 samples; 162 from Zone A and 34 from Zone B.

Denison carried out correlation analyses of the bulk density values against uranium grades which indicated a strong relationship between density and uranium grade (% U_3O_8) shown in Figure 14-11. The relationship can be represented by the following polynomial formula which is based on a regression fit.

$$y = 0.0008x^2 – 0.0077x + 2.3361$$

where *y* is dry bulk density (g/cm^3) and *x* is the uranium grade in % U_3O_8. In some cases when the samples are very clay rich, core fatigue (sample crumbles) prevented the wax from being applied and SG was calculated using the wet/dry method only. Figure 14-12 shows a strong correlation between the methodologies and RPA is satisfied that either methodology is suitable for determining SG.

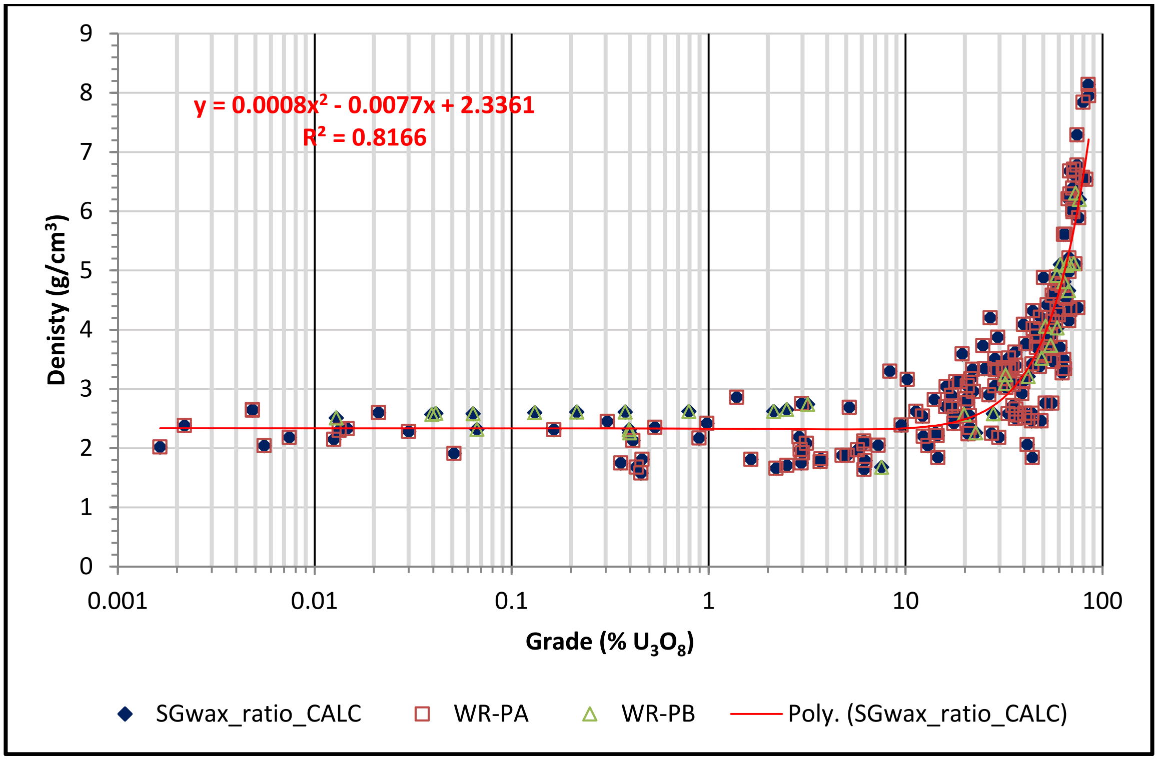


Figure 14-11. Logarithmic Plot of Dry Bulk Density vs Uranium Grade – Phoenix Deposit

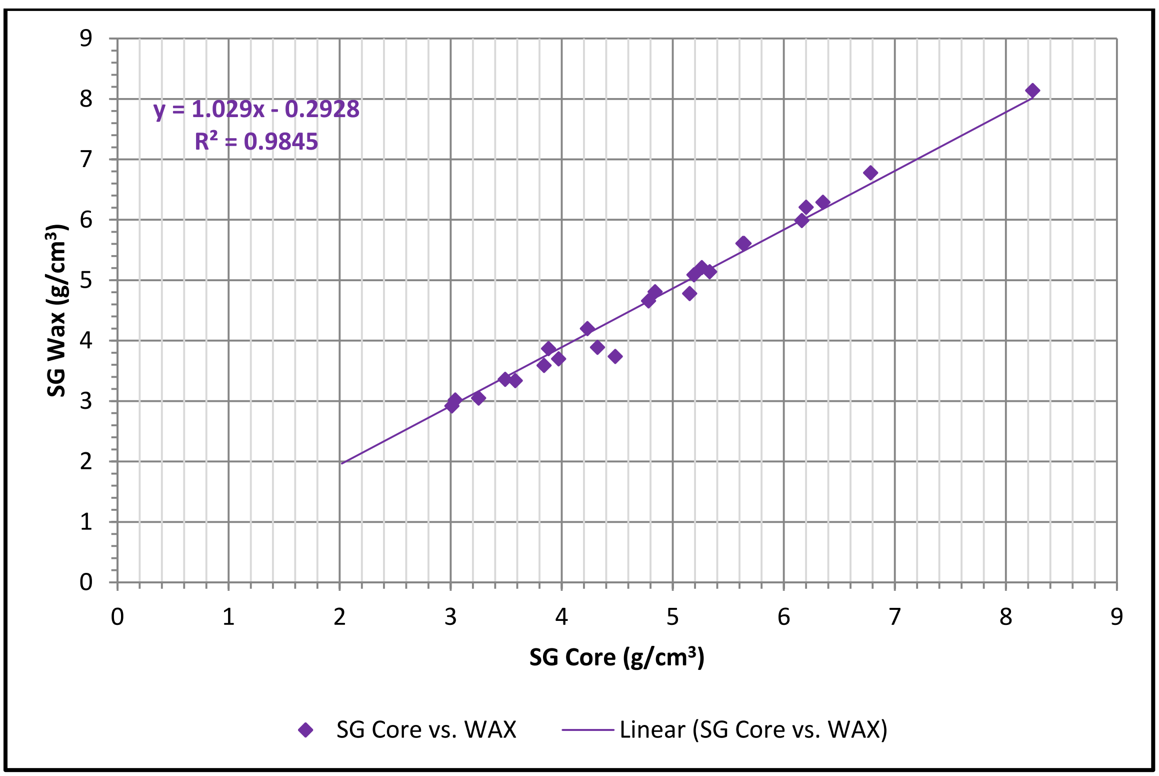


Figure 14-12. Dry Bulk Density Wax vs Dry/Wet Methods – Phoenix Deposit

The regression curve in Figure 14-11 is relatively flat at a grade less than 10% U_3O_8, with density relatively constant at 2.33 g/cm^3. At grades greater than 20%, dry bulk density increases with higher uranium grades. There are a number of strongly mineralized samples that have low dry bulk densities and vice versa, which results in significant scatter in dry bulk density values. The lower bulk density values associated with strongly mineralized samples may be attributed to the amount of clay alteration in the samples. Generally, clay alteration causes decomposition of feldspar and mafic minerals with resultant replacement by lighter clay minerals as well as loss of silica from feldspar that lowers the dry bulk density of the rock.

Denison has estimated a dry bulk density value for each grade value in the drill hole database by using the polynomial formula shown above. In RPA's opinion, the SG sampling methods and resulting data are suitable for mineral resource estimation at Phoenix.

14.3.2 Gryphon Deposit

Based on 279 dry bulk density determinations, Denison developed a formula relating bulk density to grade which was used to assign a density value to each assay. Bulk density values were used to weight grades during the resource estimation process and to convert volume to tonnage.

Denison carried out correlation analyses of the bulk density values against uranium grades (% U_3O_8) as shown in Figure 14-13. The relationship can be represented by the following polynomial formula which is based on a regression fit.

$$y = 4E\text{-}05x^2 + 0.0193x + 2.2684$$

where *y* is dry bulk density (g/cm^3) and *x* is the uranium grade in % U_3O_8. The available SG values for the assay data were reviewed and accepted by RPA and used to assign bulk density values to each sample.

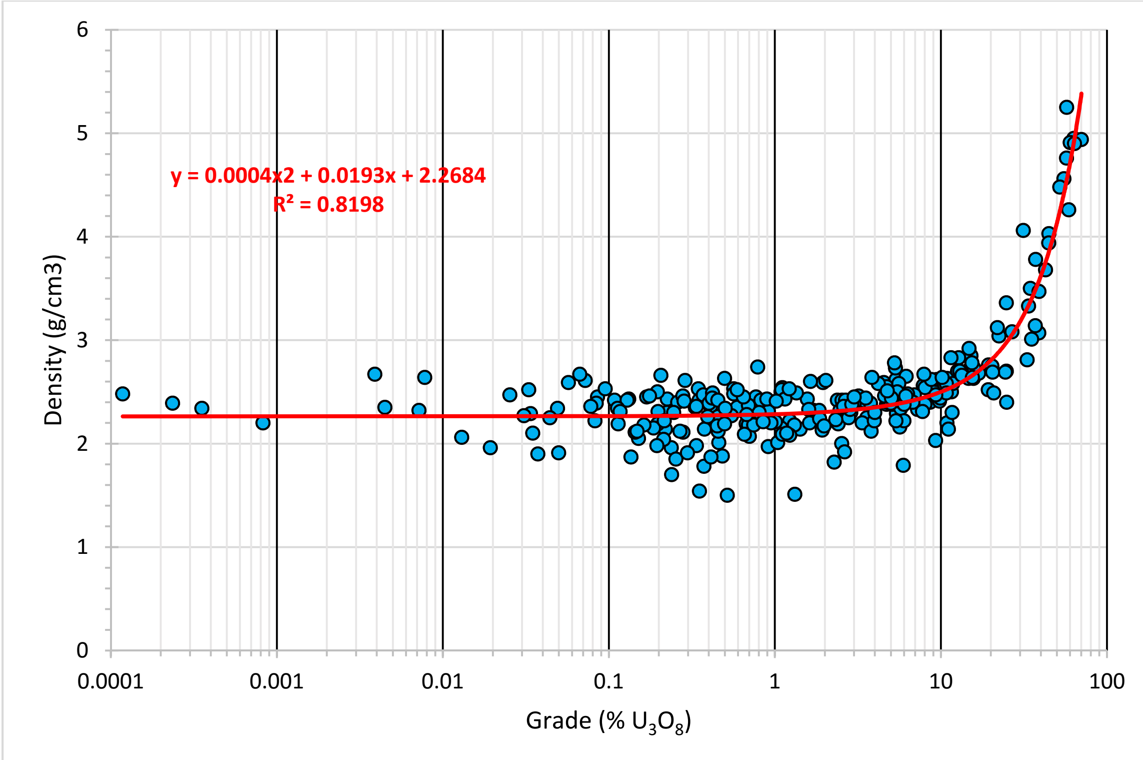


Figure 14-13. Logarithmic Plot of Dry Bulk Density vs Uranium Grade – Gryphon Deposit

Denison has estimated a dry bulk density value for each grade value in the drill hole database by using the polynomial formula shown above. In RPA's opinion, the SG sampling methods and resulting data are suitable for mineral resource estimation at Gryphon.

14.4 Statistics

14.4.1 Treatment of High-Grade Values

Where the assay distribution is skewed positively, or approaches log normal, erratic high-grade assay values can have a disproportionate effect on the average grade of a deposit. One method of treating these outliers in order to reduce their influence on the average grade is to cut or cap them at a specific grade level. In the absence of production data to calibrate the cutting level, inspection of the assay distribution can be used to estimate a first pass cutting level.

Phoenix Deposit

Although the Phoenix deposit is a high-grade uranium deposit, adequate sample support, the use of high-grade domains, and lack of apparent high-grade outliers made high-grade capping unnecessary. The influence of high-grade values, however, was restricted during the block estimation process as discussed below under interpolation parameters.

Gryphon Deposit

The mineralization wireframe models were used to code the drill hole database and to identify samples within the mineralized wireframes. These samples were extracted from the database on a group-by-group basis, subjected to statistical analyses for their respective domains, and then analyzed by means of histograms and probability plots. A total of 3,587 samples were contained within the mineralized wireframes. The sample statistics are summarized by zone in Table 14-4. The coefficient of variation (CV) is a measure of variability of the data.

Table 14-4. Descriptive Statistics of Gryphon Uranium Assay (% U_3O_8) by Domain

Wireframe	Domain	Count	Min	Max	Mean	Variance	StDev	CV
A1LG	101	784	0.00	32.00	0.59	3.75	1.94	3.30
A2	102	464	0.00	29.70	0.73	5.74	2.40	3.28
A3	103	53	0.00	4.56	0.33	0.59	0.77	2.37
A4	104	38	0.00	2.28	0.08	0.14	0.37	4.59
B1	201	470	0.00	40.50	0.64	7.00	2.65	4.15
B2	202	272	0.00	38.40	1.51	18.35	4.28	2.83
B3	203	80	0.00	36.00	1.59	25.48	5.05	3.18
B4	204	6	0.00	0.15	0.05	0.00	0.06	1.37
B5	205	19	0.00	1.00	0.11	0.05	0.23	2.08
B6	206	15	0.00	0.60	0.09	0.02	0.16	1.81
B7	207	4	0.00	0.32	0.10	0.02	0.15	1.44
C1	301	199	0.00	42.50	1.49	31.21	5.59	3.74
C4	304	19	0.00	0.40	0.10	0.02	0.13	1.27
C5	305	13	0.00	0.30	0.05	0.01	0.08	1.69
D1LG	401	363	0.00	10.80	0.39	1.19	1.09	2.82
D2	402	9	0.00	1.20	0.24	0.18	0.43	1.79
D3	403	16	0.00	9.18	1.57	9.05	3.01	1.92
D4	404	102	0.00	19.80	0.79	7.46	2.73	3.47
E1	501	143	0.00	23.10	3.09	22.90	4.79	1.55
E2	502	245	0.00	38.60	0.80	10.62	3.26	4.09
A1HG	1011	189	0.00	40.60	6.83	79.44	8.91	1.30
D1HG_HW	4011	34	0.00	19.80	4.18	32.37	5.69	1.36
D1HG_MD	4012	21	0.00	22.60	6.77	56.17	7.50	1.11
D1HG_FW	4013	29	0.00	32.40	8.12	106.40	10.32	1.27

RPA is of the opinion that the influence of high-grade uranium assays must be reduced or controlled and uses industry best practice methods to achieve this goal, including capping of high-grade values. RPA employs a number of statistical analytical methods to determine an appropriate capping value, including preparation of frequency histograms, probability plots, decile analyses, and capping curves. Using these methods, RPA examined the selected capping values for the mineralized domains in the Gryphon deposit. Examples of the capping analysis are shown in Figure 14-14 and applied to the data set for the mineralized domains.

Review of the resource assay histogram and log-normal probability plots within the wireframe domains and a visual inspection of high-grade values on vertical sections suggest cutting erratic grade values to 5% to 30% U_3O_8 which impacted 64 (1.8%) values of 3,587 assays. Capped assay statistics by zones are summarized in Table 14-5 and compared with uncapped assay statistics.

In RPA's opinion, the selected capping values are reasonable and have been correctly applied to the raw assay values for the Gryphon mineral resource estimate. The assays are capped prior to compositing.

Table 14-5. Statistics of Gryphon Capped Assays by Domain

Domain	Count	Capping		Min		Max		Mean		Variance		StDev		CV	
		Level	No.	Raw	Cap	Raw	Cap	Raw	Cap	Raw	Cap	Raw	Cap	Raw	Cap
101	784	10	5	0.00	0.00	32.00	10.00	0.59	0.52	3.75	1.54	1.94	1.24	3.30	2.38
102	464	10	6	0.00	0.00	29.70	10.00	0.73	0.64	5.74	2.89	2.40	1.70	3.28	2.66
103	53	10	0	0.00	0.00	4.56	4.56	0.33	0.33	0.59	0.59	0.77	0.77	2.37	2.37
104	38	10	0	0.00	0.00	2.28	2.28	0.08	0.08	0.14	0.14	0.37	0.37	4.59	4.59
201	470	15	4	0.00	0.00	40.50	15.00	0.64	0.57	7.00	3.54	2.65	1.88	4.15	3.32
202	272	15	8	0.00	0.00	38.40	15.00	1.51	1.32	18.35	10.33	4.28	3.21	2.83	2.43
203	80	10	3	0.00	0.00	36.00	10.00	1.59	1.07	25.48	6.18	5.05	2.49	3.18	2.33
204	6	10	0	0.00	0.00	0.15	0.15	0.05	0.05	0.00	0.00	0.06	0.06	1.37	1.37
205	19	10	0	0.00	0.00	1.00	1.00	0.11	0.11	0.05	0.05	0.23	0.23	2.08	2.08
206	15	10	0	0.00	0.00	0.60	0.60	0.09	0.09	0.02	0.02	0.16	0.16	1.81	1.81
207	4	10	0	0.00	0.00	0.32	0.32	0.10	0.10	0.02	0.02	0.15	0.15	1.44	1.44
301	199	20	4	0.00	0.00	42.50	20.00	1.49	1.18	31.21	13.54	5.59	3.68	3.74	3.13
304	19	10	0	0.00	0.00	0.40	0.40	0.10	0.10	0.02	0.02	0.13	0.13	1.27	1.27
305	13	10	0	0.00	0.00	0.30	0.30	0.05	0.05	0.01	0.01	0.08	0.08	1.69	1.69
401	363	5	4	0.00	0.00	10.80	5.00	0.39	0.36	1.19	0.76	1.09	0.87	2.82	2.45
402	9	10	0	0.00	0.00	1.20	1.20	0.24	0.24	0.18	0.18	0.43	0.43	1.79	1.79
403	16	10	0	0.00	0.00	9.18	9.18	1.57	1.57	9.05	9.05	3.01	3.01	1.92	1.92
404	102	10	3	0.00	0.00	19.80	10.00	0.79	0.63	7.46	3.44	2.73	1.85	3.47	2.93
501	143	15	7	0.00	0.00	23.10	15.00	3.09	2.90	22.90	17.42	4.79	4.17	1.55	1.44
502	245	15	3	0.00	0.00	38.60	15.00	0.80	0.68	10.62	4.93	3.26	2.22	4.09	3.27
1011	189	30	9	0.00	0.00	40.60	30.00	6.83	6.59	79.44	66.74	8.91	8.17	1.30	1.24
4011	34	20	0	0.00	0.00	19.80	19.80	4.18	4.18	32.37	32.37	5.69	5.69	1.36	1.36
4012	21	20	2	0.00	0.00	22.60	20.00	6.77	6.64	56.17	52.25	7.50	7.23	1.11	1.09
4013	29	20	6	0.00	0.00	32.40	20.00	8.12	7.04	106.40	68.53	10.32	8.28	1.27	1.18

			Capping Grade		
Decile Analysis	**Percentile**	**Uncapped**	**30**	**20**	**15**
Total Metal		675	653	592	534
Percent Metal Loss		0%	3%	12%	21%
Average Grade		6.08	5.88	5.33	4.81
CV		1.40	1.33	1.20	1.10
Capping Grade Percentile		1	0.962	0.920	0.875
Number of Caps		0	9	18	28
Metal Content	0.9	5%	5%	6%	6%
	0.91	3%	3%	3%	3%
	0.92	3%	3%	3%	3%
	0.93	3%	3%	3%	3%
	0.94	3%	4%	3%	3%
	0.95	6%	6%	5%	4%
	0.96	5%	5%	3%	3%
	0.97	5%	5%	3%	3%
	0.98	5%	5%	3%	3%
	0.99	9%	7%	5%	4%
	0.9 - 1	47%	45%	40%	34%

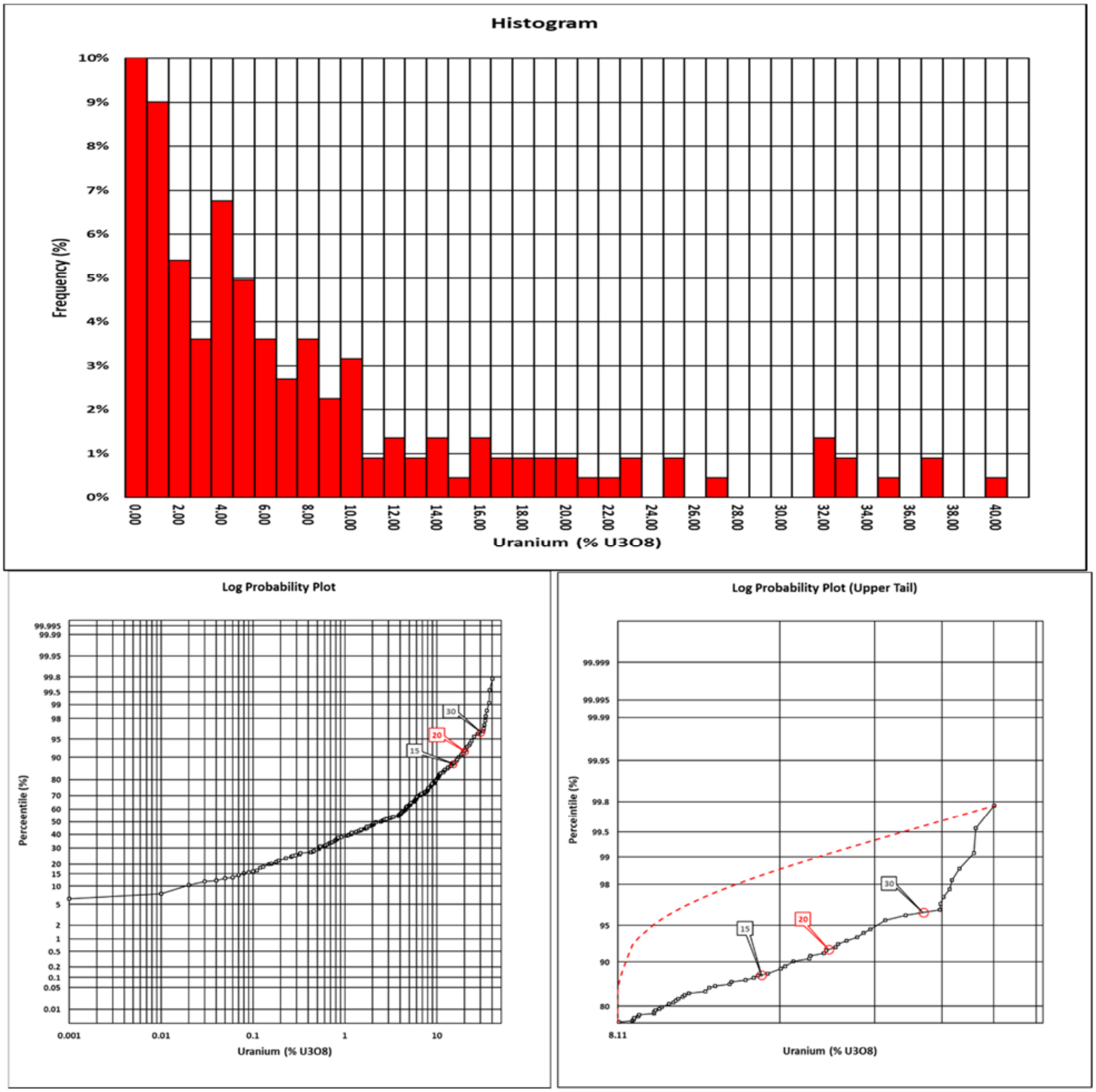


Figure 14-14. Zone A1-HG (1001) Log Normal Probability and Histogram Plot – Gryphon Deposit

14.4.2 Composites

As discussed in Sections 10 and 11, all drill core samples with chemical assays are 0.5 m long and all radiometric measurements are 0.1 m long. Radiometric measurements are used in lieu of chemical assays where core recovery is less than 80%.

Composites were created from the capped, raw assay values using the down-hole compositing function of the Vulcan modelling software package. The composite length used for interpolation was chosen considering the predominant sampling length, the minimum mining width, style of mineralization, and continuity of grade. The raw assay data contains samples having irregular sample lengths. Sample lengths range from 0.5 cm to 1.0 m within the wireframe models, with 99.1% of the samples taken at 0.5 m intervals. Given this distribution, and considering the width of the mineralization, RPA chose to composite to one-meter lengths. Assays within the wireframe domains were composited starting at the first mineralized wireframe boundary from the collar and resetting at each new wireframe boundary. Assays were capped prior to compositing. This can result in residual short composites at the bottom of the wireframes. These short composites were retained if they were from 0.5 m to 1.0 m long and were added to the previous full-length composite.

Approximately 23% of the drill holes used for the Phoenix deposit Zone A resource estimate and approximately 25% of those used for the Zone B resource estimate have radiometric measurements. No radiometric data were used in the Gryphon resource estimate.

Phoenix Deposit

Separate composite files were prepared for the Zone A HG domain, Zone A LG domain, Zone B HG domain, Zone B LG domain, and Zone A basement domain. Table 14-6 lists descriptive statistics of composite grade and GxD for each of these domains.

Figure 14-15 shows histograms of grade for each of these domains. Figure 14-16 shows grade versus density plots of these domains.

Table 14-6. Basic Statistics of Grade and GxD Composites for Phoenix Deposit Zones A and B HG and LG Domains

Statistic	Zone A Grade			Zone B Grade	
	HG	LG	BSMT	HG	LG
Mean	34.86	1.77	1.56	21.65	1.57
Standard Error	1.93	0.14	0.36	3.74	0.31
Median	31.52	0.59	0.32	17.14	0.53
Mode	#N/A	0.18	0.00	#N/A	0.25
Standard Deviation	21.62	2.69	4.26	15.85	2.64
Sample Variance	467.56	7.23	18.12	251.25	6.99
Kurtosis	-0.69	10.25	23.16	-1.02	4.65
Skewness	0.45	2.81	4.72	0.54	2.36
Range	82.31	20.13	27.66	49.24	10.86
Minimum	0.29	0.01	0.00	1.46	0.01
Maximum	82.60	20.14	27.66	50.69	10.87
Sum	4,357.3	607.7	214.9	389.7	113.0
Count	125	344	138	18	72
Coefficient of Variation	0.62	1.52	2.73	0.73	1.68

	Zone A GxD			Zone B GxD	
	HG	LG	BSMT	HG	LG
Mean	156.50	4.20	4.48	77.51	3.75
Standard Error	12.99	0.36	1.24	16.89	0.76
Median	107.54	1.36	0.88	43.68	1.24
Mode	#N/A	0.42	1.93	#N/A	0.35
Standard Deviation	145.26	6.63	14.28	71.67	6.46
Sample Variance	21,101.66	43.93	203.78	5,136.66	41.74
Kurtosis	0.77	15.12	31.86	-0.87	5.24
Skewness	1.27	3.23	5.49	0.84	2.46
Range	595.34	56.99	101.48	212.74	27.49
Minimum	0.69	0.02	0.00	3.42	0.02
Maximum	596.02	57.01	101.49	216.16	27.51
Sum	19,562.5	1,445.5	595.6	1,395.2	270.0
Count	125	344	133	18	72
Coefficient of Variation	0.93	1.58	3.19	0.92	1.72

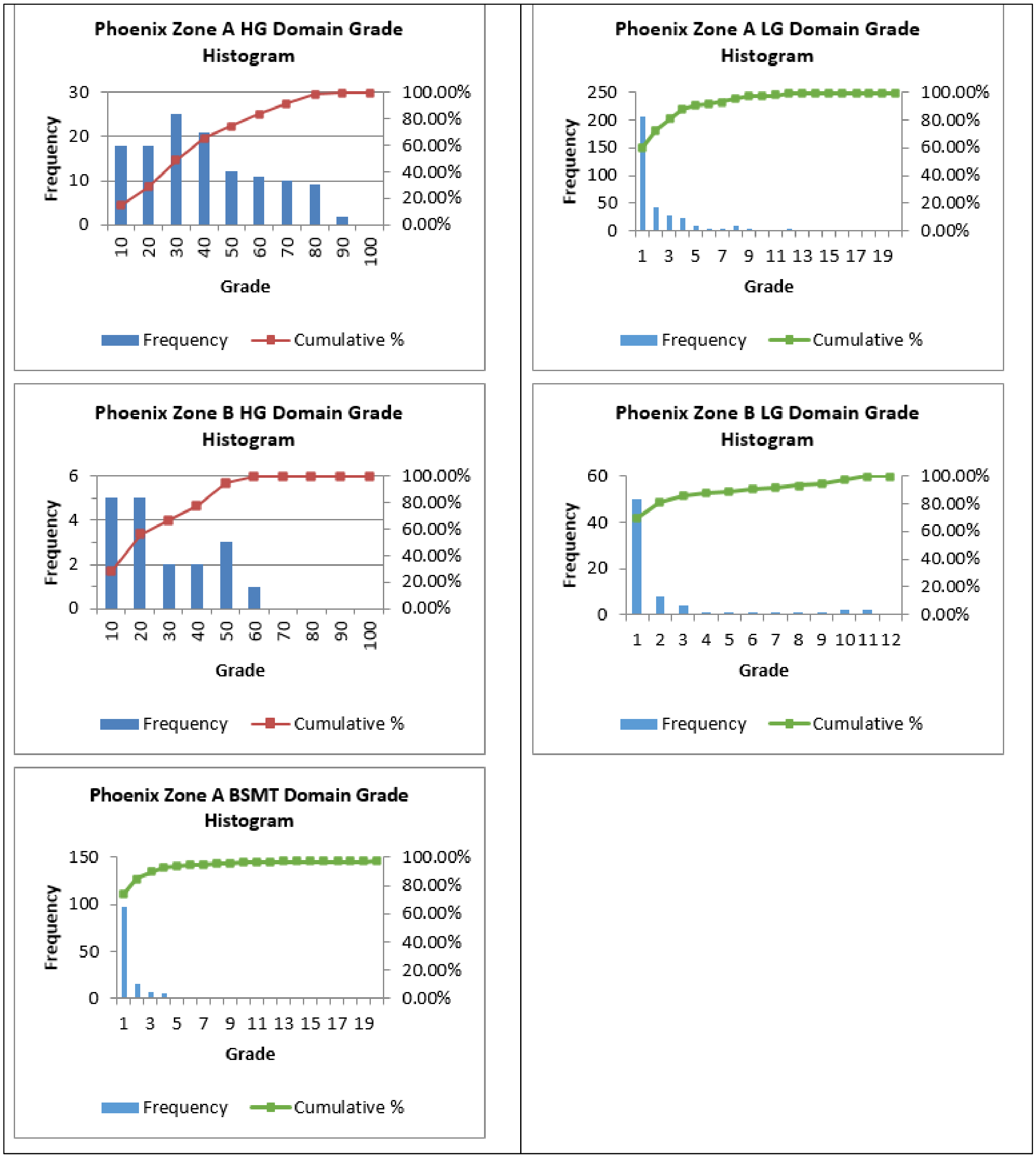


Figure 14-15. Grade Composite Histograms for Phoenix Deposit Zones A and B HG and LG Domains

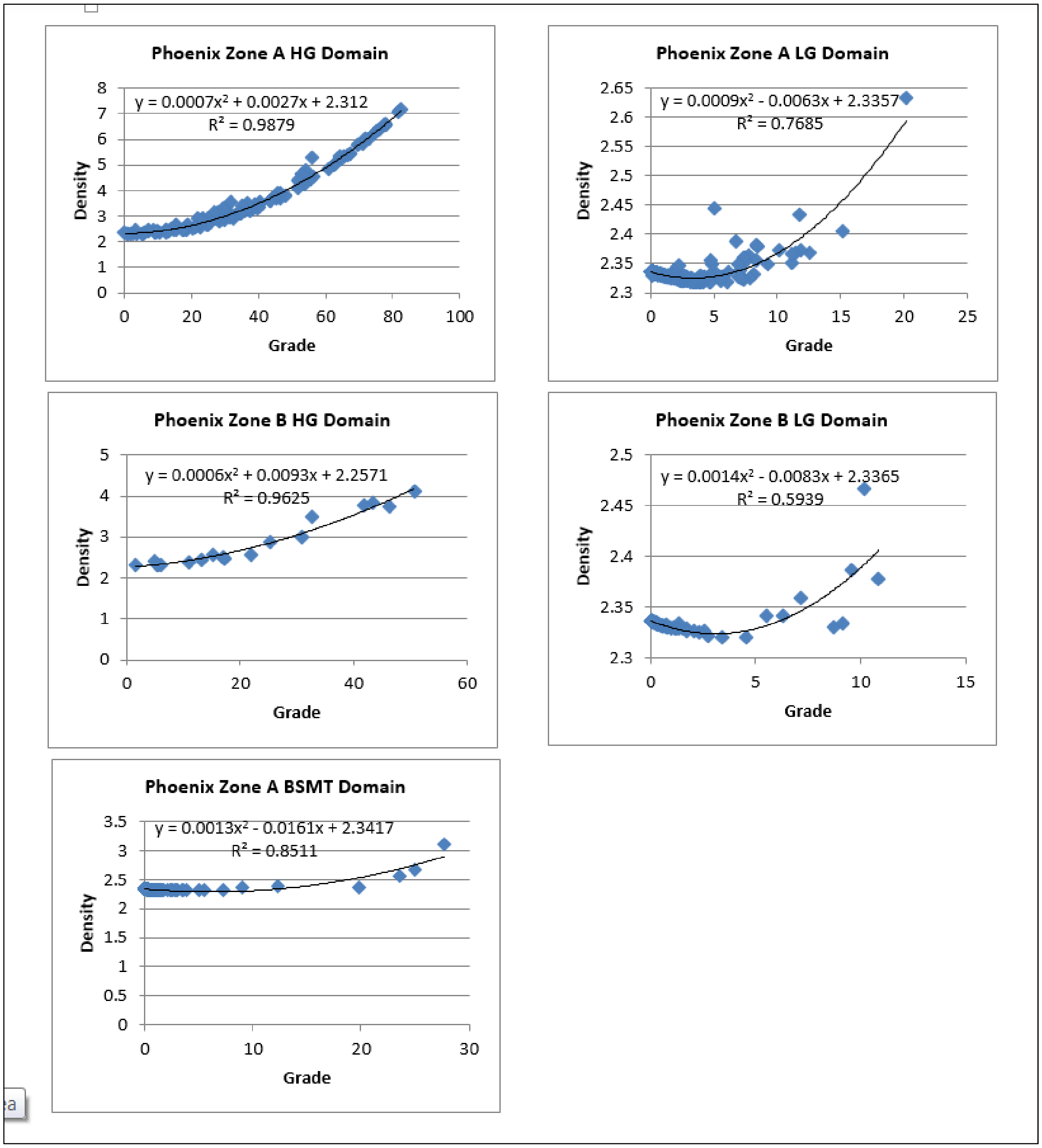


Figure 14-16. Grade vs Density Plots for Phoenix Deposit Zones A and B HG and LG Domains

Gryphon Deposit

Assays were capped prior to compositing. Table 14-7 shows the composite statistics by domain.

Table 14-7. Descriptive Statistics of Gryphon Deposit Composite Uranium Assay by Domain

Domain	Count	Min	Max	Mean	Variance	StDev	CV
101	382	0.00	7.63	0.55	1.10	1.05	1.91
102	263	0.00	9.58	0.57	1.71	1.31	2.31
103	38	0.00	2.42	0.26	0.25	0.50	1.91
104	11	0.00	0.95	0.11	0.08	0.28	2.68
201	258	0.00	10.95	0.51	2.08	1.44	2.84
202	157	0.00	11.63	1.09	5.27	2.30	2.11
203	51	0.00	9.27	0.95	3.94	1.98	2.10
204	5	0.00	0.08	0.03	0.00	0.04	1.28
205	10	0.00	0.50	0.10	0.02	0.15	1.43
206	12	0.00	0.30	0.05	0.01	0.09	1.66
207	4	0.00	0.21	0.06	0.01	0.10	1.61
301	109	0.00	19.04	1.02	8.70	2.95	2.90
304	19	0.00	0.27	0.04	0.01	0.08	1.96
305	15	0.00	0.11	0.02	0.00	0.03	1.60
401	260	0.00	5.00	0.31	0.52	0.72	2.31
402	6	0.00	0.95	0.18	0.14	0.38	2.11
403	18	0.00	7.34	1.08	4.49	2.12	1.97
404	63	0.00	9.87	0.49	2.13	1.46	2.98
501	43	0.00	9.63	1.18	5.77	2.40	2.03
502	121	0.00	10.72	0.68	3.49	1.87	2.73
1011	91	0.00	30.00	7.03	37.81	6.15	0.88
4011	15	0.07	14.87	4.60	18.90	4.35	0.95
4012	10	0.05	16.49	7.67	37.99	6.16	0.80
4013	14	0.00	19.97	6.91	37.31	6.11	0.88

14.5 Variography – Continuity Analysis

14.5.1 Phoenix Deposit

For Zone A, RPA reviewed variograms of grade and GxD for the HG domain composite data and grade for the LG domain composite data. Variograms were prepared in the down-hole direction, along a north-easterly strike direction, and horizontally across the strike direction. Variograms were of fair quality considering the limited number of composite data. The nugget effect was approximately 10% of the sill. The GxD variograms were similar to those of grade. The variograms suggested approximate ranges for the Zone A HG domain of 2.4 m down-hole, 35 m along strike, and 10 m or less across strike; and for the Zone A LG domain, 2.1 m down-hole, 25 m or less along strike, and 25 m across strike. These ranges were used to derive search ellipse dimensions for block interpolations.

14.5.2 Gryphon Deposit

Zone specific variography was undertaken, however, the number of samples is not adequate to generate meaningful variograms to derive kriging parameters.

14.6 Interpolation Parameters

14.6.1 Phoenix Deposit

Three-dimensional block models were constructed using Maptek Vulcan Mine Modelling Software. The variables G, D, and GxD were interpolated using an inverse distance squared (ID2) algorithm for each mineralized domain. Hard boundaries were employed at domain contacts, so that composites from within a given domain could not influence block grades in other domains. Table 14-8 shows the block model parameters and Table 14-9 lists the variables used.

Table 14-8. Phoenix Block Model Parameters

Model name:	**phx5_HG_zonea_u2**
History list:	phx5_HG_zonea23May2014.bhst
Format:	extended
Structure:	non-regular
Smooth:	no
Number of blocks:	1808
Number of variables:	12
Number of schemas:	1
Origin:	476,725.0 6,373,800.0 30.0
Bearing/Dip/Plunge:	52.0 0.0 0.0
Offset:	820.0 120.0 200.0
Model name:	**phx5_HG_zoneb_u2**
History list:	phx5_HG_zoneb23May2014.bhst
Format:	extended
Structure:	non-regular
Smooth:	no
Number of blocks:	324
Number of variables:	12
Number of schemas:	1
Origin:	476,725.0 6,373,800.0 30.0
Bearing/Dip/Plunge:	52.0 0. 0.0
Offset:	820.0 120.0 200.0
Model name:	**phx5_LG_zonea_u2**
History list:	phx5_LG_zonea23May2014.bhst
Format:	extended
Structure:	non-regular
Smooth:	no
Number of blocks:	5417
Number of variables:	12
Number of schemas:	1
Origin:	476,725.0 6,373,800.0 30.0
Bearing/Dip/Plunge:	52.0 0. 0.0
Offset:	820.0 120.0 200.0

Model name:	phx5_LG_zoneb_u2
History list:	phx5_LG_zoneb23May2014.bhst
Format:	extended
Structure:	non-regular
Smooth:	no
Number of blocks:	1506
Number of variables:	12
Number of schemas:	1
Origin:	476,725.0 6,373,800.0 30.0
Bearing/Dip/Plunge:	52.0 0. 0.0
Offset:	820.0 120.0 200.0

Table 14-9. Phoenix Block Model Variables

Variables	Default	Type	Description
den	-99.0	float	density
gxd_d	-99.0	float	gxd / den
gxd	-99.0	float	grade (raw) x density
grade_id2	-99.0	float	interpolated raw grade ID2
grade_ok	-99.0	double	interpolated grade ordinary kriging
nsamp	-99.0	short	number of samples per estimate
nholes	-99.0	short	number of holes per estimate
strat	unclass	name	stratigraphy
nn	-99.0	double	nearest neighbour
est_flag_id	-99.0	integer	estimation flag for ID
est_flag_ok	-99.0	integer	estimation flag for OK
ore	-99.0	integer	zones 1-13

For Zones A and B, blocks were 5 m long along the main northeast trend, 2 m wide across the main trend, and 1 m high. For the Zone A basement domain, blocks were 2 m long along the main northeast trend, 1 m wide across the main trend, and 1 m high. A whole block approach was used whereby the block was assigned to the domain where its centroid was located.

The interpolation strategy involved setting up search parameters in two passes for each domain. Search ellipses were oriented with the major axis oriented parallel to the dominant north-easterly trend of the zones. The semi-major axis was oriented horizontally, normal to the major axis (across strike) and the minor axis was vertical.

GxD and D were interpolated into the model using an initial pass. Blocks which did not receive an interpolated grade were then interpolated in the second pass, which resulted in all blocks being populated. Block grade was derived from the interpolated GxD value by dividing that value by the interpolated density value for each block. Grades not weighted by density (G) were also interpolated as a check.

In order to reduce the influence of very high-grade composites, grades greater than a designated threshold level for each domain were restricted to shorter search ellipse dimensions. If the search ellipse contained a composite greater than the specified grade, it was used for interpolation only if it fell within the restricted search ellipse. The threshold grade levels were chosen from the basic statistics and from visual inspection of the apparent continuity of very high grades within each domain.

Search parameters are listed in Table 14-10 for the Phoenix deposit Zones A and B, HG and LG domains. Major axis is horizontal along the main mineralized trend of N52°E, semi-major axis is horizontal normal to the main trend, and the minor axis is vertical.

Table 14-10. Phoenix Deposit Block Model Interpolation Parameters

Deposit and Domain	Pass	Search Radii (m)			Number of Composites Used		
		Major	Semi-major	Minor	Min	Max	Max per DH
A Deposit HG	First	35	15	8	3	8	2
	Second	50	25	10	3	8	2
	Restricted >60% U_3O_8	15	6	4	3	8	2
A Deposit LG	First	35	15	8	3	8	2
	Second	50	25	10	3	8	2
	Restricted >6% U_3O_8	15	6	4	3	8	2
A Deposit Basement	First	10	10	4	2	6	2
	Second	20	20	4	2	6	2
	Restricted >3% U_3O_8	10	10	4	2	6	2
B Deposit HG	First	35	15	6	3	8	2
	Second	50	25	10	3	8	2
	Restricted >40% U_3O_8	15	5	4	3	8	2
B Deposit LG	First	35	15	6	3	8	2
	Second	50	25	10	3	8	2
	Restricted >4% U_3O_8	15	5	4	3	8	2

Figure 14-17 is a 3D isometric view looking downward to the north at the Zone A block model with colour coded grades. Higher grades are red and green. The blocks shown are mostly in the LG domain. Figure 14-18 is an isometric view looking downward to the north at the HG domain of the Zone A block model with colour coded grades. Higher grades are red and purple.

14.6.2 Gryphon Deposit

A regular block model was created using a parent block size of 5 m (along strike) by 1 m (across strike) by 2 m (bench height) resulting in a total of 49,140,000 blocks. The model origin (lower-left corner at lowest elevation) is at UTM coordinates 475,325.0 mE, 6,376,050.0 mN and -400 m elevation. A whole block approach was used whereby the block was assigned to the domain where its centroid was located. The model fully encloses the modelled resource wireframes and the azimuth of the block model was appropriately rotated 20° so as to align with the overall strike of the mineralization within the given model area. A summary of the block model extents is provided in Table 14-11. Figure 14-19 is an isometric view looking downward to the northwest at the LG and HG domains of the A1 lens.

Table 14-11. Gryphon Block Model Parameters

Block Model Parameter		Value
X	Minimum (mE)	475,325
	Block dimension (m)	5
	Number of blocks	156
	Length (m)	780
Y	Minimum (mN)	6,376,050
	Block dimension (m)	1
	Number of blocks	630
	Length (m)	630
Z	Minimum (elev)	-400
	Block dimension (m)	2
	Number of blocks	500
	Length (m)	1,000
Rotation (Bearing/Dip/Plunge)		20°/0°/0°

A number of attributes were created to store such information as material density, estimated uranium grades, wireframe code, mineral resource classification, etc. for each block model area as listed in Table 14-12.

Table 14-12. Gryphon Block Model Variables

Variables	Default	Type	Description
class	-99.0	double	classification
den	-99.0	double	density
den_ok	-99.0	double	
gxd_d	-99.0	double	gxd / den
gxd_d_ok	-99.0	double	
gxd	-99.0	double	grade (raw) x density
gxd_ok	-99.0	double	
grade_id2	-99.0	double	interpolated raw grade ID2
grade_ok	-99.0	double	interpolated grade ordinary kriging
nsamp	-99.0	short	number of samples per estimate
nholes	-99.0	short	number of holes per estimate
strat	unclass	name	stratigraphy
nn	-99.0	double	nearest neighbour
nn_distance	-99.0	double	distance to nearest neighbour
est_flag_id	-99.0	integer	estimation flag for ID
est_flag_ok	-99.0	integer	estimation flag for OK
ore	-99.0	integer	mineralized wireframes (domains)

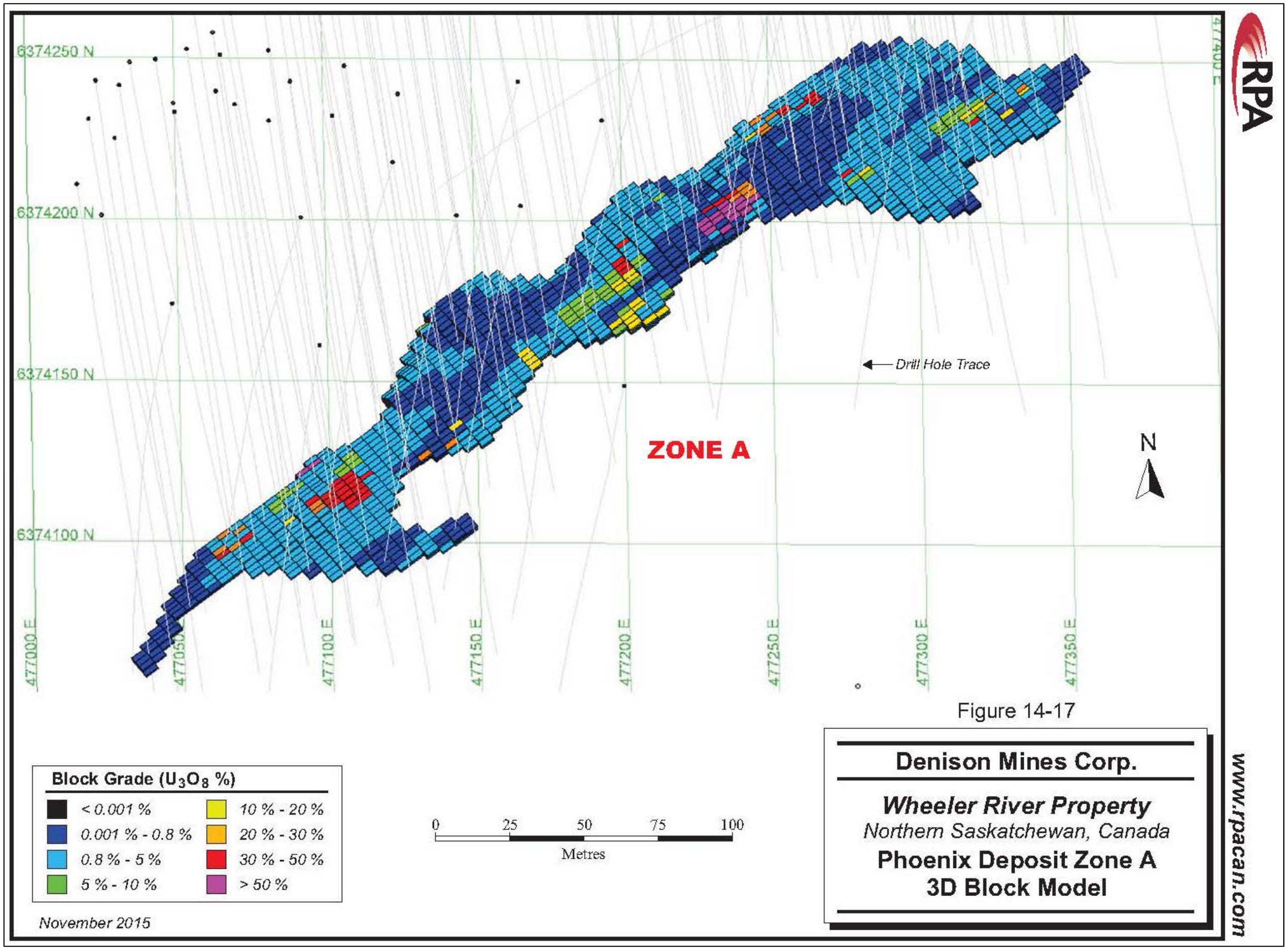


Figure 14-17. Phoenix Deposit Zone A 3D Block Model

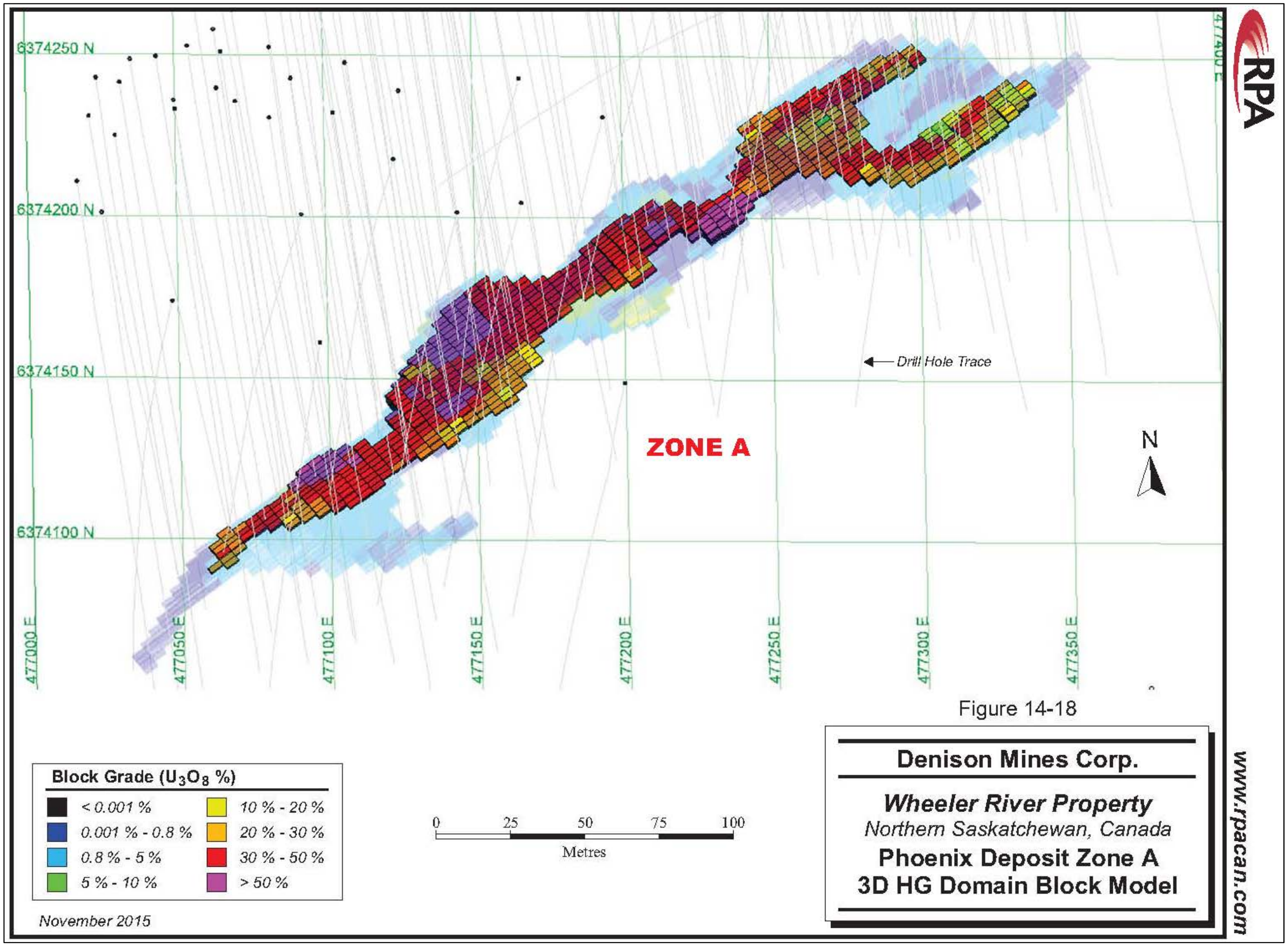


Figure 14-18. Phoenix Deposit Zone A 3D HG Domain Block Model

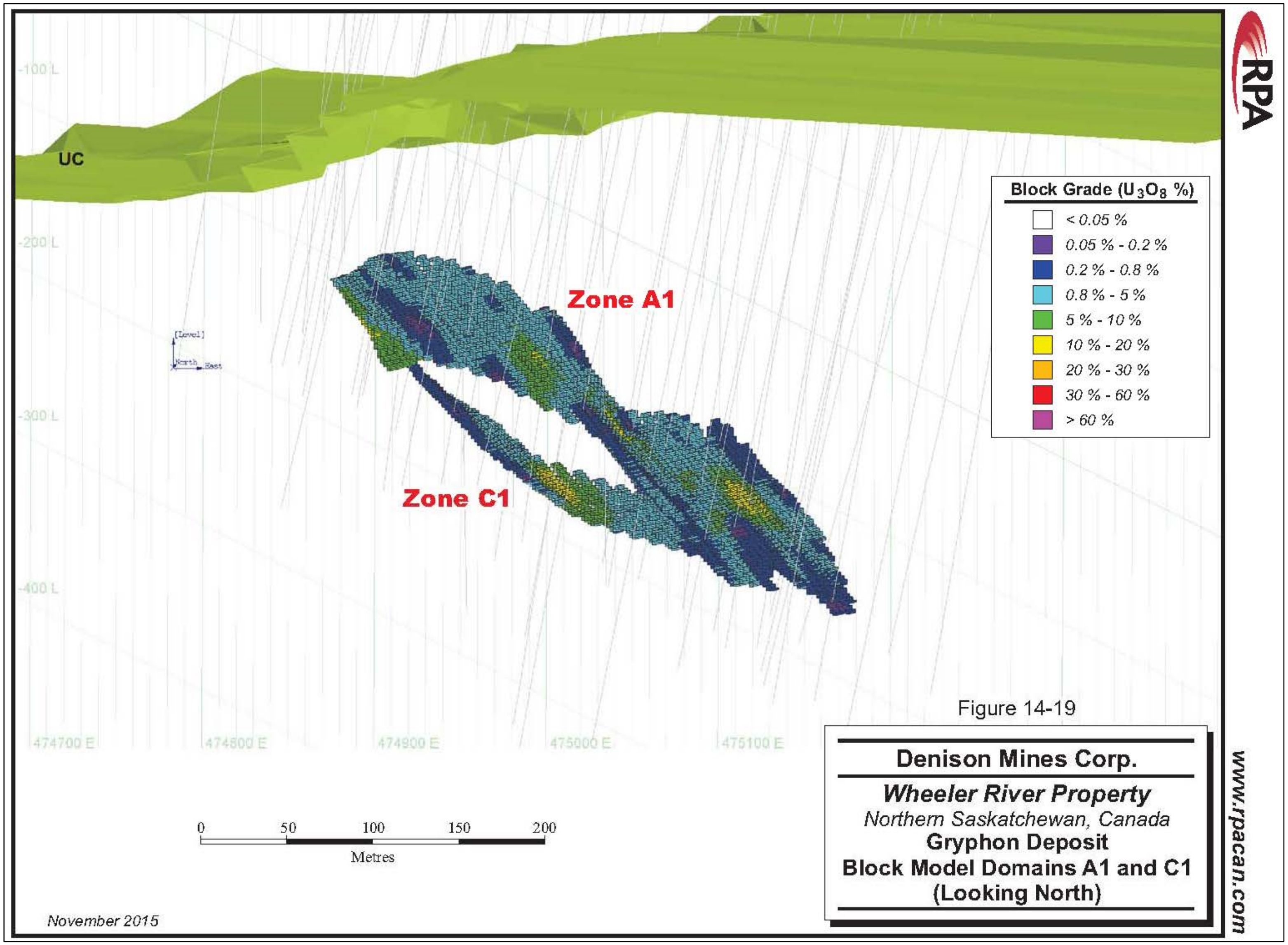


Figure 14-19. Gryphon Deposit Block Model Domains A1 and C1 (Looking North)

The block model was developed using standard methods with interpolation completed using ID^2, as variograms were not considered appropriate to derive kriging parameters.

Search ellipse dimensions were chosen following a review of drill hole spacing and interpolation efficiency. Search ellipsoid geometry of the major, semi-major, and minor axes was oriented into the plane of the mineralization with the major axis oriented at parallel to the dominant north-easterly trend of the domains. The semi-major axis was oriented vertically, normal to the major axis (down plunge), and the minor axis was oriented horizontally (across strike).

The interpolation strategy involved setting up search parameters in a series of three estimation runs for each individual domain. First, second, and third pass search ellipses maintained a 1:0.6:0.4 anisotropic ratio with the first pass search ellipse dimensions of 50 m by 30 m by 20 m. The second pass doubled the search ellipse dimensions and the third pass quadrupled the search ellipse dimensions.

For pass number one and two, the maximum number of samples to be used per drill hole was set to three, with the minimum number of samples required for an estimate set to four and a maximum number of samples required for an estimate set to nine. For pass number three, a maximum number of samples to be used per drill hole was set to three, with the minimum number of samples required for an estimate set to one and a maximum number of samples required for an estimate set to two. This process ensured that 100% of the blocks were estimated.

Density weighting was used, whereby GxD and D were interpolated into blocks using one-meter composites with 0.5 m assays capped prior to compositing. Block grade is the GxD divided by D. Grades were also interpolated as a check. Hard boundaries were used to limit the use of composites between domains.

In order to reduce the influence of very high-grade composites, grades greater than a designated threshold level for the A3-HG and other domains were restricted to 50% range of the first pass search ellipse dimension of 25 m by 25 m by 5 m (high yield restriction) within each interpolation run. The threshold grade levels of 20% for the A1-HG, 10% for B1 and C1, and 5% for E2 domains were chosen from the basic statistics and from visual inspection of the apparent continuity of very high grades within each domain, which indicated the need to limit their influence to approximately half the distance of the main search. Interpolation parameters are listed in Table 14-13 for the Gryphon deposit mineral resource domains.

Table 14-13. Gryphon Block Model Estimation Parameters

Domain	Capping	Estimation Parameters (Pass 1 / Pass 2 / Pass 3)						
		Search Ellipsoid	Max Samples per Hole	Min Samples	Max Samples	Bearing (Z)	Plunge (Y)	Dip (X)
A1LG	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	55	-40	-45
A2	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	40	-25	-45
A3	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	40	-25	-45
A4	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	25	-25	-45
B1	capped at 15% restricted at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	40	-25	-45
B2	capped at 15%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	47	-35	-30
B3	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	47	-25	-30
B4	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	40	-25	-40
B5	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	20	0	-45
B6	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	20	0	-45
B7	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	20	0	-45
C1	capped at 20% restricted at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	47	-28	-45
C4	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	47	-28	-30
C5	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	47	-28	-30
D1LG	capped at 5%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	110	-25	15

Domain	Capping	Estimation Parameters (Pass 1 / Pass 2 / Pass 3)						
		Search Ellipsoid	Max Samples per Hole	Min Samples	Max Samples	Bearing (Z)	Plunge (Y)	Dip (X)
D2	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	110	-20	0
D3	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	110	-20	-15
D4	capped at 10%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	110	-45	15
E1	capped at 15%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	110	0	-45
E2	capped at 15% restricted at 5%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	20	0	-45
A1HG	capped at 30% restricted at 20%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	20	-40	-45
D1HG_FW	capped at 20%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	55	-25	15
D1HG_MD	capped at 20%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	110	-25	15
D1HG_HW	capped at 20%	50x30x20 / 100x60x40 / 200x120x80	3	4 / 4 / 1	4 / 9 / 1	110	-25	15

14.7 Block Model Validation

The Phoenix and Gryphon deposit block models were validated by the following checks:

- Comparison of domain wireframe volumes with block volumes;
- Visual comparison of composite grades with block grades;
- Comparison of block grades with composite grades used to interpolate grades; and
- Comparison with estimation by a different method.

In RPA's opinion, block model validation is reasonable and acceptable.

14.7.1 Volume Comparison

Wireframe volumes were compared to block volumes for each domain at the Phoenix and Gryphon deposits. This comparison is summarized in Table 14-14 and results show that there is good agreement between the wireframe volumes and block model volume. The difference is less than 2%, except for the Zone B HG, D1HG_MD, and D3 domains where the difference ranges from 3.5% to 6% due to the small volume of the wireframe combined with the whole block approach.

Table 14-14. Volume Comparison for Wireframe and Blocks by Domain

Deposit and Zone	Wireframe				Block Model		% Difference
	Points	Triangles	Surface Area	Volume (m^3)	Blocks	Volume (m^3)	
Phoenix Deposit							
Zone A HG	4,965	9,926	16,732	17,999	1,808	18,080	0.45%
Zone A LG	13,313	26,682	49,758	54,270	5,416	54,160	-0.20%
Zone B HG	308	612	3,722	3,109	324	3,240	4.21%
Zone B LG	1,604	3,254	14,911	15,142	1,492	14,920	-1.47%
Zone A Basement	132	260	2,009	2,253	1,115	2,230	-1.02%
Gryphon Deposit							
A1HG	2,203	4,402	45,059	61,186	6,040	60,400	-1.28%
A1LG	5,299	10,790	178,134	229,550	23,055	230,550	0.44%
A2	3,075	6,146	136,941	189,380	18,908	189,080	-0.16%
A3	717	1,426	25,285	22,130	2,244	22,440	1.40%
A4	225	446	8,540	7,690	756	7,560	-1.69%
B1	2,988	5,972	100,639	155,368	15,505	155,050	-0.20%
B2	1,188	2,372	49,821	88,674	8,886	88,860	0.21%
B3	617	1,230	25,330	31,643	3,127	31,270	-1.18%
B4	114	224	4,754	4,700	493	4,930	4.89%
B5	218	432	7,410	8,566	862	8,620	0.63%
B6	331	658	14,907	16,468	1,618	16,180	-1.75%
B7	208	412	8,772	7,499	743	7,430	-0.92%
C1	380	756	31,658	55,514	5,552	55,520	0.01%
C4	297	590	14,953	15,283	1,554	15,540	1.68%
C5	290	576	12,564	11,505	1,161	11,610	0.91%
D1HG_HW	324	640	6,859	6,813	695	6,950	2.02%
D1HG_MD	240	472	4,579	4,284	454	4,540	5.97%
D1HG_FW	243	482	5,206	5,943	609	6,090	2.48%
D1LG	2,650	5,358	74,619	154,890	15,450	154,500	-0.25%
D2	111	218	3,561	4,519	446	4,460	-1.30%
D3	254	504	7,360	7,779	749	7,490	-3.72%
D4	1,368	2,732	46,866	66,983	6,794	67,940	1.43%
E1	396	788	8,080	15,818	1,533	15,330	-3.09%
E2	1,048	2,092	46,243	65,517	6,548	65,480	-0.06%

14.7.2 Visual Comparison

Block grades were visually compared with drill hole composites on cross-sections, longitudinal sections, and plan views. Visual validation comparing assay and composite grades to block grade estimates showed reasonable correlation with no significant overestimation or overextended influence of high grades in all domains for both the Phoenix and Gryphon deposits.

14.7.3 Statistical Comparison

Statistics of the block grades are compared with statistics of composite grades in Table 14-15 for all blocks and composites within the Phoenix and Gryphon deposit domains. Block and composite grades are weighted by density. RPA is of the opinion that the difference between the final block grades and composite grades should in general fall within ± 10%, provided that there are an appropriate number of composite values and that declustering of the data has been accounted for.

RPA is of the opinion that the composite grades appear to be reasonable and average block grades by zone are within approximately 10% of average composite grades, particularly for indicated mineral resources and for larger zones in terms of contained pounds. In many cases, larger differences are related to a low number of composite values and small wireframe volume.

Table 14-15. Statistics of Block Grades Compared to Composite Grades by Domain

Phoenix Deposit															
Domain	Count		Min		Max		Mean (Den Wt)			Variance		StDev		CV	
	Cmp	BM	Cmp	BM	Cmp	BM	Cmp	BM	% Δ	Cmp	BM	Cmp	BM	Cmp	BM
Zone A HG	125	1,808	0.29	4.62	82.60	82.38	34.86	39.18	12.4%	467.56	244.16	21.62	15.63	0.62	0.40
Zone A LG	344	5,417	0.01	0.03	20.14	19.88	1.77	1.73	-2.3%	7.23	2.98	2.69	1.72	1.52	1.00
Zone A BSMT	138	138	0.00	0.00	27.66	27.82	1.56	1.35	-13.5%	18.12	16.91	4.26	4.11	2.73	3.04
Zone B HG	18	324	1.46	3.46	50.69	48.32	21.65	25.71	18.8%	251.25	113.73	15.85	10.66	0.73	0.41
Zone B LG	72	1,506	0.01	0.01	10.87	10.49	1.57	1.34	-14.6%	6.99	2.71	2.64	1.65	1.68	1.23
Gryphon Deposit															
Domain	Count		Min		Max		Mean (Den Wt)			Variance		StDev		CV	
	Cmp	BM	Cmp	BM	Cmp	BM	Cmp	BM	% Δ	Cmp	BM	Cmp	BM	Cmp	BM
A1HG	91	6,040	0.00	0.69	30.00	26.55	7.84	7.60	-3.1%	37.81	12.17	6.15	3.49	0.88	0.46
A1LG	382	23,055	0.00	0.00	7.63	7.19	0.57	0.62	8.6%	1.10	0.57	1.05	0.75	1.91	1.23
A2	263	18,908	0.00	0.00	9.58	8.21	0.59	0.62	5.0%	1.71	0.62	1.31	0.79	2.31	1.26
A3	38	2,240	0.00	0.00	2.42	1.95	0.27	0.30	11.0%	0.25	0.05	0.50	0.23	1.91	0.78
A4	11	756	0.00	0.01	0.95	0.49	0.11	0.08	-27.7%	0.08	0.01	0.28	0.10	2.68	1.31
B1	258	15,480	0.00	0.00	10.95	7.62	0.54	0.54	-0.7%	2.08	0.80	1.44	0.90	2.84	1.67
B2	157	8,862	0.00	0.00	11.63	10.13	1.18	1.19	0.9%	5.27	1.59	2.30	1.26	2.11	1.06
B3	51	3,127	0.00	0.01	9.27	5.17	1.01	1.11	9.9%	3.94	1.03	1.98	1.01	2.10	0.92
B4	5	447	0.00	0.01	0.08	0.05	0.03	0.03	-7.1%	0.00	0.00	0.04	0.00	1.28	0.14
B5	10	862	0.00	0.01	0.50	0.34	0.10	0.15	47.0%	0.02	0.01	0.15	0.10	1.43	0.66
B6	12	1,484	0.00	0.01	0.30	0.22	0.05	0.04	-19.1%	0.01	0.00	0.09	0.03	1.66	0.73
B7	4	743	0.00	0.01	0.21	0.14	0.06	0.06	-5.8%	0.01	0.00	0.10	0.04	1.61	0.73
C1	109	5,545	0.00	0.00	19.04	16.95	1.15	1.00	-12.5%	8.70	4.11	2.95	2.03	2.90	2.02
C4	19	1,523	0.00	0.00	0.27	0.21	0.04	0.04	4.1%	0.01	0.00	0.08	0.03	1.96	0.65
C5	15	1,107	0.00	0.00	0.11	0.08	0.02	0.02	-18.9%	0.00	0.00	0.03	0.02	1.60	0.88
D1HG_HW	15	695	0.07	1.11	14.87	12.95	4.92	5.01	1.8%	18.90	4.03	4.35	2.01	0.95	0.40
D1HG_MD	10	454	0.05	1.98	16.49	16.10	8.20	7.37	-10.2%	37.99	8.91	6.16	2.99	0.80	0.41
D1HG_FW	14	609	0.00	0.79	19.97	16.19	7.61	7.52	-1.2%	37.31	4.43	6.11	2.11	0.88	0.28
D1LG	260	15,426	0.00	0.00	5.00	3.74	0.32	0.30	-7.2%	0.52	0.13	0.72	0.37	2.31	1.24
D2	6	442	0.00	0.01	0.95	0.85	0.18	0.23	25.0%	0.14	0.03	0.38	0.18	2.11	0.78
D3	18	696	0.00	0.00	7.34	5.68	1.14	1.02	-10.5%	4.49	1.50	2.12	1.23	1.97	1.20
D4	63	6,794	0.00	0.00	9.87	9.11	0.52	0.47	-9.4%	2.13	0.44	1.46	0.67	2.98	1.42
E1	43	1,532	0.00	0.00	9.63	8.77	1.26	1.16	-8.1%	5.77	1.58	2.40	1.26	2.03	1.09
E2	121	6,538	0.00	0.00	10.72	9.19	0.74	0.71	-4.3%	3.49	1.44	1.87	1.20	2.73	1.70

RPA generated swath plots for Gryphon comparing the mean block grades estimated to the data (nearest neighbour estimate) in the X, Y, and Z directions. Results indicate that estimated grades conform to the informing data (Figures 14-20 through 14-22). RPA found grade continuity to be reasonable and confirmed that the block grades were reasonably consistent with local drill hole composite grades.

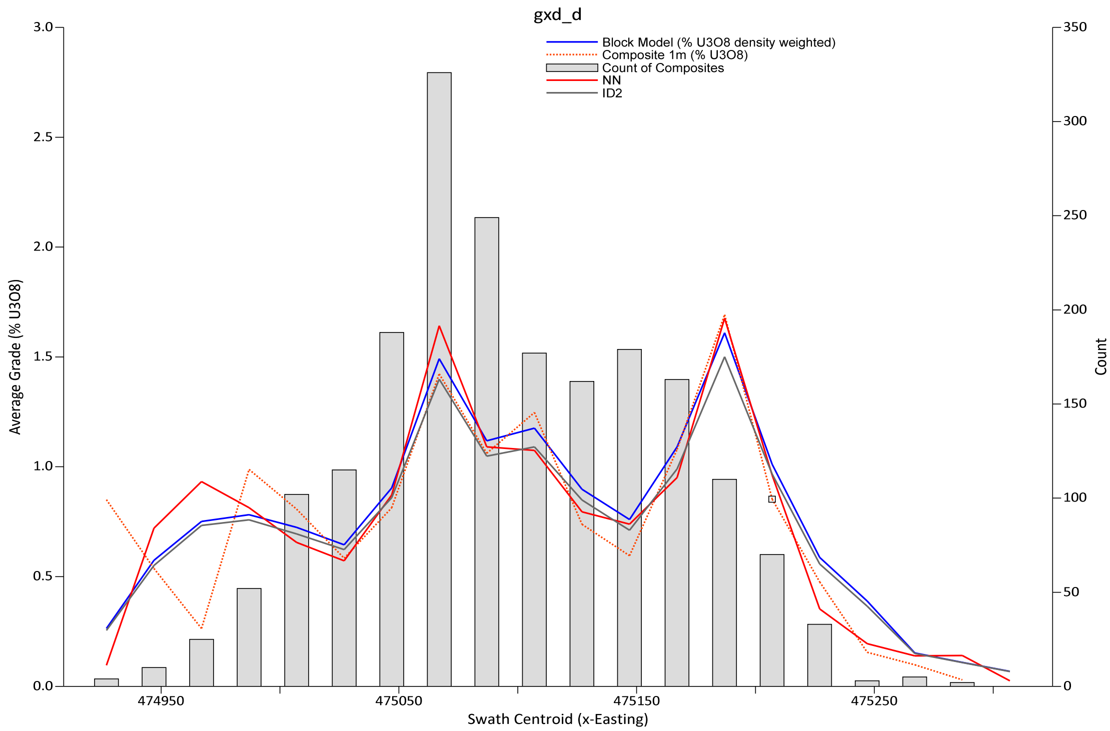


Figure 14-20. Gryphon Deposit Easting Swath Plots Comparing Block Data with Nearest Neighbour and Inverse Distance Interpolations

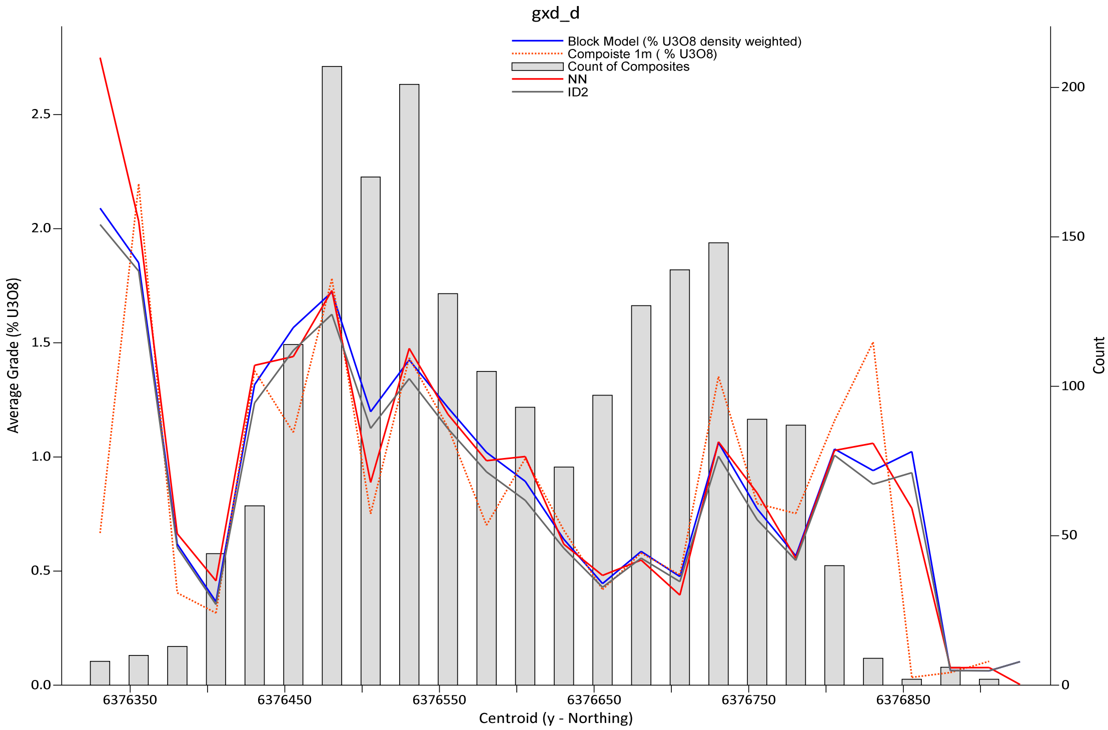


Figure 14-21. Gryphon Deposit Northing Swath Plots Comparing Block Data with Nearest Neighbour and Inverse Distance Interpolations

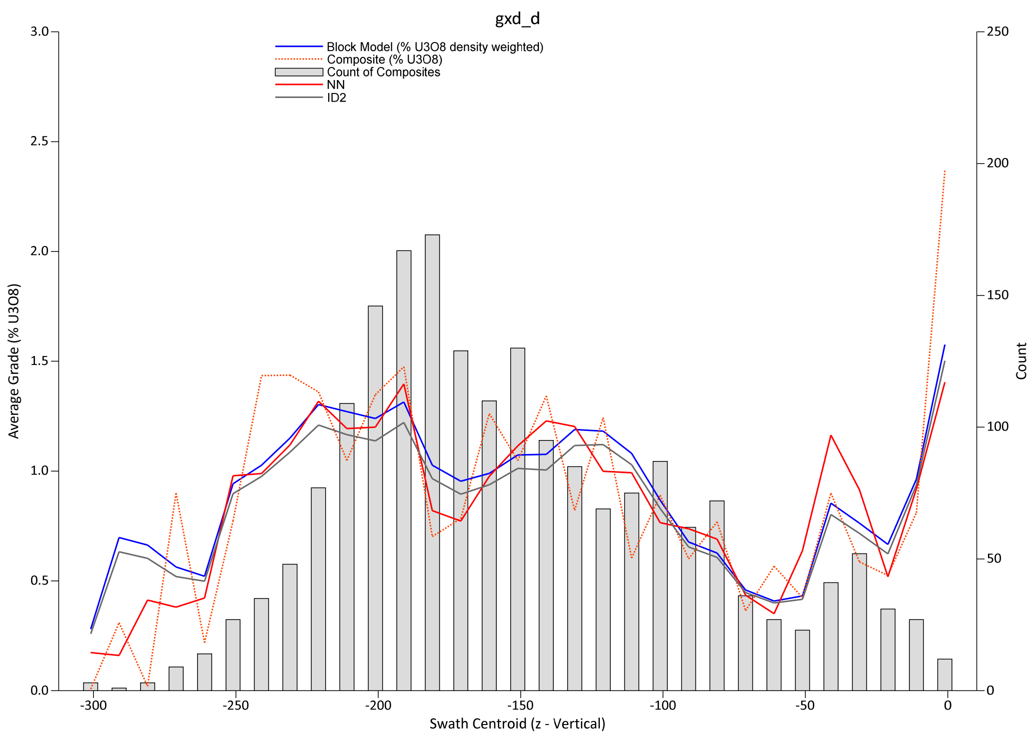


Figure 14-22. Gryphon Deposit Vertical Swath Plots Comparing Block Data with Nearest Neighbour and Inverse Distance Interpolations

14.7.4 Check by Different Estimation Methods

Phoenix Deposit

RPA has carried out check estimates of the Denison ID2 block models of the Phoenix deposit using the contour method.

For the contour method (Agnerian and Roscoe, 2002), grade times thickness times density (GxTxD) values for each drill hole intercept were plotted on plans and contoured. The areas between the contours were measured and multiplied by the average value in the contour interval. The GxTxD values are proportional to pounds of U_3O_8 per square meter and the sum of these values times area are converted to total pounds of U_3O_8 for each domain. Thickness times density (TxD) values were also plotted on plans and contoured. The areas between the contours were measured and multiplied by the average value in the contour interval. The sum of the TxD values multiplied by the area represents tonnage for each of the domains. For the contour method check on the Phoenix deposit Zone A HG domain, the tonnes, grade, and contained pounds of U_3O_8 estimated by the contour method are in the same general range as the ID^2 block model estimate.

RPA carried out check estimates of Gryphon block model using nearest neighbour and unweighted density grade estimations.

14.8 Cut-off Grade

14.8.1 Phoenix Deposit

The cut-off grade of 0.8% U_3O_8 is based on internal conceptual studies by Denison and a price of USD$50/lb U_3O_8. The HG domains are not sensitive to cut-off grades less than 5% U_3O_8 while the LG domains are quite sensitive to cut-off grade. RPA recommends that the cut-off grade should be revisited during future resource estimations on the Phoenix deposit.

Table 14-16 and Figure 14-23 show the sensitivity of the indicated mineral resource to cut-off grade. It can be seen that, although there is some sensitivity of the tonnes and grade to cut-off grade, the contained pounds of U_3O_8 are much less sensitive to cut-off grade. The cut-off grade affects essentially only the LG domains of Zones A and B because virtually all of the blocks in the HG domains of Zones A and B are above the 5% U_3O_8 cut-off grade.

Table 14-16. Phoenix Deposit Indicated Mineral Resource Sensitivity to Cut-off Grade

Cut-off % U_3O_8	Grade % U_3O_8	Tonnes	Mlbs U_3O_8
0.5	16.94	188,900	70.5
0.8	19.13	166,200	70.2
1.0	20.60	154,000	69.9
1.5	24.23	129,800	69.3
2.0	27.40	113,700	68.7
3.0	32.42	94,700	67.7
5.0	38.07	79,100	66.3

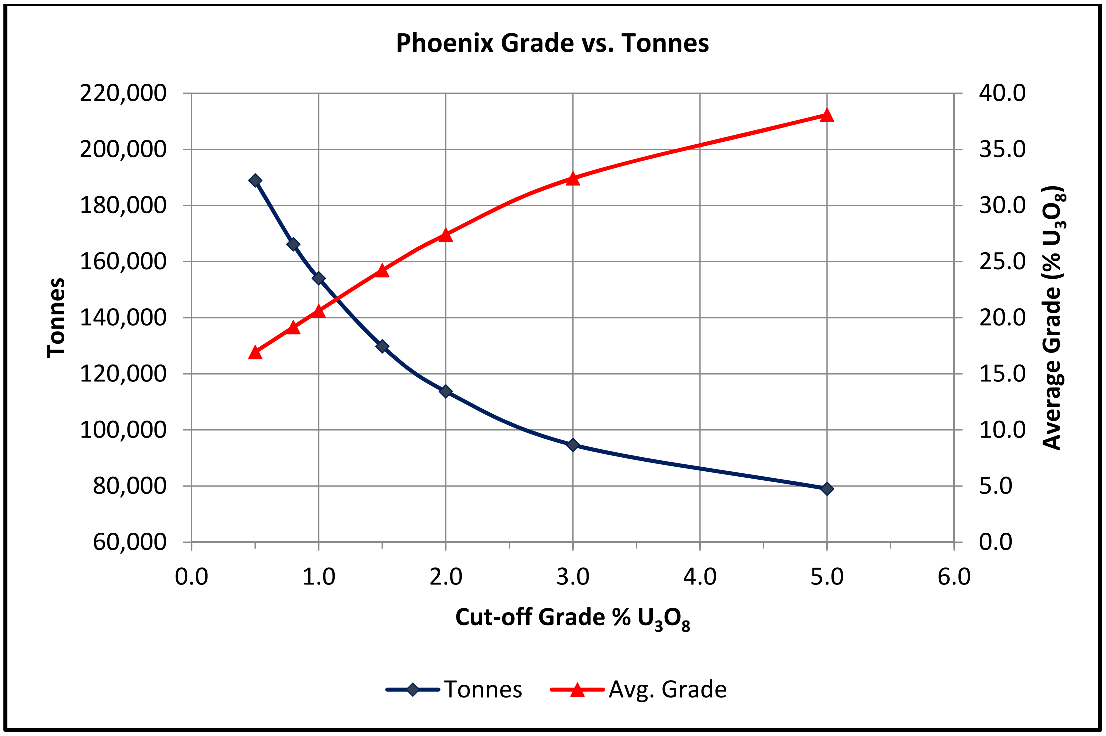


Figure 14-23. Phoenix Indicated Mineral Resource Tonnes and Grade at Various Cut-off Grades

14.8.2 Gryphon Deposit

RPA estimated a potential underground mining cut-off grade using assumptions based on historical and known operating costs on mines operating in the Athabasca Basin. Table 14-17 shows the breakeven cut-off grade estimate by RPA using a price of USD$55/lb U_3O_8 and based on assumptions for processing plant recovery, total operating cost, and incremental component of operating cost. The estimated cut-off grade of 0.2% U_3O_8 is in line with the cut-off grade of 0.2% that RPA understands is used at the Rabbit Lake mine, which is basement mineralization similar geologically to Gryphon.

Table 14-17. Gryphon Deposit Cut-off Grade Calculation

Item	Quantity
Price in USD$/lb U_3O_8	USD$55
Processing plant recovery	97%
Operating cost per tonne	CAD$546
Incremental operating cost component (60%)	CAD$260
Cut-off grade	0.2%

Table 14-18 and Figure 14-24 show the sensitivity of the Gryphon block model to various cut-off grades. RPA notes that, although there is some sensitivity of average grade and tonnes to cut-off grade, the contained pounds are less sensitive.

Table 14-18. Gryphon Deposit Inferred Mineral Resource Sensitivity to Cut-off Grade

Cut-off % U_3O_8	Grade % U_3O_8	Tonnes	Mlbs U_3O_8
0.2	1.686	1,715,573	63.8
0.4	2.231	1,234,563	60.7
0.6	2.715	965,924	57.8
0.8	3.196	780,118	55.0
1.0	3.730	633,097	52.1
1.2	4.151	546,002	50.0
1.4	4.525	482,726	48.2
1.6	4.830	438,527	46.7
1.8	5.149	398,046	45.2
2.0	5.483	360,905	43.6

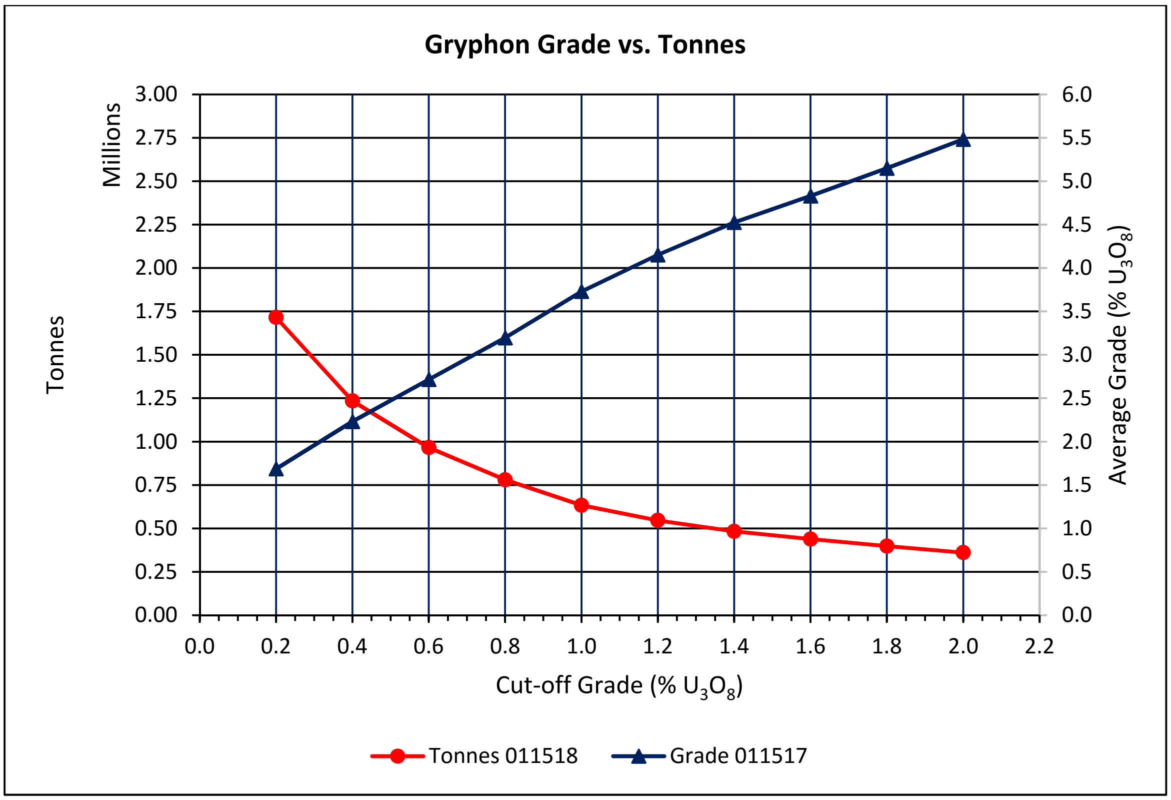


Figure 14-24. Gryphon Inferred Mineral Resource Tonnes and Grade at Various Cut-off Grades

14.9 Classification

Definitions for resource categories used in this report are consistent with those in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) and adopted by NI 43-101. As per CIM (2014), a mineral resource is defined as "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction." Mineral resources are classified into measured, indicated, and inferred categories. A mineral reserve is defined as the "economically mineable part of a measured and/or indicated mineral resource" demonstrated by studies at prefeasibility or feasibility level as appropriate. Mineral reserves are classified into proven and probable categories.

14.9.1 Phoenix Deposit

The mineral resources for the Phoenix deposit are classified as indicated and inferred based on drill hole spacing and apparent continuity of mineralization.

At Zone A, the drill hole spacing is approximately 10 m on sections spaced 25 m apart. The classification of indicated based on drill hole density and good grade continuity along strike is appropriate in RPA's opinion for all of the LG and HG domains. The Zone A basement domain is classified as inferred because of uncertainty of grade continuity due to the small number of drill holes.

At Zone B, the drill hole spacing is approximately 10 m on sections spaced 25 m apart. The classification of indicated is appropriate in RPA's opinion for most of the LG and HG domains. In the northeastern part of Zone B, drill hole sections are spaced at approximately 35 m and the most north-easterly drill hole does not correlate well spatially with other drill holes because its elevation is slightly lower. This part of Zone B is classified as inferred because there is some uncertainty in the continuity of grade in both the HG and LG domains. Figure 14-25 shows the area of inferred mineral resources along with indicated mineral resources at Zone B.

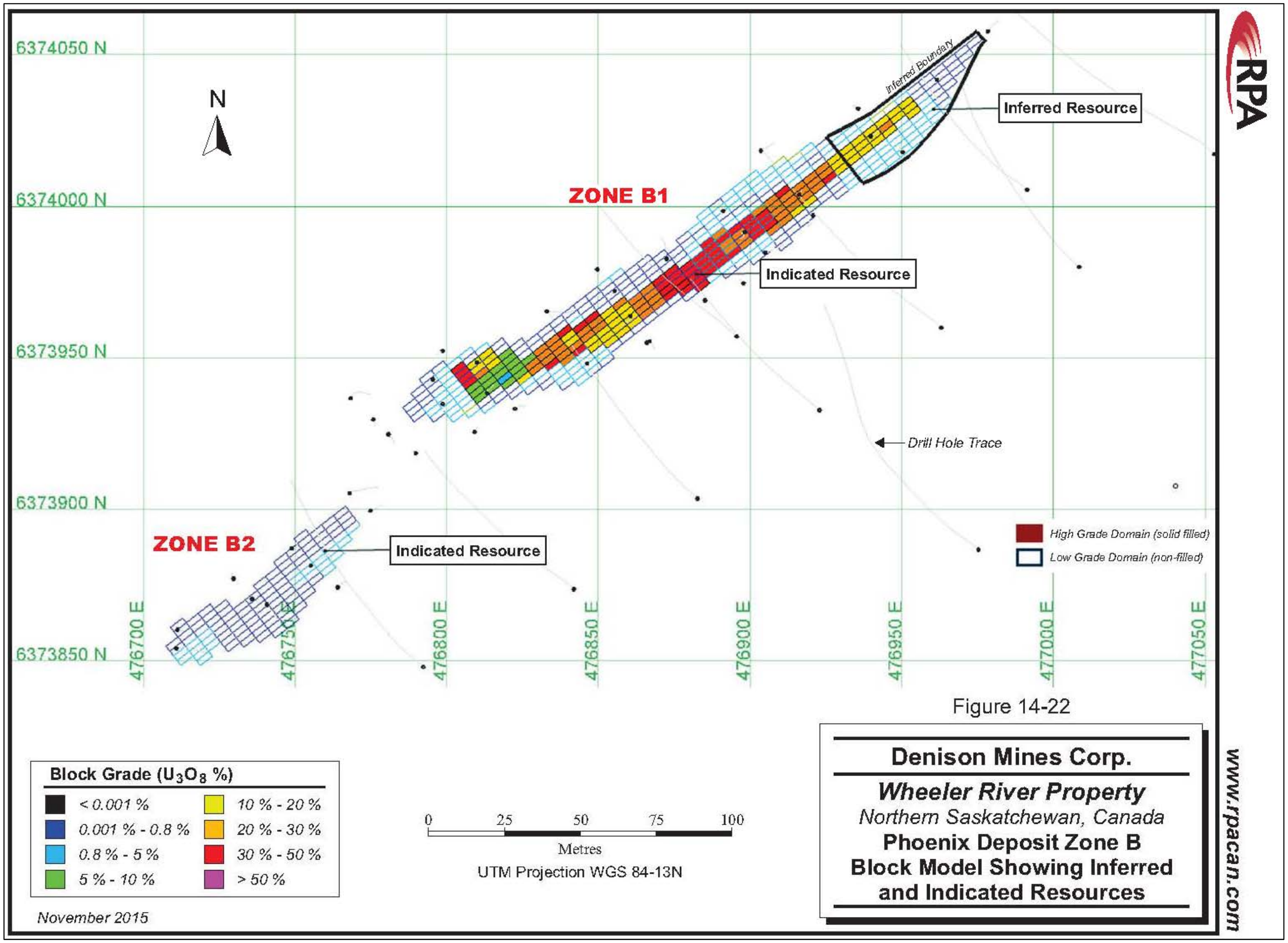


Figure 14-25. Phoenix Deposit Zone B Block Model Showing Inferred and Indicated Resources

14.9.2 Gryphon Deposit

CIM definitions were followed for mineral resource classification at Gryphon. The mineralized material for each domain was classified into the indicated or inferred mineral resource category on the basis confidence in the geological interpretation, the demonstrated continuity of the mineralized structures, and the drill hole spacing.

Mineral resources for the Gryphon deposit are classified into indicated and inferred categories based on the following parameters:

- Indicated Mineral Resources: Defined by 25 m by 25 m drill spacing and a nearest neighbour distance of ≤ 20 m with strong geological continuity between drill hole intercepts.
- Inferred Mineral Resources: Defined by drill spacing that is greater than 25 m by 25 m and a nearest neighbour distance of ≤ 100 m with reasonable continuity assumed between holes. Due to the uncertainty that may be attached to inferred mineral resources, it cannot be assumed that all or any part of an inferred mineral resource will be upgraded to an indicated or measured mineral resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability.

Figure 14-26 and Table 14-19 show the statistical distribution of the indicated and inferred categories based on distance to the nearest neighbour.

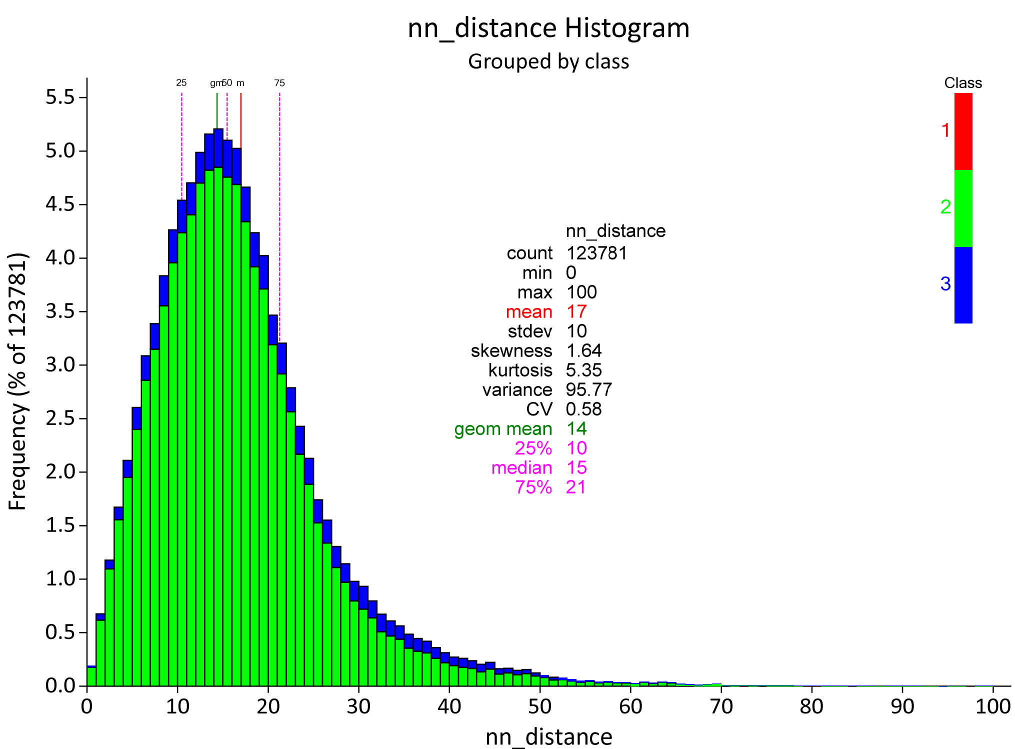


Figure 14-26. Histogram Classification of Gryphon Deposit based on Nearest Neighbour Distance (Class: 1=Measured, 2=Indicated, and 3=Inferred)

Table 14-19. Gryphon Histogram Summary Statistics of NN Distance vs Classification

Classification	Domain	Count	Min (m)	Max (m)	Mean (m)	Variance	StDev (m)	CV
Indicated	2	112,194	0.00	99.00	16.00	85.66	9.00	0.56
Inferred	3	11,587	0.00	100.00	22.00	169.00	13.00	0.60

14.10 Mineral Resource Estimate

Table 14-20 lists the mineral resource estimate for the Wheeler River property by domain and resource category. The effective date of the resource estimate is January 30, 2018. The Phoenix cut-off grade of 0.8% U_3O_8 is based on internal conceptual studies by Denison and a price of USD$50/lb U_3O_8, while a cut-off grade of 0.2% U_3O_8 for Gryphon is based on RPA estimates using assumptions based on historical and known mining costs on mines operating in the Athabasca Basin at a price of USD$55/lb U_3O_8.

For the Phoenix and Gryphon deposits, total indicated mineral resources are estimated at 1,809,000 tonnes at an average grade of 3.3% U_3O_8 containing 132.1 million pounds of U_3O_8. Total inferred mineral resources are estimated at 82,000 tonnes at an average grade of 1.7% U_3O_8 containing 3.0 million pounds of U_3O_8.

In RPA's opinion, the estimation methodology is consistent with standard industry practice and the Wheeler River property mineral resource estimate is considered to be reasonable and acceptable.

Table 14-20. RPA Mineral Resource Estimate - Wheeler River Project – January 30, 2018

Category	Deposit and Domain	Tonnes	Grade (% U_3O_8)	Million lb U_3O_8
Indicated	Gryphon A1HG	148,000	7.6	24.7
Indicated	Gryphon A1LG	365,000	0.8	6.7
Indicated	Gryphon A2	262,000	1.0	5.5
Indicated	Gryphon A3	36,000	0.4	0.3
Indicated	Gryphon B1	161,000	1.1	3.7
Indicated	Gryphon B2	158,000	1.5	5.2
Indicated	Gryphon B3	59,000	1.3	1.7
Indicated	Gryphon C1	105,000	1.2	2.7
Indicated	Gryphon D1HG_HW	17,000	5.0	1.8
Indicated	Gryphon D1HG_MD	11,000	7.4	1.8
Indicated	Gryphon D1HG_FW	15,000	7.5	2.5
Indicated	Gryphon D1LG	153,000	0.6	1.9
Indicated	Gryphon D4	89,000	0.7	1.4
Indicated	Gryphon E2	65,000	1.1	1.7
Indicated	Phoenix Zone A HG	62,900	43.2	59.9
Indicated	Phoenix Zone A LG	84,300	2.4	4.4
Indicated	Phoenix Zone B HG	8,500	28.0	5.2
Indicated	Phoenix Zone B LG	10,700	2.9	0.7
Subtotal Indicated	Gryphon	1,643,000	1.7	61.9
Subtotal Indicated	Phoenix	166,000	19.1	70.2
Total Indicated		1,809,000	3.3	132.1

Category	Deposit and Domain	Tonnes	Grade (% U_3O_8)	Million lb U_3O_8
Inferred	Gryphon A4	2,000	0.3	0.0
Inferred	Gryphon B5	10,000	0.3	0.1
Inferred	Gryphon D2	5,000	0.4	0.0
Inferred	Gryphon D3	13,000	1.2	0.4
Inferred	Gryphon E1	31,000	1.3	0.9
Inferred	Gryphon E2	12,000	2.0	0.5
Inferred	Phoenix Zone A HG	0	0.0	0.0
Inferred	Phoenix Zone B HG	1,000	14.5	0.2
Inferred	Phoenix Zone B LG	5,000	1.8	0.2
Inferred	Phoenix Zone A Basement	3,000	10.2	0.7
Subtotal Inferred	Gryphon	73,000	1.2	1.9
Subtotal Inferred	Phoenix	9,000	5.8	1.1
Total Inferred		82,000	1.7	3.0

Notes:

1. CIM Definitions (2014) were followed for classification of mineral resources.
2. Mineral resources for the Gryphon deposit are estimated at an incremental cut-off grade of 0.2% U_3O_8 using a long-term uranium price of USD$50 per lb, and a USD$/CAD$ exchange rate of 0.75. The cut-off grade is based on incremental operating costs for low-grade material.
3. Mineral resources for the Phoenix deposit are reported above a cut-off grade of 0.8% U_3O_8. Mineral resources for the Phoenix deposit were last estimated in 2014 to reflect the expansion of the high-grade zone. As no new drilling has been completed at Phoenix since that time, the mineral resource estimates for the Phoenix deposit remain current.
4. High-grade mineralization was capped at 30% U_3O_8 and restricted at 20% U_3O_8 for the A1HG and capped at 20% $U_3O_8$for the D1HG with no search restrictions.
5. Low-grade mineralization was capped at 20% U_3O_8 for the C1 domain with search restrictions applied to U_3O_8 grades greater than or equal to 10.0% U_3O_8.
6. Low-grade mineralization was capped at 15% U_3O_8 for the B1, B2, E1, and E2 domains with search restrictions applied to U_3O_8 grades greater than or equal to 10.0% U_3O_8 for the B1 domain and 5.0% U_3O_8 for the E2 domain.
7. Low-grade mineralization was capped at 10% U_3O_8 for the A1-A4, B3-B7, C4-C5, and D2-D4 domains with no search restrictions.
8. Low-grade mineralization was capped at 5% U_3O_8 for the D1 domain with no search restriction.
9. Bulk density is derived from grade using a formula based on 196 measurements from Phoenix and 279 measurements from Gryphon.
10. A minimum mining width of 2 meters was used.
11. Numbers may not add due to rounding.

15 Mineral Reserve Estimates

15.1 Introduction

A mineral reserve is defined by the CIM within the CIM Definition Standards on Mineral Resources and Mineral Reserves, as adopted by CIM Council on 10 May 2014, as follows:

"A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Preliminary Feasibility or Feasibility level as appropriate that includes application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could be reasonably justified."

The CIM guidelines require that only material categorized as measured or indicated resources be considered for potential mineral reserves.

Table 15-1 summarizes the mineral reserve estimate, of which Denison's share is 63.3%. The effective date of the mineral reserve estimate is September 1, 2018. Details of the estimation methodology follow.

Table 15-1. Mineral Reserve Estimate – Wheeler River Project – September 1, 2018

Deposit	Category	Tonnes	Grade (% U_3O_8)	Million lbs U_3O_8 (100% Basis)	Million lbs U_3O_8 (Denison 63.3%)
Phoenix	Probable	141,000	19.1	59.7	37.8
Gryphon	Probable	1,257,000	1.8	49.7	31.5
Total		**1,399,000**	**3.5**	**109.4**	**69.2**

Notes:

1. CIM definitions (2014) were followed for classification of mineral reserves.
2. Mineral Reserves are stated at a process plant feed reference point.
3. Mineral reserves for the Phoenix deposit are reported at the mineral resource cut-off grade of 0.8% U3O8. The mineral reserves are based on the block model generated for the May 28, 2014 mineral resource estimate. A mining recovery factor of 85% has been applied to the mineral resource above the cut-off grade.
4. Mineral reserves for the Gryphon deposit are estimated at a cut-off grade of 0.58% U3O8 using a long-term uranium price of USD$40/lb, and a USD$/CAD$ exchange rate of 0.80. The mineral reserves are based on the block model generated for the January 30, 2018 mineral resource estimate. The cut-off grade is based on an operating cost of CAD$574/tonne, milling recovery of 97%, and 7.25% fee for Saskatchewan royalties. Mineral reserves include for diluting material and mining losses.
5. Numbers may not add due to rounding

15.2 Mineral Reserve Estimate – Phoenix

15.2.1 Summary

The mineral reserve for Phoenix is estimated at 59.7 million pounds of U_3O_8 with an average grade of 19.1% over 141,000 tonnes as summarized in Table 15-2. The mineral reserve was prepared by Woodard & Curran Inc. based on the mineral resources prepared by RPA. The ISR process has been designed to a level appropriate for a PFS. The mineral reserve estimate stated herein is consistent with CIM definitions and is suitable for public reporting. As such, the mineral reserves can only be based on measured and indicated mineral resources and cannot include any inferred mineral resources. The Phoenix mineral resource does not include any measured resource material. Indicated resources are converted directly to probable reserves.

Table 15-2. Mineral Reserve Estimate - Phoenix

Category	Million lbs U_3O_8	Grade	Tonnes
Proven	0	0	0
Probable	59.7	19.1%	141,000
TOTAL	**59.7**	**19.1%**	**141,000**

Notes:
1. CIM definitions (2014) were followed for classification of mineral reserves.
2. Mineral reserves are stated at a processing plant feed reference point
3. Mineral reserves for the Phoenix deposit are reported at the mineral resource cut-off grade of 0.8% U_3O_8. The mineral reserves are based on the block model generated for the May 28, 2014 mineral resource estimate. A mining recovery factor of 85% has been applied to the mineral resource above the cut-off grade.

For the purposes of combining the mineral reserves at the project into one estimate, the Phoenix mineral reserve estimate of 59.7 million pounds of U_3O_8 has been converted to an equivalent conventional reserve with both quantity (tonnes) and quality (grade) based on the mineral resource at the cut-off value of 0.8% U_3O_8 and an application of an 85% mining recovery.

15.2.2 Mining Recovery

Mining recovery has been included in the mineral reserve estimate and was determined through comprehensive laboratory tests on core samples representative through the deposit. Results from typical ISR tests, agitation leach tests, and column leach tests gave guidance to the recoverability of the deposit. These results were considered against benchmarked ISR operation recoveries within the context of comparing the grade distribution and geometry between the typical low-grade and dispersed roll front deposit benchmarked operations and the high-grade and concentrated Phoenix deposit. Based on this analysis, an overall mining recovery factor of 85% has been applied against the in situ resource.

The Phoenix orebody is well defined and well understood, geologically. It has been drilled to 10 m by 25 m spacings across the orebody. The genesis and structural complexity of the orebody are well understood as well. There are no outlying elements of the orebody requiring further drill interrogation. No inferred resources were included in the evaluation. For these reasons, no reductions in mineable reserves have been made due to geological understanding.

Surface expression of the orebody is free of incumbrancers and access to the wellfield has no physical barriers. The orebody has been described previously as small, compact, and well understood, with no outlying smaller areas, and the entire orebody can readily be targeted with ISR wells. No reduction in reserves were made for these reasons.

15.2.3 Geology and Hydrogeology

Ground conditions throughout the Phoenix deposit are highly variable and fractured. Initially, the lixiviant will be expected to dissolve mineralization rather quickly along the higher permeable zones. As the orebody is dissolved, it is expected that permeability will increase, allowing the lixiviant to contact continually expanding areas of the ore zone. In zones of lower permeability, a longer residence time may be required to allow for the lixiviant to contact less permeable areas. Reserve reduction due to accessibility of ore to the leaching solution has been incorporated into the 85% mining recovery factor.

Successful ISR operations require that the orebody be below an aquitard to prevent migration of leach fluids into the overlaying rock masses, causing both environmental contamination as well as loss of metal and subsequent production. The Phoenix orebody does not benefit from the presence of such a lithological unit above it. For this reason, an extensive ground freezing program has been designed and factored into the project planning. This freeze curtain will anchor into the underlaying basement rock in all directions around the orebody, keeping fluids contained and restricting contamination of the Athabasca sandstones. The presence of a program of this design has eliminated the need to remove any reserves due to this factor.

15.2.4 Metallurgy

Extensive metallurgical testing that has been completed on the Phoenix orebody (both conventional leach tests and ISR tests) demonstrates that the ore is amenable to leaching via acid leach with no tested part of the orebody demonstrating lower amenability factors. Thus, no additional reductions in reserves were included for leaching.

Metallurgical laboratory test work demonstrated that precipitation recoveries of 98.5% were achievable. Although this metallurgical recovery was considered in determining the applicability of ISR to the Phoenix deposit, it was not included in the mineral reserve estimate quantities as it is stated at a process plant feed reference point. It is included in the economic analysis.

15.2.5 Cut-off Grade

The mineral reserve estimate uses the same cut-off grade as the mineral resource estimate, 0.8% U_3O_8. This was considered prudent given the compact geometry of the Phoenix deposit and that ISR is not a selective mining method.

Benchmarked ISR projects use a cut-off measured against grade thickness, in units of %U_3O_8-feet. Presently, a cut-off grade thickness of 0.3%U_3O_8-feet is commonly used. This value is calculated by multiplying the vertical thickness of the deposit and the average grade of the deposit across that vertical section. In comparison to the Phoenix deposit, this cut-off value is low.

The mineral reserve cut-off grade of 0.8% at Phoenix, using the smallest vertical geological unit in the mineral resource block model of 1.0 m, equates to a cut-off grade thickness of 2.6%U_3O_8-feet. Note that the average grade thickness of the Phoenix deposit is approximately 150%U_3O_8-feet. Similar to the mineral resource cut-off grade description in Section 14.8.1, the in situ quantity of U_3O_8 is not sensitive to lower cut-off grade thicknesses less than 15% U_3O_8-feet.

At cut-off grades in the range of 0.8% U_3O_8, the Phoenix orebody is insensitive to cut-off grade variability, due to the high-grade nature of this deposit.

15.3 Mineral Reserve Estimate – Gryphon

15.3.1 Summary

The mineral reserve for Gryphon is estimated at 49.7M lbs U_3O_8 over 1.26M tonnes grading 1.8% U_3O_8, as summarized in Table 15-3. The mineral reserve was prepared by Stantec. The resource block model was prepared by RPA.

The mine design and mineral reserve estimate have been completed to a level appropriate for a PFS. The mineral reserve estimate stated herein is consistent with CIM definitions and is suitable for public reporting. As such, the mineral reserves are based on measured and indicated mineral resources, and do not include any inferred mineral resources. The Gryphon block model did not include any measured resource material. Indicated resources are converted directly to probable reserves. The inferred resources contained within the mine design are classified as waste.

Table 15-3. Mineral Reserve Estimate - Gryphon

Category	Tonnes	Grade (% U_3O_8)	Million lbs U_3O_8
Proven	0	0.0	0
Probable	1,257,000	1.8	49.7
TOTAL	**1,257,000**	**1.8**	**49.7**

Notes:
1. CIM definitions (2014) were followed for classification of mineral reserves.
2. Mineral reserves are stated at a processing plant feed reference point
3. Mineral reserves for the Gryphon deposit are estimated at a cut-off grade of 0.58% U3O8 using a long-term uranium price of USD$50/lb and a USD$/CAD$ exchange rate of 0.80. The mineral reserves are based on an operating cost of $574/tonne, milling recovery of 97%, and 7.25% fee for Saskatchewan royalties. Mineral reserves include for diluting material and mining losses.

15.3.2 Mining Recovery

The mining recovery used in the mineral reserve estimate was determined through two means:

- Planned mining recovery (recovery by design); and
- Mining recovery (recovery due to operational losses).

As part of automatic generation of stopes shapes, an initial mining recovery was established that excludes the portion of block model cells which are located outside of the designed stope and development shapes. A mining recovery factor of 95% (based on benchmark mining recovery data) is applied on the tonnage of material reported inside the designed shapes.

15.3.3 Mining Dilution

The dilution used in the mineral reserves estimation was incorporated through two means:

- Internal dilution (dilution by design); and
- External dilution (dilution by over break and by loading/mucking).

Internal dilution refers to the part of low-grade and/or waste material incorporated during the stope design process, in which portions of waste blocks were incorporated inside the shape of stopes. This material is considered as dilution sourced by the design process. The average internal dilution for Gryphon is 39.5%.

Geometry of the mineralized zone, mining method applied, operational conditions, geomechanical characterization, and presence of water contributes to the amount of over break material outside of the designed stopes limits that will be mined with the stope. This material is considered as dilution by over break. Wireframe shells were produced to estimate the grade of the over break material.

After the blasting process and as part of loading and hauling, some material from external sources is incorporated with the mineralized rock. This material is considered as dilution by loading and is mainly sourced from backfill adjacent to blasted rock and from shotcrete located in the undercuts. This combined dilution is also called external dilution and is estimated at 20.0% for Gryphon with a grade of 0.43% U_3O_8 (a combination of over break material and backfill/waste rock). The unplanned dilution grade is estimated using Datamine 5D Planner Mineable Shape Optimizer Routine at 0.4 m over break on each the hangingwall and footwall.

15.3.4 Cut-off Grade

The mineral reserve was estimated based on a breakeven cut-off grade (COG) of 0.58% U_3O_8. The COG was estimated using the parameters shown in Table 15-4.

Table 15-4. Mineral Reserve Cut-off Grade Estimation - Gryphon

Item	Value	Unit
Mine Operating Cost	\$150	\$/t
Sustaining Capital Cost	\$0	\$/t
G&A Costs	\$99	\$/t
Surface Transportation Cost	\$50	\$/t
Mill Operating Cost	\$275	\$/t
Total Cost	**\$574**	**\$/t**
Unplanned Dilution	**20**	**%**
Unplanned Dilution Grade	**0.43**	**%**
Uranium Price	**50.00**	**CAD/lb**
Royalties	**7.25**	**%**
Mill Recovery	**97**	**%**
Cut-off - Mill Feed Grade	**0.58**	**%**

Source: Stantec – Cut-Off Grade Analysis for Gryphon Deposit memo dated March 13, 2018

15.3.5 Mining Block Shapes

Stantec used Mineable Shape Optimizer (MSO) software to produce stope shapes based on an in situ cut-off grade of 0.61% U_3O_8 (with external dilution, the cut-off grade would reduce to 0.58% U_3O_8) and practical design criteria. Only stope shapes with an average diluted U_3O_8 grade greater than the estimated COG (0.58%) were considered in the Mineral Reserve.

Parameters used to define stope shapes are summarized in Table 15-5.

Table 15-5. Stope Design Parameters - Gryphon

Stope Design Parameter Values	
Stope Height Maximum	15 m
Stope Height Minimum (uppers)	7.5 m
Stope Length Maximum (along strike)	20 m
Stope Length Minimum	10 m
Stope Width Maximum (FW to HW)	100 m
Stope Width Minimum	3 m
Minimum Dip	50°
Minimum Waste Pillar	2.5 m

Stope height and length define the minimum design shape. The minimum width is determined by minimum dimensions required to adequately operate the equipment to achieve the mining cycle, and the maximum width is used to define stope shapes from footwall to hangingwall.

In the case where a maximum width is designed which is greater than geomechanical constraints permit, additional stopes will be designed, called “panels”, to maximize resource recovery.

16 Mining Methods

16.1 Summary

This mining study is based on utilization of ISR recovery for mining of the Phoenix deposit and underground longhole stoping of the Gryphon deposit. After strategic evaluation of the mining options, Denison has planned to initiate the Phoenix development first, followed by Gryphon. The advantages for this development sequence include:

- Lower initial capital costs;
- Shorter timeframes to production;
- Attractive project economics at current market prices, allowing development to occur immediately;
- Generation of cashflow from Phoenix production will reduce or eliminate need for external financing for Gryphon deposit development and construction; and
- Provision of additional time for optimization of Gryphon mineral processing designs and/or negotiation of toll milling commercial agreements.

The life of mine production for the Wheeler River property is expected to be 14 years for the recovery of both deposits, providing a total of 109.4 million pounds of U_3O_8. Phoenix production duration is 11 years and provides a steady state production rate of 6.0 million pounds of U_3O_8 per year. Gryphon production duration is seven years, overlapping with Phoenix production ramp down, and provides a steady state production rate of 9.0 million pounds of U_3O_8 per year.

16.1.1 Phoenix Deposit

In 2016, Denison completed a preliminary economic analysis of the Phoenix deposit based on the application of an underground jet boring system (JBS), similar in nature to the technology applied at the Cigar Lake mine. In order to ensure a conservative and viable evaluation for the PEA, Denison only allowed for the application of proven mining methods. The goal was to provide a base case evaluation without the added risk of new technologies or innovations.

The PEA evaluation identified several significant disadvantages for the JBS mining method:

- Significant capital expenditures: Extraction using JBS would require the development of at least one separate ventilation raise from surface to underground, development of a 3 km long access drift from Gryphon to Phoenix, development of underground infrastructure at Phoenix, and establishment of a freeze curtain around the deposit.
- Significant time to development: Development of the above-mentioned ventilation raise, access drift, and freeze curtain was expected to require 5-6 years to complete.
- Limited production: JBS is currently able to produce ~10 t/hr at Cigar Lake.
- High operating cost: Due to the nature of the mining cycle and underground mining operations, operating costs were estimated in the PEA at ~$29/lb U_3O_8.
- High technical risk: Development of JBS for Cigar Lake required an extensive period of time to complete. Furthermore, during site development, several catastrophic events occurred related to these technical challenges that further extended development timelines and increased costs.

Despite the high-grade nature of the Phoenix deposit, extraction using JBS technology did not achieve the financial results targeted by Denison. As a result of the PEA analysis, Denison initiated a broader evaluation and assessment of potential extraction technologies which included those from other mining industry partners, civil applications, and the oil and gas industry, among several others. In all, a total of 32 potential extraction techniques were evaluated over the course of two years. The final two preferred technologies were advanced into a PFS level assessment, with Denison selecting ISR as the preferred mining method.

ISR mining has become a standard uranium production method, following early adaptation and use in the 1960s. Its application to amenable uranium deposits in certain sedimentary formations has grown owing to competitive production costs and low surface impacts. ISR operations are found in a number of countries, including USA, Australia, Kazakhstan, Uzbekistan, and India. In 1997, the ISR share in total global uranium production was 13%, and by 2011, it had grown to 46%. ISR mining is expected to remain a major uranium production method into the future. There has been continual development and improvement of ISR techniques, particularly in the two decades since the IAEA published the Manual of Acid In Situ Leach Uranium Mining Technology, IAEA-TECDOC-1239.

In an ISR operation, the mining solution is pumped through the underground orebody to dissolve the minerals. After dissolution, the solution (now referred to as the pregnant solution) is recovered and pumped to surface. Once on surface, the solution is transported to a processing plant and the uranium is recovered in much the same way as in any other uranium mill. As a result of this mining approach, there is minimal surface disturbance, no tailings, and minimal waste rock generated.

Benefit of ISR operations include:

- Established safety practices and procedures to ensure health and safety of workers.
- Minimal environmental impacts, including low noise, dust, and air emissions, low water consumption levels, minimal surface disturbance, and full rehabilitation of the area.
- Ability to scale production up or down to meet market demands.
- Insensitivity to ore grades (i.e. lixiviants will dissolve the uranium at any grades).
- Low initial capital costs and short timeframe to production.
- Low operating costs.

For a deposit to be considered viable for ISR extraction, it must have three general characteristics:

1. Mineralization must be located in permeable ground to allow the mining solution (i.e. lixiviant) to interact with the uranium mineralization.
2. Mineralization must be readily dissolvable by the mining solution.
3. Mineralization must be confined to the resource by either natural geological features (i.e. impermeable clay or other geological formations) or by artificial means (i.e. pumping, freeze walls). This is done for a variety of reasons, including:
    a. Maximizing recovery of the mineralization once the uranium is dissolved into solution by preventing outflow of the pregnant solution into the regional groundwater.
    b. Minimizing the dilution of the lixiviant with regional groundwater and avoidance of higher treatment costs to recover the uranium.
    c. Minimizing the potential for environmental damage.

The ISR approach at Phoenix meets all of these parameters. It is important to note that, traditionally, grade of the mineralization is not a key criterion to determine applicability of an ISR operation. Grade will naturally impact the economic viability of the deposit, but it has limited bearing on the applicability of ISR for low-grade conventional ISR operations.

While the planned extraction for the Phoenix deposit meets the key parameters, there are important differences between conventional ISR operations and that envisioned for the Phoenix deposit. In conventional ISR operations, containment is typically completed using natural impermeable layers in the geological strata and/or by creating a natural drawdown of the water table towards the ore zone (i.e. pumping out more solution than injecting). At Phoenix, there is a natural impermeable layer below the deposit, but the ground is otherwise hydraulically connected to the regional groundwater throughout the Athabasca Basin. Due to the high-water flows and movements through the deposit and sandstone, creating a depression in the water table was estimated to be impractical. Therefore, in order to contain the lixiviant to within the mineralized zones, an artificial freeze wall will be established surrounding the deposit. Freezing technology and methodologies are well established throughout the world and in the Athabasca Basin.

A second difference is the ability for mining solutions to permeate the ore zone. In conventional ISR operations, the geology of the ore zones is required to be relatively homogeneous in terms of permeability to allow the lixiviant to come into contact with the low grade (ppm levels) of uranium mineralization spread throughout the deposit. Conventional deposits grade in the ppm and as a result, mass and volume loss of the mineralization ground during dissolution of the orebody is not a factor to be considered. Conversely, the Phoenix deposit does not have homogeneous permeability. The geology of the deposit is highly variable, with severe fracturing, broken and desilicified sands, and zones of high clays and high-grade uranium metals. With zones grading upwards of 40% U_3O_8, the resulting loss of mass and volume of the uranium will be significant to the operation, especially since permeability is expected to increase as the orebody is dissolved during the mining process.

A third major difference is the concentration of the lixiviant once it has dissolved the uranium. In conventional ISR operations, soluion mill feed concentrations are in the mg/L levels. This requires the use of ion exchange or solvent extraction processing equipment to concentrate the uranium to allow for the efficient precipitation and packaging of the final product. To meet annual production requirements, the volume of solution to be processed to recover the uranium is quite large. Conversely, due to the high-grade nature of the Phoenix deposit, laboratory test work has demonstrated that concentrations of the solution have been consistently above 10 g/L and as much as 27 g/L. At this level of concentration, much smaller volumes of solutions are required to be processed for an equivalent production level. As a result, direct precipitation of the uranium is viable, which eliminates the need for ion exchange or solvent extraction circuits. Capital costs are reduced, as are personnel and reagent consumption costs during operations. Thus, operations are streamlined, and operating costs are much lower.

Finally, the last major difference is the lack of limitations that are faced by conventional ISR operations. Due to the nature of the mineralization formation (i.e. roll front deposits), the mineralization is typically spread out over several kilometers. The low-grade nature of the deposit combined with drill hole spacing, reagent consumption, and surface piping and pumping for distribution systems all contribute to creating economic thresholds which impact the viability of some deposits. Conversely, at Phoenix, the mineralization is confined to a relatively small area (~1 km x 50 m) with a readily leachable mineralization. As a result of the smaller surface area, infrastructure costs (i.e. drill hole spacing, reagent consumption, and surface piping systems) are reduced compared to conventional ISR operations to the point where it is immaterial to the overall economics of the deposit.

16.1.2 Gryphon Deposit

The extraction strategy for Gryphon has not changed from the 2016 PEA. The planned mining method for Gryphon is conventional longhole stoping with backfill. Longhole stoping is a widely used conventional mining method used in both the Canadian uranium industry as well as in the broader mining industry in base metals, gold, and other commodities.

Access to the Gryphon deposit will be established through two shafts. The primary shaft will provide for movement of personnel and supplies, ore/waste hoisting, and fresh air to the underground operations. The second shaft will be solely for exhaust air and secondary egress. Both shafts will be excavated through blind boring methods. Blind bored shafts have been selected for vertical access in favour of typical full-face shaft sinking with cover grouting or freeze curtain protection. Blind bored shafts offer more competitive costs and construction schedules, and a reduced risk profile while sinking through saturated ground conditions. A composite steel/concrete liner will be installed over the full length of the shaft and grouted into basement rock.

In the underground operation, initial underground development will focus on establishment of permanent infrastructure and flow through ventilation between the main shaft and the exhaust shaft. Most of the permanent infrastructure will be located on the 500 m level, the level of the main shaft station. Following this, development priorities will be to establish access to the E series lense (E Zone), which provides early opportunity for ore production and waste rock storage (in mined out stopes). As mining is initiated in the E Zone, ramp development will continue to provide access to the remainder of the ore zones.

Ore will be hoisted to surface and transported to the McClean Lake mill for processing. A two-year ramp-up to full production is allowed for, with the full production rate set at 9 Mlbs/year.

16.2 Hydrogeology

Groundwater flow at the project is fracture controlled and defined by two primary hydrogeological units. The shallowest, a regionally extensive sandstone and conglomerate dominated formation known as the Athabasca Group, unconformably overlies a Crystalline Basement composed of metasedimentary and granitoid gneisses. Secondary permeability (i.e. fractures, bedding planes, and joints) is anticipated to be the primary component for groundwater flow in both units.

Horizontal groundwater levels and gradients are consistent with major regional lakes, which generally flow in an easterly direction. Lakes likely act hydraulically as flow through features, dominating the overall groundwater flow system and keeping the water table near or at the ground surface. No measurable vertical gradients are observed at the deposits, with deep vibrating wire piezometers displaying water levels near or at regional lake levels. Global recharge estimates near the Wheeler River site suggest a mean recharge from precipitation of 2.4% of the mean annual precipitation.

16.2.1 Field and Lab Test Work

Hydrogeological investigations have been ongoing in the field and in laboratories since 2014. Packer, open hole, and cross hole tests have been completed in conjunction with exploration drilling programs at both deposits. As well, permeability tests have been completed on sections of available competent core within the Phoenix deposit. Open hole water level surveys have been completed across the site in 2015 and 2017.

The hydraulic conductivity related field and lab test work data is summarized in Table 16-1.

Table 16-1. Hydraulic Conductivity Related Data Set

Deposit	Test Type	Location	Number of Tests
Phoenix	Field - Packer	Sandstone	11
		Unconformity	6
		Basement	15
	Field - Open Hole		3
	Lab - Permeability	Unconformity	11
Total Phoenix			**46**
Gryphon	Field - Packer*	Sandstone	54
		Unconformity	12
		Basement	62
	Field - Open Hole		8
	Field – Cross Hole		4
Total Gryphon			**140**

*Includes 34 packer tests completed within production shaft pilot hole

16.2.2 Hydrogeological Properties

Water Levels

From the open hole water level survey results, horizontal gradients are consistent with the major regional lakes with water levels near or at the ground surface (500 to 520 MASL). Near the Phoenix deposit, groundwater flow is found to be north-easterly, while Gryphon is relatively stagnant, indicating a flat water table between the major nearby lakes (Figure 16-1). The close proximal match of water levels with regional lakes indicates that groundwater flow is likely controlled by the regional lakes, which act as sources and discharge zones (i.e. subsurface flow through lakes).

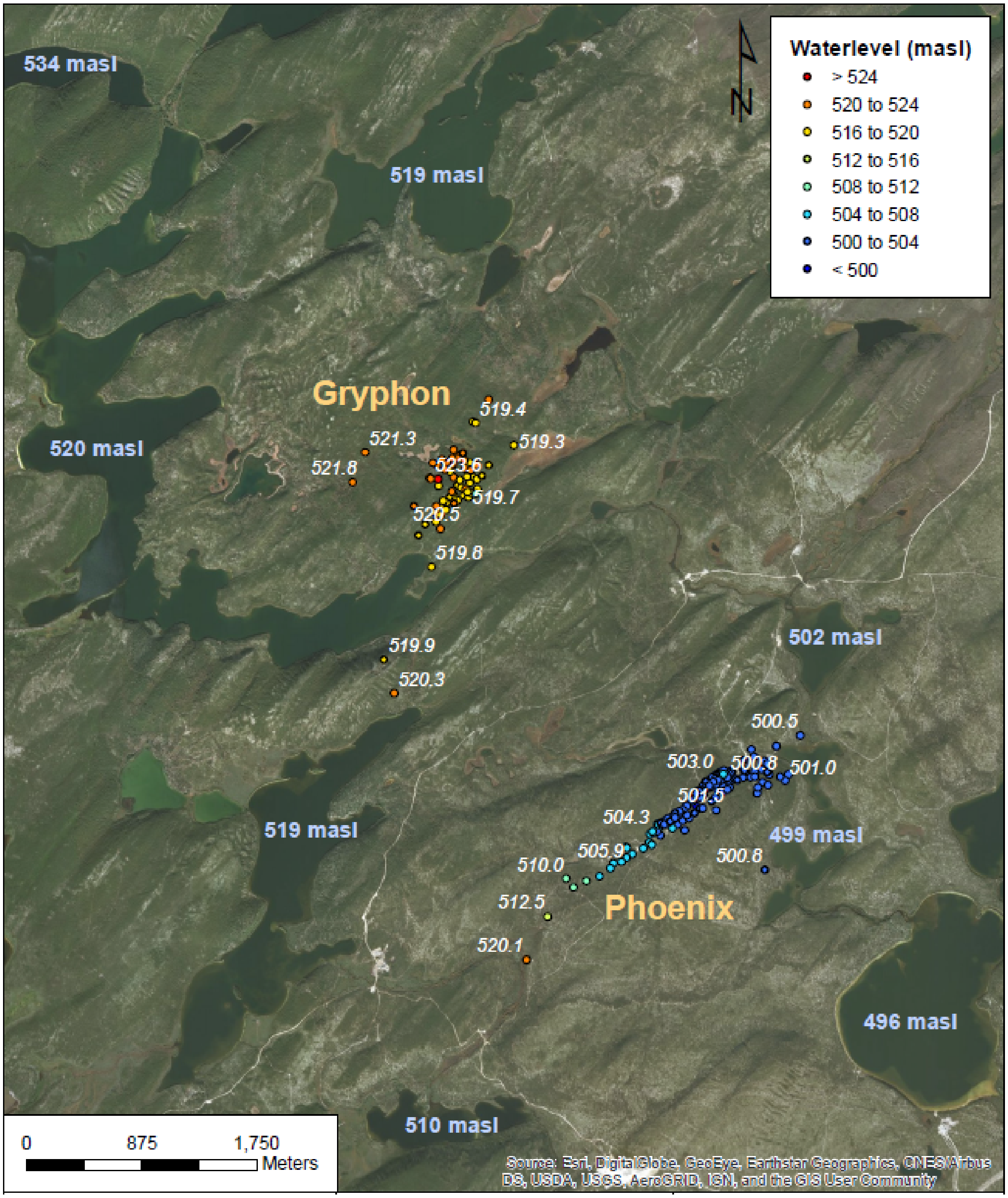
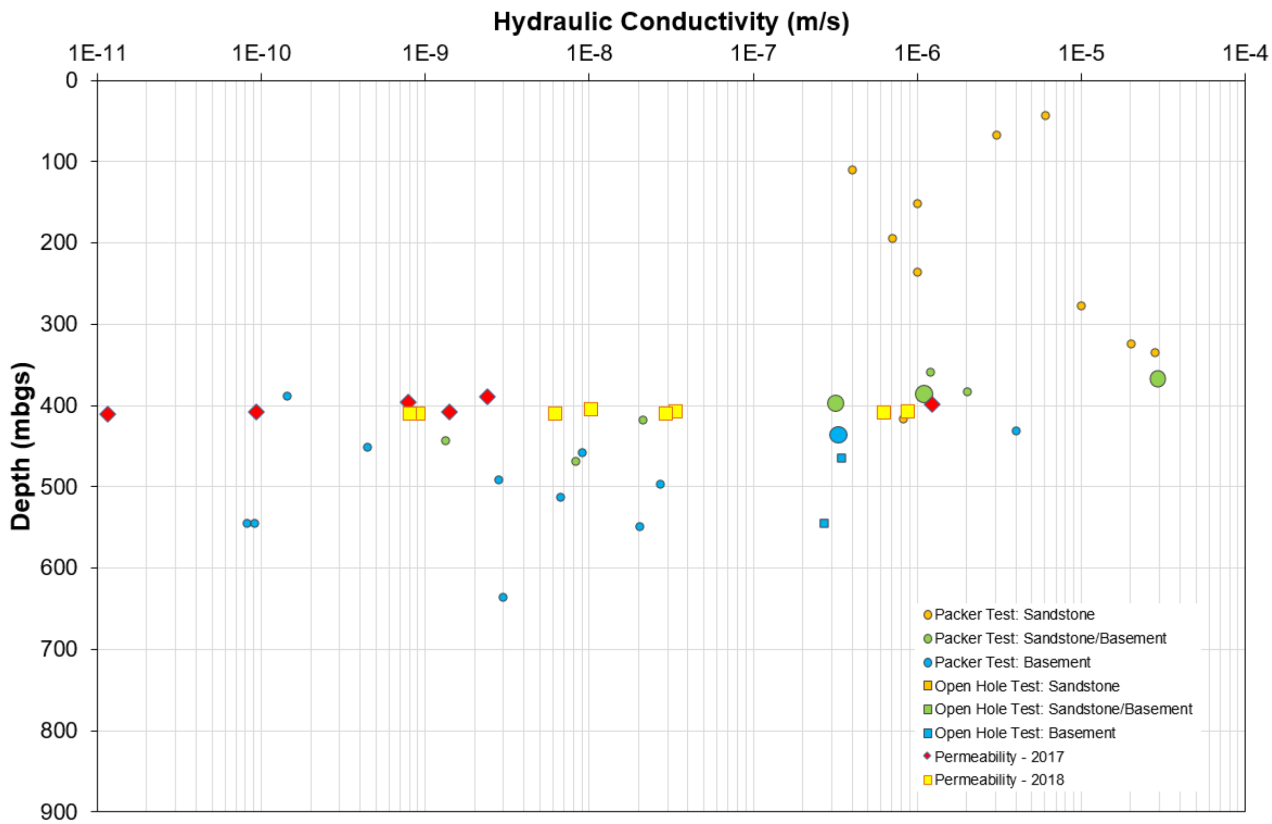


Figure 16-1. Open Hole Groundwater Levels Compared to Regional Lakes

Phoenix Test Work Results

The hydraulic conductivity test results can be summarized as follows (Figure 16-2):

- Six field packer tests completed at the unconformity – hydraulic conductivity values (K values) ranged from 1.26×10^{-10} m/s to 7.41×10^{-5} m/s.
- Eleven lab permeability tests completed on competent exploration diamond drill core – K values ranged from 9.37×10^{-11} m/s to 1.23×10^{-6} m/s.



Source: SNC-Lavalin– Uranium Core Hydraulic Conductivity Testing dated July 6, 2018 (Permeability – 2018)
Source: SRK– Flow Rate Predictions for Proposed Phoenix In-Situ Recovery Wheeler River Project dated June 2018

Figure 16-2. Phoenix Hydraulic Conductivity Data Near Unconformity

Phoenix Numerical Modelling

Numerical groundwater modelling has been completed using the modelling software FEFLOW to understand the sensitivity of ISR wellfield design and operating practice on fluid flow. The SRK modelling report, titled Flow Rate Predictions for Proposed Phoenix In-Situ recovery, Wheeler River Project dated June 2018, is summarized below.

A two-dimensional model was created that was bound by geological outline of the defined mineral resource in the A Zone. This simplified approach was used based on the assumption that the freeze cap, to be established above the deposit and the crystalline basement below the deposit, would be effectively impermeable. Homogenous K values were assigned to the model and incrementally increased by roughly half an order of magnitude to determine the effect of K on flow rates. The lab permeability test work was used as a lower boundary for the average K values modelled.

As well, the well cluster (a single unit of injection and recovery wells) geometry, number of active well clusters, well spacing, and pressure differential between injection and recovery wells were varied to determine their effect on flow rates.

The following relationships were concluded from the modelling:

- A hexagonal well cluster geometry (a unit of six perimeter injection wells with a single central recovery well) provided greater flow rates than a square well cluster geometry (a unit of four perimeter injection wells with a single central recovery well).

- The number of active well clusters has a proportional effect on flow (i.e. two active well clusters will result in twice as much flow).
- Well spacing demonstrated an inverse, quasi-liner relation (i.e. halving the well spacing resulted in a 30% increase in flow).
- The pressure differential between injection and recovery wells has a proportional effect on flow.

Based on the ISR operating assumptions being used for the purposes of the PFS, where 10 active hexagonal well clusters at a well spacing of 10 meters and pressure differential of 1 MPa, the target flow rate of 500 L/minute will require a K value of approximately 1 x 10^{-6} m/s (Figure 16-3).

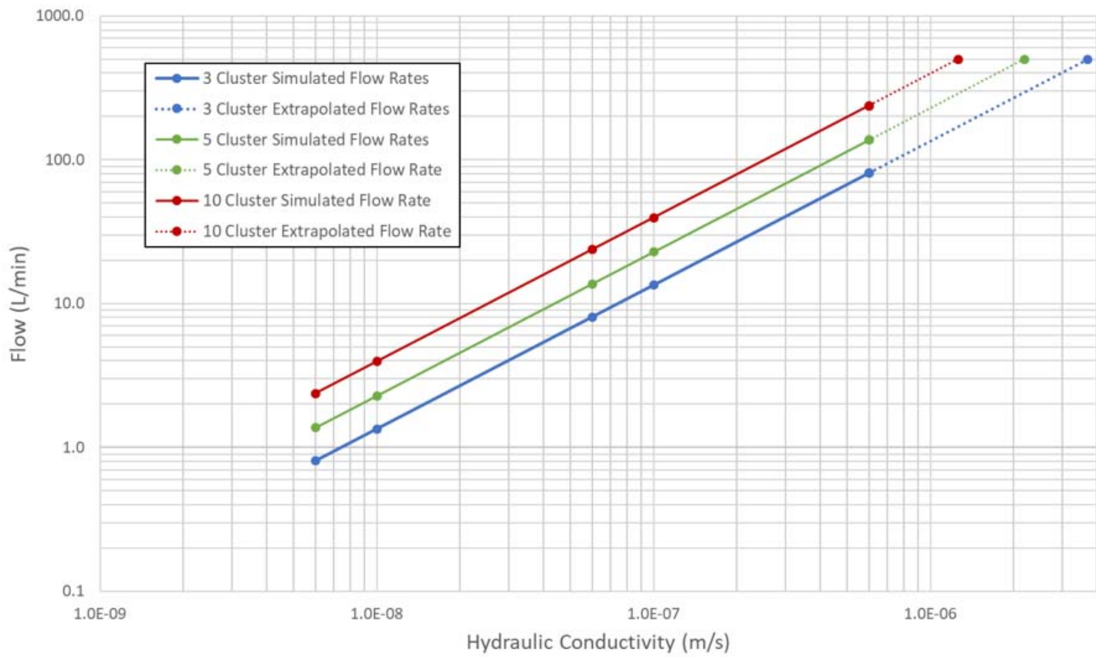


Figure 16-3. Effect of Hydraulic Conductivity and Number of Active Well Clusters on Flow

Gryphon Test Work Results

The K values in the Gryphon deposit area plotted against depth are summarized in Figure 16-4. On average, interpreted K values are typical of fracture rock systems, ranging between 1 x 10^{-11} and 3 x 10^{-5} meters/second (m/s), with a geometric mean of 3 x 10^{-8} m/s and a log-transformed standard deviation of 1.6 orders of magnitude. Subdivision of hydraulic tests into the two primary geological units, sandstone and basement (Figure 16-5), indicate:

- Sandstone geometric mean of 8 x 10^{-8} m/s with a log-transformed standard deviation of 1.4 orders of magnitude.
- Basement geometric mean of 6 x 10^{-9} m/s with a log-transformed standard deviation of 1.3 orders of magnitude.

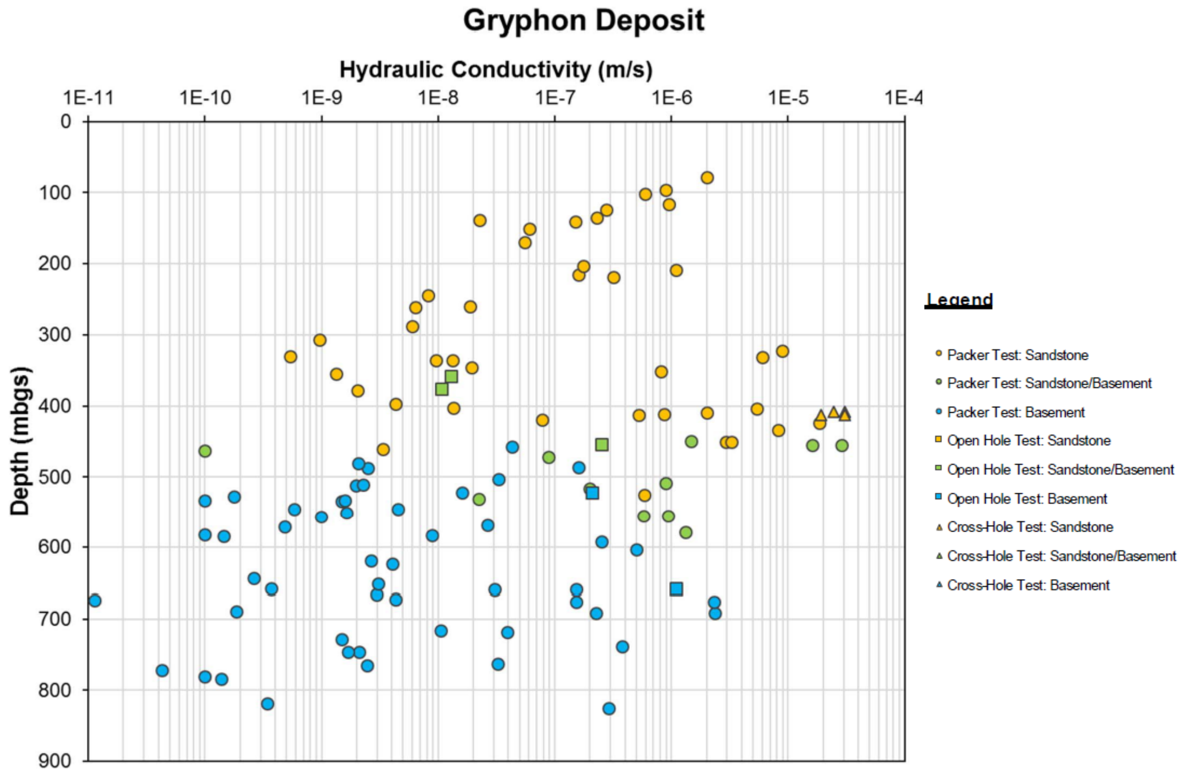


Figure 16-4. Gryphon Hydraulic Conductivity vs Depth

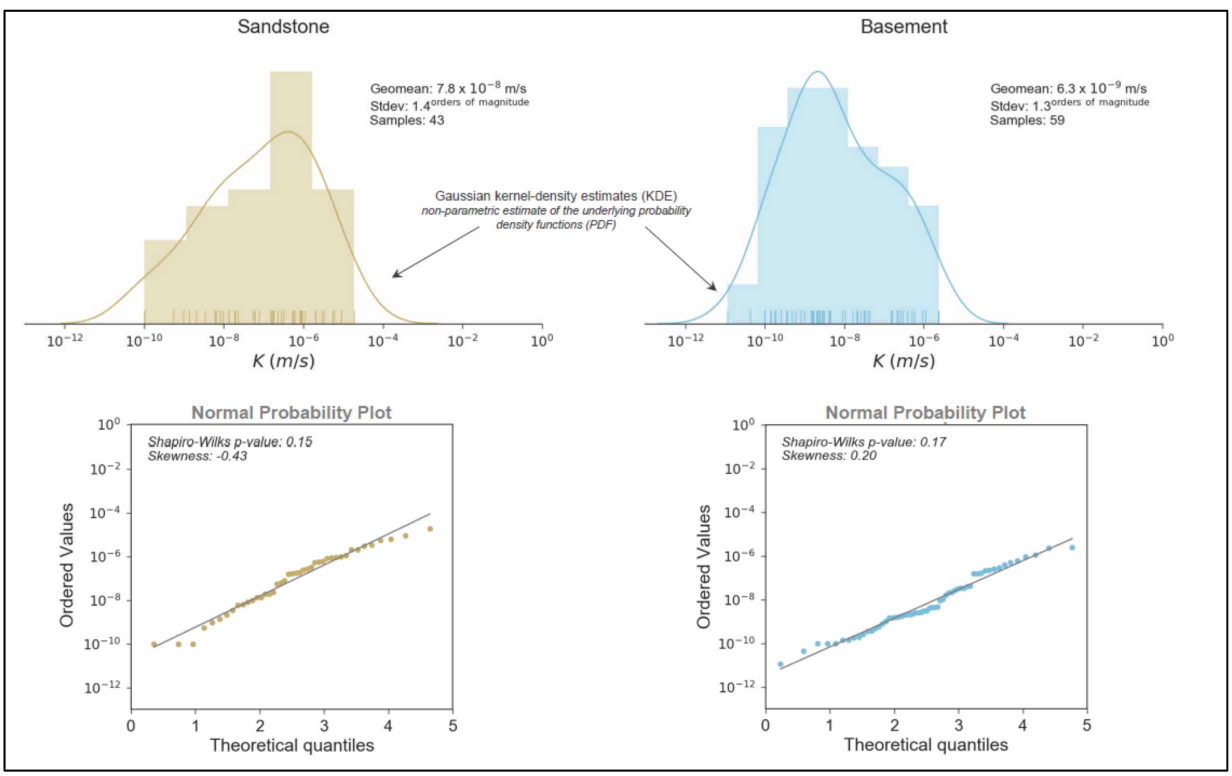


Figure 16-5. Gryphon Packer Test Summary Statistics

Gryphon Numerical Modelling

Numerical groundwater modelling has been completed using the modelling software FEFLOW based on the available hydrogeological data to estimate mine water inflow rates to the proposed underground mine. Inflow rates were estimated using stochastic simulation techniques with the hydraulic conductivity varied based on field testing to date. The SRK modelling report titled Flow Rate Predictions for Proposed Gryphon Underground Mine, Wheeler River Project dated April 2018 is summarized below.

The groundwater model encompassed the potential catchment area for the Gryphon deposit, based on surface drainage patterns and regional lakes, covering a total area of 475 km^2 and the base of the model was set at a constant depth of 2,900 meters below surface. The finite element mesh that made up the model was composed of approximately 94,000 nodes.

Mine water inflow was estimated using Monte Carlo techniques, where the hydraulic conductivity of individual model elements was randomly varied given the K value distributions illustrated in Figure 16-5 using a method that considered the spatial similarity of hydraulic conductivity over distance.

The production and ventilation shafts were assumed watertight and thus excluded from the model.

Additional sensitivities scenarios were completed to explore the influence of cover grouting during development. These scenarios look at potential inflow reduction if effective grouting is carried out in the upper parts of the mine as development occurs. The grouting was assumed to be a meter thick and reduce hydraulic conductivities to 1 x 10^{-9} m/s.

In total, 1,000 unique realizations of the model were used to estimate the uncertainty in mine inflow predictions. The results suggest a median (50th percentile) inflow of 430 m^3/hr. Summary statistics for various mine water inflows are tabulated in Table 16-2. The presented percentile statistics represent best estimates of the probability of inflow non-exceedance and were used to determine mine planning parameters (i.e. a P90 suggests that, given the constraints of the model, there is a 90% probability that the actual inflows to the Gryphon underground will be less than the P90 value).

Table 16-2. Estimated Mine Inflow Rates

Statistic	**Statistical Abbreviation**	**Estimated Inflow Rate (m^3/hr)**	
		Base Model (no grouting)	**Grouting (K = 1 x 10-9 m/s, thickness = 1.0 m)**
Median (50th Percentile)	P50	430	170
90th Percentile	P90	580	200
99th Percentile	P99	730	220

Source: SRK – Inflow Predictions for Proposed Gryphon Underground Mine – Wheeler River Project dated April 2018.

Gryphon Uncontrolled Inflow

Another risk when mining in the basement, but in proximity to the unconformity in the Athabasca Basin, is an uncontrolled high inflow from back instabilities and 'chimneying' into the overlying sandstone. The numerical model cannot estimate this flow. However, a review of high inflow events from existing mines in the Athabasca Basin suggests that the highest inflow risks are associated with roof collapse on underground workings, and subsequent catastrophic "chimneying" into the

Athabasca sandstone. Such events at the McArthur River and Cigar Lake mines resulted in inflow rates of 800 m^3/hr (Liu et al., 2008), and 1,000-1,500 m^3/hr (Bashir and Hatley, 2012), respectively.

16.3 Geotechnical

A prefeasibility level geotechnical evaluation has been completed to assess and characterize the rock mass conditions at the Gryphon deposit for the proposed underground mining. As it is proposed that the Phoenix deposit be mined via ISR methods, geotechnical analysis was not included in the PFS.

The critical geotechnical aspects that typically require consideration for the Athabasca Basin uranium deposits that are applicable to the Gryphon deposit include:

- Proximity to the regional unconformity and potential for high pressure and large volumes of water associated with the Athabasca sandstone.
- Presence of major structures with potential for poor ground conditions or hydraulic connection to aquifers within the Athabasca sandstone.
- Rock mass conditions and weakening clay alteration of basement rocks from mineralizing events.
- Rock mass conditions in infrastructure areas.

16.3.1 Geological, Structural, and Alteration Models

The basement geologic units from the hangingwall (HW) to the footwall (FW) include (Figure 16-6):

- Pelitic gneiss in the area of the shafts and off-shaft infrastructure and ramp.
- Pelitic gneiss and pegmatite in the area of the upper ramp.
- Graphitic pelitic gneiss for most capital ramp and orebody access development.
- Underlying the hangingwall graphitic pelitic gneiss (“Upper Graphite”) is quartzite (“Quartz-Pegmatite Assemblage”).
- Hangingwall mineralization is located in the contact area between the quartzite (“Quartz-Pegmatite Assemblage”) and graphitic pelitic gneiss (“Upper Graphite”).
- Footwall mineralization is located in the graphitic pelitic gneiss (“Lower Graphite”) and pegmatite (“Basal Pegmatite”).
- Unconformity is underlain by a paleoweathered zone, grading to fresh basement.

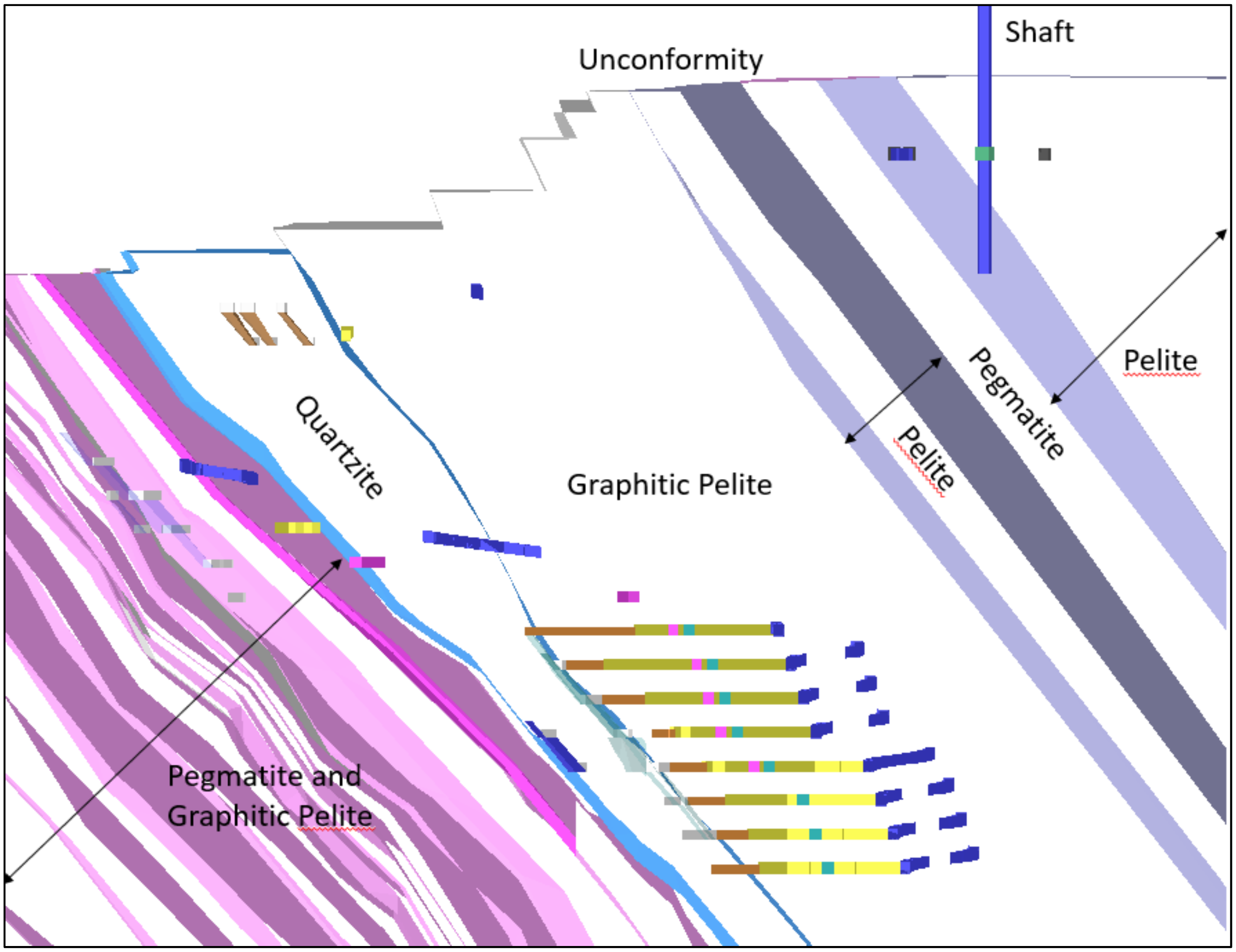


Figure 16-6. Gryphon Simplified Geological Section (also shows the location of mine design features)

Gryphon's structural setting is characterized by a series of thrust faults displacing the unconformity upwards to the southeast in multiple steps. These structures are generally located at the contact between relatively less competent graphitic pelitic gneisses and more competent quartz-pegmatites, pegmatites, and pelitic gneiss units. The faults are brittle in nature and can be described as a combination of cataclasites and gouges with intervals of blocky and friable core. The most significant structures occur at the contact of the upper graphite with the overlying pelitic gneisses and at the base of the Upper Graphite in contact with the underlying Quartz-Pegmatite Assemblage. These structures are termed the Offset Fault and Graphitic Fault (G-Fault), respectively (Figure 16-7). As well, there are several fault interpretations cross-cutting the ore zone, NE, EW, and WNW faults, as shown in plan in Figure 16-8.

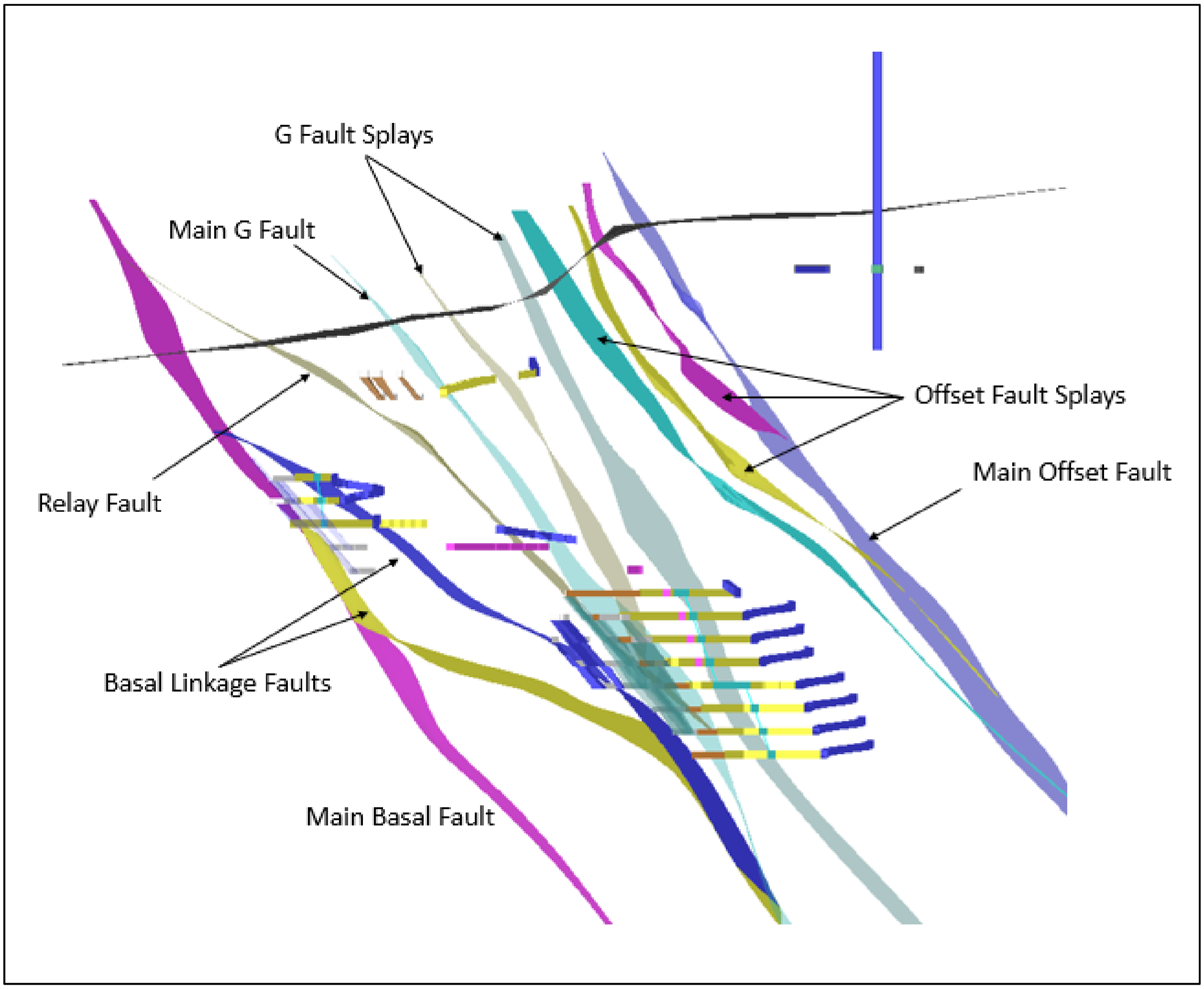


Figure 16-7. Typical Section of Major Structures at Gryphon

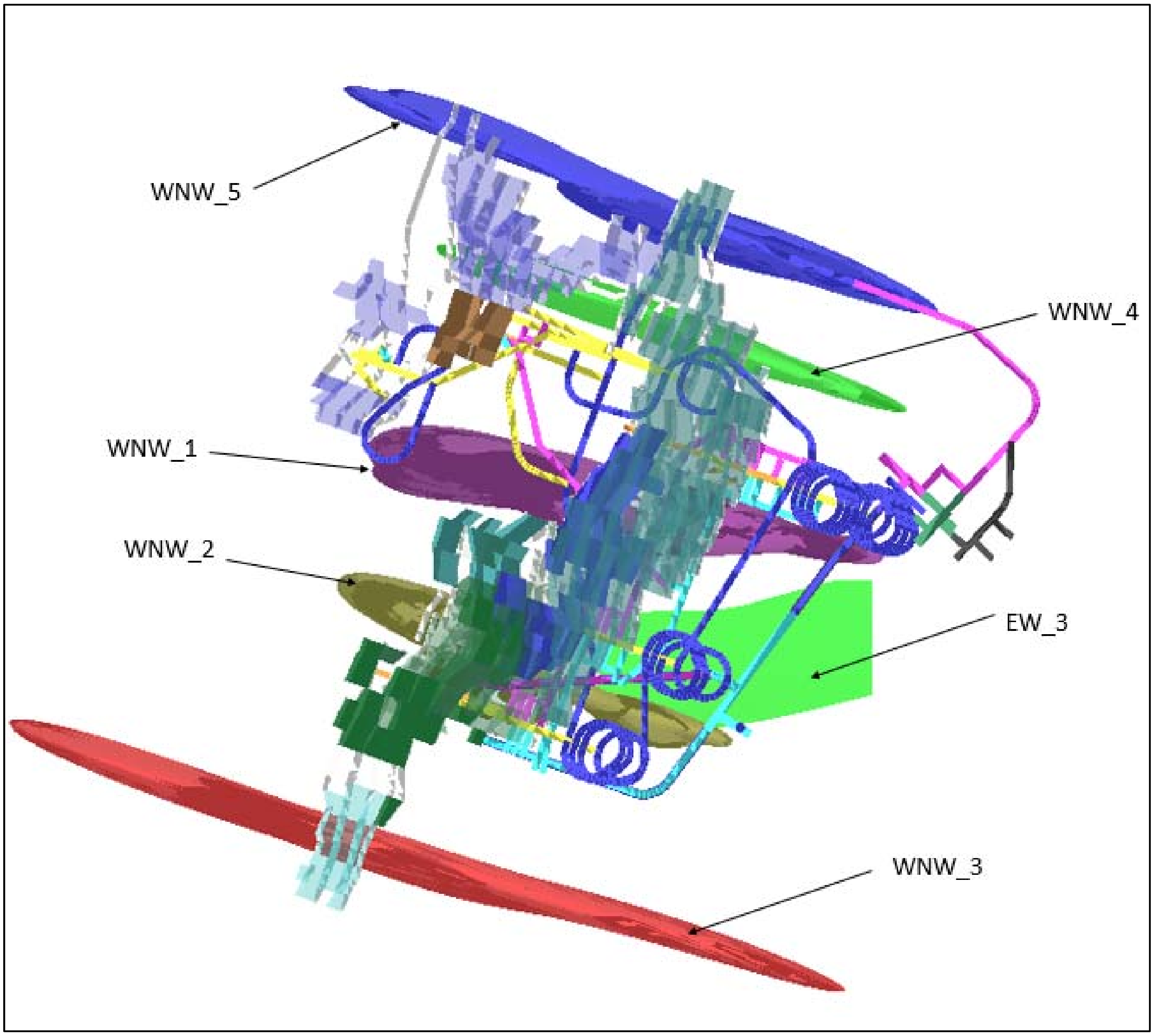


Figure 16-8. Plan of Cross-cutting Fault Zones

There are currently 36 interpreted fault structures. Typically, the faults are discrete and limited to approximately 1 meter in maximum width. General geotechnical conditions associated with some of the prominent faults include:

- The Offset Fault and associated splays occur between the shaft and the deposit and thus the ramp between the shaft and deposit must cross these faults. The mine design avoids the faults where conditions are interpreted to be worst and widest.
- The main G-Fault is likely to have the most impact on stoping, due to expected poorer quality conditions when intersecting the ore zone.
- The five interpreted cross-cutting fault zones (WNW-1 to WNW-5) intersect perpendicular to the strike of mineralization along the deposit. From core logging data, the maximum fault damaged horizontal width has been estimated at approximately 5 meters (SRK 2017). The areas of poorest ground are predicted in WNW_1, between WNW_1 and WNW_2, and WNW_3. The conditions associated with these areas have been considered in the geotechnical analysis; impacts, but are likely to be local, on a single stope scale or less.

- Fault EW_3, intersected by shaft hole WR-695, indicates good quality rock mass conditions and relatively low hydraulic conductivity.
- Core logging data suggests the EW_4, EW_5, and NE faults are not anticipated to have a material impact on mining.

A conceptual alteration model is depicted in Figure 16-9. This aids in understanding the areas impacted by hydrothermal alteration and the type and relative intensity of alteration.

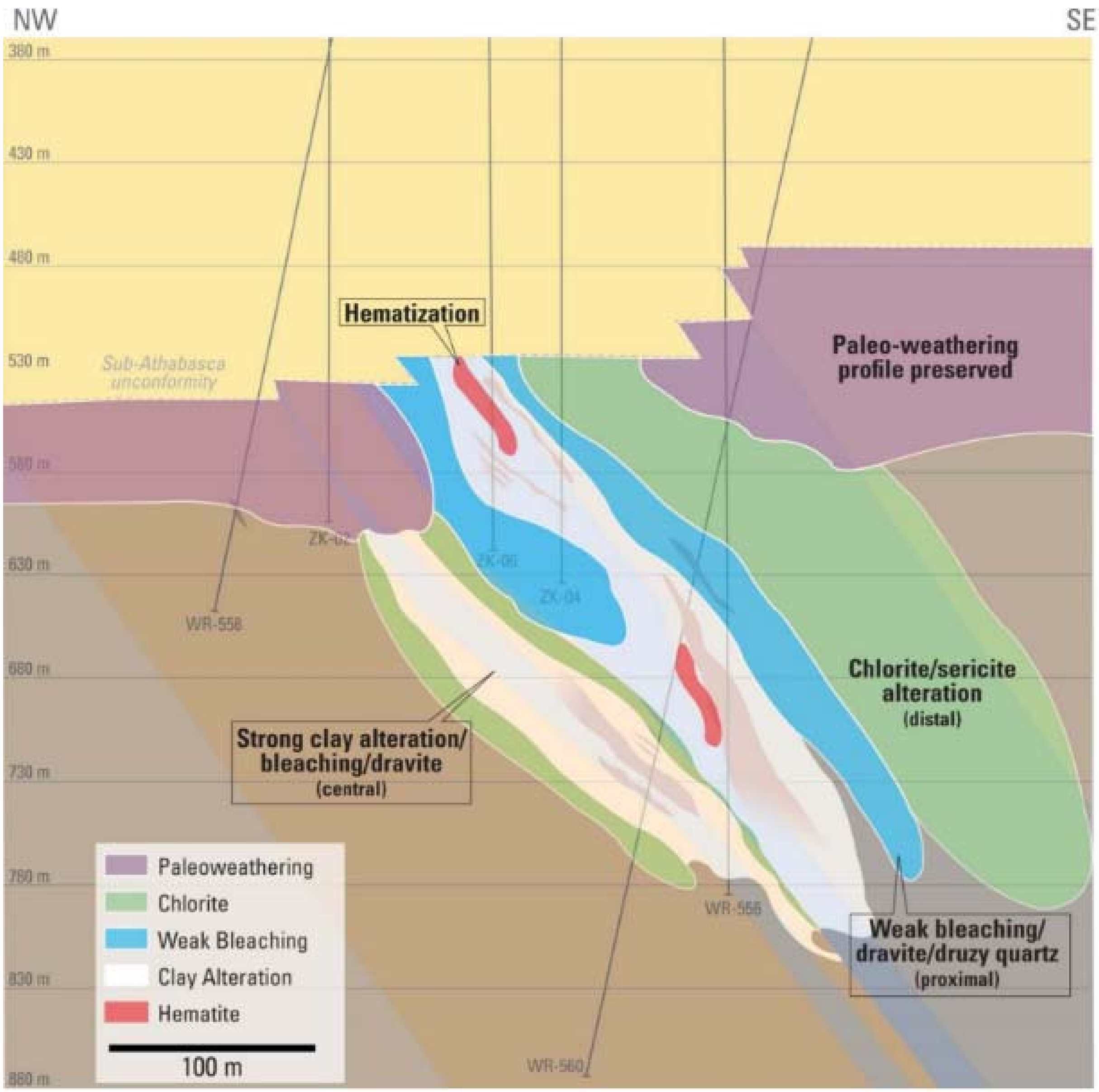


Figure 16-9. Gryphon Alteration Conceptual Model

16.3.2 Rock Mass Characterization

The general rock mass characteristics by lithology from HW to FW are as follows:

- Pelitic Gneiss is a foliated, fine grained, strong to very strong rock (average 70 MPa), with an RMR >60 (good to very good). It is interpreted to be semi-massive to blocky with favorable joint conditions. It has a strength anisotropy depending upon load orientation with respect to the foliation.
- Pegmatite is medium to coarse grained and massive to semi-massive. It is strong to very strong (50-100+ MPa) with an RMR >60, classified as a good to very good rock mass.
- Graphitic Pelitic Gneiss in the immediate stope HW (Upper Graphite). It has an approximate intact strength between <5 (rarely) and (mostly) 50 MPa with an average RMR of 60. The minimum lab tested strength is 12 MPa, but locally strengths are estimated to be lower. It has a strength anisotropy depending upon load orientation with respect to the foliation.
- Quartz Pegmatite Assemblage is approximately 20+ MPa higher in strength than the Upper Graphite with an average RMR ~65. It has a strength anisotropy depending upon load orientation with respect to the foliation.
- A "Paleoweathered Zone", generally from 3 m to 10 m thick, is superimposed on the crystalline rocks and occurs immediately below the unconformity. This zone is of significantly poorer quality than unaltered basement rock (RMR <40). At the production shaft location (WR-695), this zone has an approximate RMR of 60 and intact rock strength averaging 35 to 40 MPa.

For predicting rock mass behavior and identifying potential failure mechanisms, it is important to characterize the rock mass considering several factors:

- Geology/mineralogy (clay content and susceptibility to rock mass degradation upon exposure; friability and rock mass "durability").
- Fabric/foliation (strength anisotropy, "fabric or foliation strength").
- Discontinuities (frictional characteristics, orientation, block size/shape).
- Intact strength and discontinuity mechanical properties (point load testing and lab testing).
- Quantitative rock mass classification (RMR/Q).
- Groundwater conditions.
- Pre-mining in situ stress regime.

The primary concern with the intensity of clay alteration is its impact in reducing the rock mass strength and its tendency to degrade over time from exposure to the elements. The stopes and immediate access development will commonly be within this alteration halo, to varying intensities. This halo can be considered as a geotechnical domain, overprinting lithological contacts.

The basement rock within the deposit area varies between massive to very blocky, intensely foliated to disintegrated at discrete fault locations. There is an increase in joint density (decreasing block size) in connection with major structural features and alteration/weathering. The frictional characteristics of discontinuities are poorer in proximity to lithologic contacts, major structural features, hydrothermal alteration/mineralization, and paleoweathering.

There is a dominant foliation which results in an anisotropic strength and is dependent on the load orientation to foliation. Point load test (PLT) and lab test samples have been loaded parallel thru perpendicular to foliation. Most of the PLT has been diametral and more tests have been performed parallel and sub-parallel to foliation versus perpendicular. There is a marked decrease in unconfined compressive strength within the graphitic pelitic gneiss versus any other major lithology, averaging 50 MPa versus 70+ MPa for most of the other units. The PLT data has been summarized in Table 16-3.

Table 16-3. 2018 Point Load Testing Program

Lithology	Equivalent UCS (MPa)	Min - Max (MPa)	No. Samples
Pelitic Gneiss	79	1-287	192
Pegmatite	101	5-240	143
Graphitic Pelitic Gneiss	49	0-173	272
Quartzite	73	9-167	31
Pelitic Quartzite	183	31-208	17
Garnetiferous Pelite	57	0-201	66
Psammite	79	15-153	12
Corderirite Pelite	79	16-183	10
Basement Axial	66	2-181	129
Basement Diametral	82	-	9,528

Geotechnical logging of all exploration drilling has been continuous since 2015, which includes over 150,000 meters of drilling on approximately 25 meter by 25 meter spacing. A full suite of standardized rock mass classification parameters has been logged, including RQD, intact rock strength, fracture count, joint count, friability index, joint roughness, joint weathering, joint aperture, etc.

Laboratory testing was completed in 2016, 2017, and 2018. Valid lab test results have been used in the geotechnical analysis, to inform intact rock properties and help with the assignment of rock mass properties for analysis.

Rock mass quality throughout the Gryphon deposit typically ranges from predominantly “FAIR” to “GOOD” using established rock mass classification terminology (RMR and Q-Systems). Within the basement units, the intact rock strength can vary between R0 (very weak) to R5 (very strong). ‘Typical’ fresh basement is classified as strong rock (R3, 50-100 MPa). Rock mass classification ranges (RMR and Q’) were generated as outlined in Table 16-4. These ranges are considered to represent ‘typical’ conditions encountered at Gryphon. They are also informed by compositing the RMR data set within each 3D geological unit and clay alteration model to assess the statistical RMR distribution within each unit. The stope shapes themselves were also composited as a final check to the suitability of the RMR design ranges.

Table 16-4. Rock Mass Classification Design Inputs

Mine Domain	RMR Design Range		Q' Design Range	
	Lower	Upper	Lower	Upper
Stope hangingwall - fair rock	50	60	2	10+
Ore zone (sills) - fair rock	45	55	2	6+
Footwall - fair to good rock (intensely clay altered areas conditions are poor)	55	65	3	10+
Shaft bottom, off-shaft ramp, and orebody ramp	70	80+	20+	
Operating development in HW (orebody access) - fair to good	55	70	4	20+
Faults or local very poor to poor quality areas	25	40	<0.1	1

The E series lenses mineralization occurring to the extreme northeast is in closer proximity to the unconformity than the remaining mining areas. As such, the rock mass quality is more variable, with RMR values typically 10 points lower than other zones/lenses. Conditions in this location will require confirmation at the next level of study.

There are a few local areas which have a markedly reduced rock mass quality, evidenced by the assessment of the RQD data. These often avoid the immediate HW and are typically concentrated more in the FW of the HW lenses.

Geotechnical data from the production shaft pilot hole suggests the proposed location is a good/workable location in terms of rock mass quality and low hydraulic conductivities (from packer testing). There is one interpreted fault structure (Fault EW_3) that potentially intersects infrastructure planned in the vicinity of the shaft bottom, but it appears that it has had minimal impact on rock quality or hydraulic conductivity in the shaft area.

16.3.3 Ground Support

The stability of proposed excavations was assessed using analytical and empirical methods. This includes consideration of the rock mass, fabric and joint orientations, and the impact of stress on stability and support. The support recommendations include patterned resin rebar rock bolting, welded-wire mesh, shotcrete (with and without fibre reinforcement), cable bolts, steel straps in very poor conditions, and/or shotcrete arches. Note that the ‘very poor’ conditions are not predicted in the stoping areas.

The ground support design was assessed and determined using the widely accepted Q-System empirical design chart after Grimstad and Barton (1993, 2013) and the commercially available Unwedge© software (Rocscience, 2017), to develop minimum ground support recommendations. Mining induced stresses were evaluated using RS3© (Rocscience, 2018), a 3D stress analysis program, to inform both development and stope support design.

Per the Q-System methodology, empirical ground support assessments for LOM infrastructure, permanent “worker-entry”, and temporary “production” headings were conducted at ‘excavation support ratios’ (ESR) of 1.3, 1.6, and 3, respectively. Ground support designs were then refined based on operational considerations, such as standardization of ground support and experience with support in analogous operations.

Ground support recommendations are shown graphically in Figure 16-10 and summarized in Table 16-5. These support recommendations are provided for the range of anticipated representative rock mass conditions at Gryphon. Figure 16-10 presents a widely used empirical design chart for estimating ground support. The green shaded area indicates the typical range in rock mass conditions. Using patterned bolting with screen as a primary means of ground support, a relatively small percentage of capital and operating development will also require 2 to 4 inches of shotcrete. Local fault areas and ore sills will require fibre reinforced shotcrete, welded wire mesh, bolt spacing of 1.2 x 1.2 m, and variable bolt length depending on span, including cable bolts.

In addition, it is anticipated that within stopes, shotcrete will be needed to provide a gamma radiation barrier to minimize worker exposure. A shotcrete thickness of 75 mm is required for gamma blocking, which will be sufficient for, and enhance, support for most rock mass conditions in ore sill development.

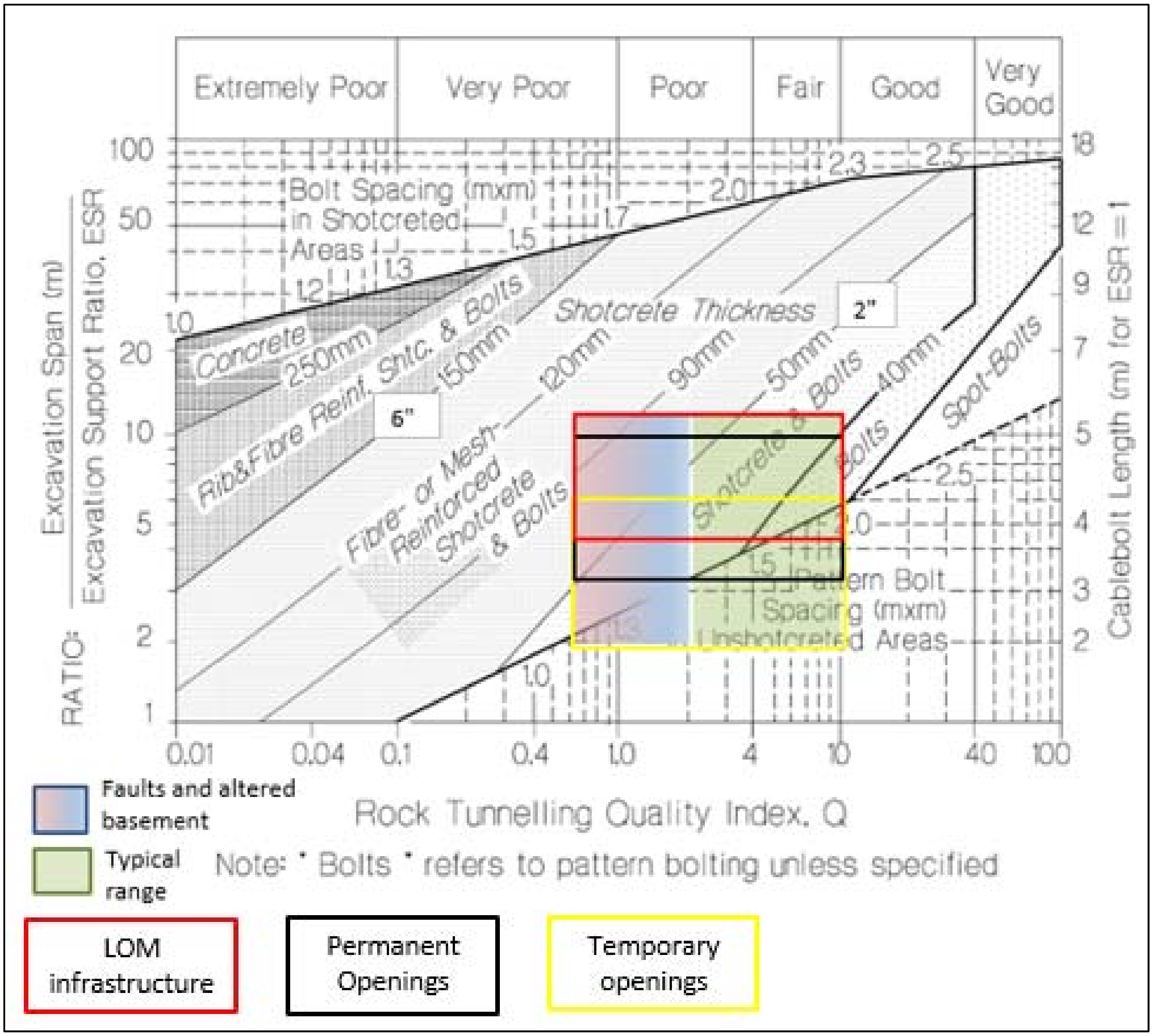


Figure 16-10. Empirical Ground Support Recommendations

Table 16-5. Ground Support Summary

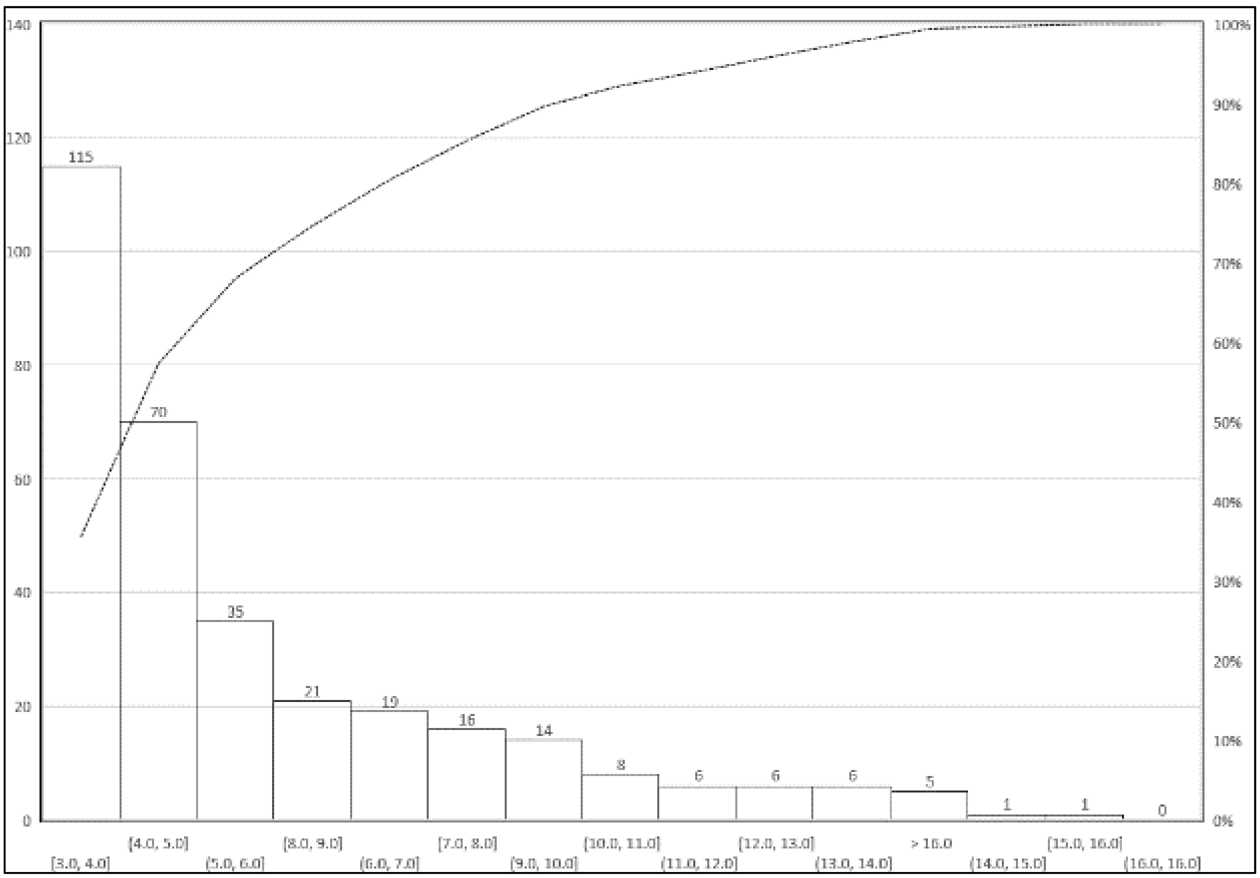
Development Type	Width (m)	Height (m)	Break Length (m)	Ground Support Elements	Coverage
Ramp off-shaft	4.5	5.0	3.8	1.8 m resin rebar on a 1.5 m x 1.5 m pattern plus 10% spot bolting/wastage	100%
				6-gauge mesh (galvanized)	100%
				50 to 100 mm shotcrete no fibre (2-4”)	5%
				4 m x 6 mm thick straps at 1.5 m spacing, back and sidewalls	<5%
Ramp orebody and lateral capital development (access)	4.5	5.0	3.8	1.8 m resin rebar on a 1.5 m x 1.5 m pattern plus 10% spot bolting/wastage	100%
				6-gauge mesh (galvanized)	100%
				50 to 100 mm shotcrete no fibre (2-4”)	5%
				4 m x 6 mm thick straps at 1.5 m spacing, back and sidewalls	5%
Lateral operating (access)	4.0	4.0	3.8	1.8 m resin rebar on a 1.2 m x 1.2 m pattern plus 10% spot bolting/wastage	100%
				6-gauge mesh	100%
				50 to 100 mm shotcrete no fibre (2-4”)	15%
				4 m x 6 mm thick straps at 1.5 m spacing, back and sidewalls	5%
Lateral development intersections	approx. 10 m			2.4 m resin rebar on a 1.2 m x 1.2 m pattern plus 10% spot bolting/wastage	100%
				6-gauge mesh	100%
				Half-span cable bolts on a 2 x 2 m pattern	100%
Ore sills 5 m or less span	Max 5	4.0	3.8	1.8 m resin rebar on a 1.2 m x 1.2 m pattern plus 10% spot bolting/wastage	100%
				6-gauge mesh (not required if spraying fibre-reinforced shotcrete prior to bolting)	100%
				50 to 100 mm shotcrete no fibre (2-4”)	10%
				4 m x 6 mm thick straps at 1.5 m spacing, back and sidewalls	10%
Ore sills 6 m or less span	Max 6	4.0	3.8	1.8 m resin rebar on a 1.2 m x 1.2 m pattern plus 10% spot bolting/wastage	85%
				2.4 m resin rebar on a 1.2 m x 1.2 m pattern plus 10% spot bolting/wastage	15%
				6-gauge mesh (not required if spraying fibre-reinforced shotcrete prior to bolting)	100%
				50 to 100 mm shotcrete no fibre (2-4”)	100%
				4 m x 6 mm thick straps at 1.5 m spacing, back and sidewalls	10%
Ore sills 6 to 8 m span	Max 8	4.0	3.8	2.4 m resin rebar on a 1.2 m x 1.2 m pattern plus 10% spot bolting/wastage	100%
				6-gauge mesh (not required if spraying fibre-reinforced shotcrete prior to bolting)	100%
				50 to 100 mm shotcrete no fibre (2-4”)	100%
				4 m x 6 mm thick straps at 1.5 m spacing, back and sidewalls	10%
Ore sills 8 to 10 m span	Max 10	4.0	3.8	2.4 m grouted rebar on a 1.2 m x 1.2 m pattern plus 10% spot bolting/wastage	100%
				Half-span cable bolts on a 2 x 2 m pattern	100%
				6-gauge mesh (not required if spraying fibre-reinforced shotcrete prior to bolting)	100%
				50 to 100 mm shotcrete no fibre (2-4”)	100%
				4 m x 6 mm thick straps at 1.5 m spacing, back and sidewalls	10%
Stope hangingwalls	Spaced approx. every 5 m where requried along strike			900 x 10 m cables installed from ore sills or HW access drifts	-

Source: North Rock – Ground Support Summary Table, June 2018

Cable bolting in local poorer quality areas, or where hangingwall faults impact stability, may be required. Based on results of stress simulations, it was identified that cable bolting may be required in areas where stoping occurs over a significant strike lengths and spans (i.e. 50 to 100 meters). Approximately 1,000 x 10 m long cable bolts have been included in the mine design that coincide with several poorer quality stress impacted zones (both elevated and distressed) identified with the numerical stress simulation.

16.3.4 Stope Stability

The mine design includes mostly longitudinal stopes with the typical level spacing of 15 meters (floor-to-floor) by 17 meters along strike, and an average width of 5.9 meters. Transverse mining methods are typically included in the design where stoping widths exceed 15 meters. Overall, the mine design includes stope widths varying from 3 to 16 meters. Approximately 68% of these stopes are <6 m wide, and approximately 85% are <9 m wide. Figure 16-11 illustrates the cumulative frequency distribution of proposed stope widths.



Source: North Rock – Gryphon PFS – Preliminary Underground Geotechnics for Mine Planning dated September 2018

Figure 16-11. Stope Width Distribution (meters)

A range of stopes dimensions were reviewed for stability using an empirical open stope design methodology known as Mathews-Potvin, or the Stability Graph Method (Hutchinson and Diederichs, 1996).

To determine the method's N' 'Stability Number', a range of Q' values were estimated using the rock mass quality estimates. For the main deposit areas (lenses A-D), a Q' value of 2 was selected for the 'conservative' case, and a Q' value of 6 was selected for the 'base' case. These are consistent with

RMR' values of about 50 to 60, respectively, and represent the range of rock mass conditions likely to be encountered during typical stope development. For the E series lenses (previously referred to as Upper-D series), Q' values of 0.6 and 2 were assumed (RMR 40-60) for the conservative and base cases, respectively. In the absence of measured in situ stress data, values for Sigma 1 (major principal stress) and related 'A' magnitudes were estimated based on reasonable approximations of relatively low induced stress in the walls, and relatively higher induced stresses in the back (A from 0.3 to 0.7). Discontinuity sets were conservatively assumed to be present and near-parallel to all surfaces (B = 0.2). Gravitational failures were assumed for the stope backs (C = 2) and gravity fall or slabbing failures was assumed for the stope walls (C = 7-8), based on the most probable mode of failure for each.

For the input parameters for the stability analyses, reasonable/likely stress conditions were assumed; the hangingwall under low confinement/low stress, with relatively higher induced stresses in the back. Discontinuity sets were conservatively assumed to be present and near parallel to all stope surfaces. Gravitational failures were assumed for the stope backs, and gravity fall or slabbing failures assumed for the footwall and endwalls, based on the most probable mode of failure for each. Note, the ore zone dip varies between approximately 45° and 60°.

Based on the information available at the time of this study, the optimized recommended stope dimensions for typical stopes are:

- Longitudinal stopes within 60° dipping mineralization:
  - Stope height = 15 meters (floor to floor)
  - Stope width = 15 meters maximum
  - Stope length = 15 meters to 25 meters maximum
- Longitudinal stopes within 45° dipping mineralization:
  - Stope height = 15 meters (floor to floor)
  - Stope width = 15 meters maximum
  - Stope length = 15 meters to 20 meters maximum
- Transverse stopes within areas where mineralization is greater than 15 meters wide:
  - Stope height = 15 meters (floor to floor)
  - Stope width = 15 meters maximum
  - Stope length = 20 meters maximum

In the uppermost proposed mining areas, a higher level of conservatism was adopted given the proximity to the unconformity and poorer rock quality conditions. The recommended maximum stope widths for longitudinal stopes is reduced to 12.5 meters wide, and the recommended maximum stope width and length for transverse stopes are both reduced to 12 meters.

Where mining shapes are stacked or en-echelon, it is recommended to mine the footwall stope lenses prior to the hangingwall stope lenses, to avoid interstitial rib pillar/stope instability issues arising from the alternative case (i.e. if hangingwall stopes are mined first, risk increases for interstitial waste pillar failure, related ground control issues, and dilution within a subsequently mined FW stope). Interstitial rib pillar instability may be due to a combination of stress, structure, orientation, and rock mass conditions. When mined as recommended, footwall to hangingwall, a minimum recommended interstitial rib pillar dimension between stopes of 2.5 meters is provided. If HW lenses are to be mined first, this dimension will need to be significantly increased (8-10 m at a minimum).

All dimensions considered herein assume that good blasting practices will be employed so that damage to the walls and stope backs will be minimized to enhance stability. It is also assumed that the backfill will be of good quality and placed in a timely manner.

Risks to the achievement of the designs presented include undefined large-scale geological structures (particularly if they act as conduits for groundwater), groundwater pressures in weak zones that cannot be effectively de-pressurized, the presence of adversely oriented discontinuities which could impact stope stability in a larger area, and more pervasive and extensive weak ground within mineralized zones than is indicated by current data. This is particularly important within the crown pillar zone between the basin unconformity and the uppermost stope back.

16.3.5 Numerical Simulations

The empirical stability results were further investigated and verified using 2D and 3D numerical stress-strain models. The models are considered suitable for studying the evolution of mining induced stresses and displacements but are not considered suitably calibrated to estimate precise magnitudes of stress or strain or infer the relative degrees of rock mass damage resulting from failing/yielding rock (i.e. micro-cracking, spalling, on-going plastic deformation, and, in the extreme case, rock-bursting).

The model results were reviewed with the aim of highlighting areas where potential damage is concentrating, where stress shadows (loss of confinement, or relaxation) are occurring, and where standoff distances or pillar thicknesses are not sufficient to isolate some critical openings from significant mining-induced stress changes.

The modelling results are used as a guide only, to inform the design process. The models' limited capacity arises from the current level of knowledge regarding geotechnical and hydrogeological conditions, in situ stress conditions, local and intermediate-scale structural features, and geotechnical 'zoning', etc. All of these can significantly constrain the extent to which the model can be used to accurately predict local and global rock mass behavior during the mine life.

16.4 Phoenix Deposit Mining Method – ISR

Development of ISR at Phoenix followed a sequential process for development including:

- Initiation of conceptual level assessments.
- Completion of preliminary laboratory analysis (outlined in Section 13.1).
- Update of the conceptual level assessment based on the laboratory analysis.
- Identification of technical risks and opportunities.
- Due diligence reviews of existing operations, site visits, and technical proceedings.
- Completion of hydrogeological modelling and assessment for ISR application.
- Evaluation of regulatory and environmental requirements for ISR operations.
- Consultation with stakeholder groups, including communities, federal and provincial regulatory agencies, and JV partners.
- Completion of PFS level evaluation of ISR production and economic analysis.

The relevant characteristics of the Phoenix deposit from a mining method selection perspective are provided below.

- In the deposit area, the surface overburden layer ranges in thickness from 20 to 30 m.
- It has been systematically drilled at a nominal section spacing of 25 m x 10 m.
- Zone A is approximately 380 m long, 36 m wide, and 2 to 17 m thick.
- Zone B is approximately 290 m long, 19 m wide, and 3 m thick.
- It is a flat lying deposit, with the deposit gradient only -3% to the northeast over the 700 m strike length of Zones A and B1.
- The deposit sits at the unconformity at depths ranging from 390 m to 420 m and is subject to the water pressures in the overlying sandstone.
- It is an extremely high-grade, high value deposit requiring careful selection of primary mining and recovery methodologies with 85% of the mineralization contained in the Phoenix Zone A high-grade corridor.
- There are some areas of lower grade on the deposit fringes.
- Geotechnical assessment indicates a very weak HW in the Broken Zone domain.
- Ground conditions within the deposit are expected to be variable.
- Rock quality in the basement rock, sufficiently below the unconformity and associated alteration or paleo weathering, will show fair to good rock mass conditions with the presence of fault structures.
- Mineralization/waste contacts are easily visible.
- Mineralization continuity is excellent at all expected cut-off grades due to the high grade of the deposit.

Key features of the Phoenix ISR include:

- Utilization of a low pH mining solution.
- Injection and recovery wells on a 10 m spacing in a hexagonal pattern with the recovery wells placed in the centre of a ring of injection wells.
- A total of 94 recovery wells and 216 injection wells are required for complete coverage of both Zones A and B.
- Use of a freeze wall to ensure separation and maximize the isolation of the mining solution from the regional groundwater.
- Annual production of 6 Mlbs/yr requiring the operation of 10 recovery wells (out of a total of 94 planned).
- Monitoring wells will be installed around the perimeter of the mineralized zone, as dictated by geologic and hydrogeologic parameters.

16.4.1 Wellfields

Wellfields are the groups of wells installed and completed in the mineralized zones that are designed to effectively target delineated mineralization and reach the desired production goals. A wellfield consists of patterns of recovery and injection wells (i.e. the pattern area). The mineralized zones are the geological sandstone units where the leaching solutions are injected and recovered via wells in an ISR wellfield.

This PFS assumes the patterns for the injection and recovery wells will follow a conventional seven-spot pattern consisting of a recovery well surrounded by six injection wells in a hexagonal shape (Figure 16-12 provides an example of a typical ISR wellfield pattern and spacing). Depending on the shape of the mineralized zone, other patterns, including five-spot, line drive, or staggered line drive patterns, may also be used. The dimensions of the patterns vary depending on the mineralized zone, the aquifer transmissivity, etc. The preliminary wellfield design developed for this report assumes injection wells will typically be on the corners of a hexagonal shape 10 meters apart, with a recovery well in the center, leaving 10 meters between the recovery well and each of the six injection wells. Costing has been developed assuming this spacing, but it is anticipated that spacing may vary between 5 and 15 meters apart. In order to effectively recover the uranium, and also to complete the groundwater restoration, the wells will be completed so that they can be used as either injection or recovery wells, allowing flow direction to be reversed at any time during the production or restoration phases of the project.

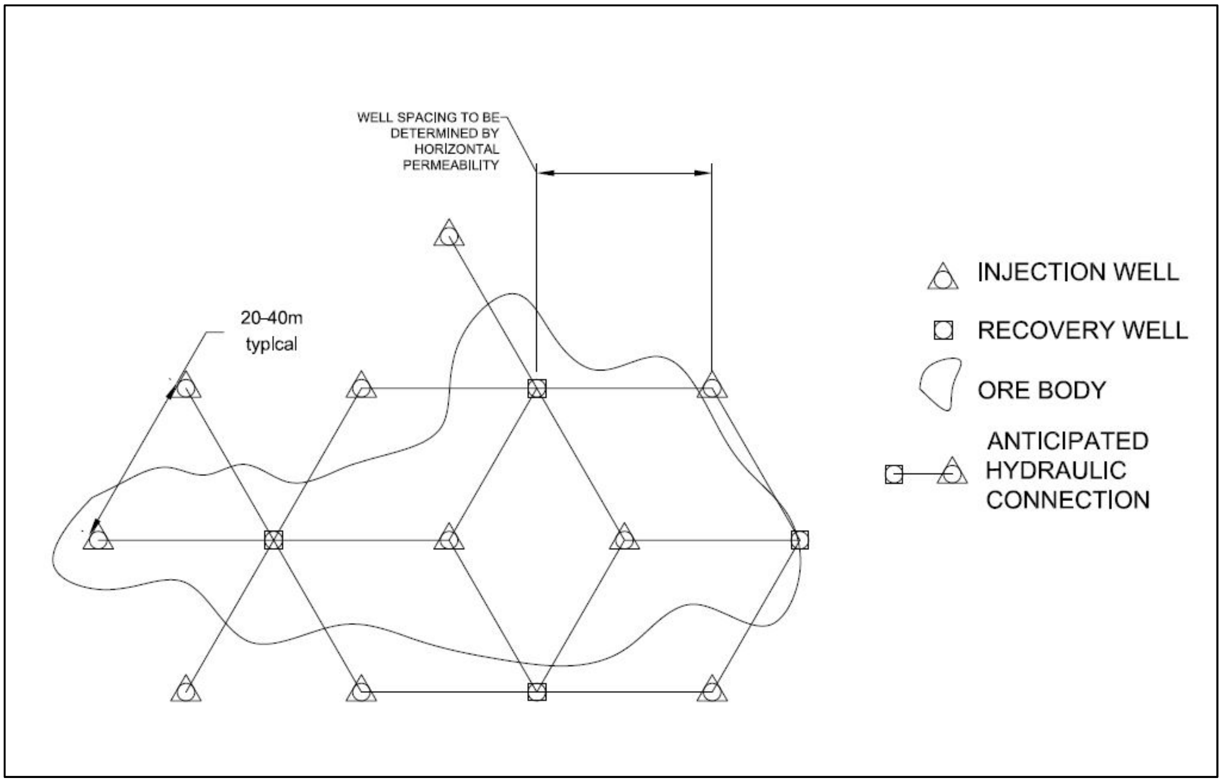


Figure 16-12. Plan View of Traditional ISR 7-Spot Pattern

16.4.2 Freeze Wall

A freeze wall is planned to be established to isolate the deposit from the surrounding sandstone and regional groundwater.

Typical ISR operations have an aquitard above and below the mining zone, naturally isolating mining from the surrounding groundwater. The Phoenix deposit does have a low permeable boundary below, the basement rock. However, above the deposit is the highly altered and permeable sandstone. By inducing a freeze cap just above the deposit and keyed into the basement rock 360 degrees around the deposit, an aquitard equivalent will be created in the sandstone surrounding the Phoenix deposit.

Ground freezing is typically applied in civil or mining applications where there is concern about both water ingress and ground stability during excavation through water bearing formations. For Phoenix ISR mining, the ground freeze concept will be applied for similar reasons: to control groundwater ingress, to control lixiviant egress (also referred to as an excursion), and to provide ground stability following potential volume loss within the higher-grade portions of the deposit.

Mitigating groundwater ingress will provide operational advantages, as dilution of the lixiviant being circulated between injection and recovery ISR wells will be effectively eliminated, thus permitting good control on volume throughput and reagent consumption in the recovery plant.

Preventing excursions of lixiviant will provide economic and environmental advantages. By maintaining control of the lixiviant inside the freeze wall, recovery of the pregnant solution will be maximized, and impact on the natural groundwater outside the freeze wall will be prevented. As well, the freeze cap will simplify the process of restoration by controlling and minimizing the volume of ground impacted by ISR mining.

There is a possibility that, due to the high grade of the deposit, the high recovery of ISR, and the quantity of material that will be dissolved into the lixiviant, ground movement due to volume loss may occur underground. To date, test work does not provide evidence of this. However, it is envisioned that the freeze cap will provide ground support to ensure that any potential ground movement due to volume loss will not propagate and negatively affect mining.

16.4.3 Freeze Design

Ground freezing involves the process of circulating a chilled brine through a pattern of holes drilled through the ground. The brine is contained within the holes via double casing, where the brine is injected within the inner casing and returns along the annulus between the inner and outer casings. The chilled brine extracts heat from the surrounding rock by forced convection with the rate of heat transfer being a function of brine temperature, flow rate, and ground thermal properties.

Ground freezing for water control and ground stability enhancement is a relatively low risk procedure and is common practice in the Saskatchewan uranium and potash mining industries for ground excavations in ground conditions similar to those above the Phoenix deposit. For Phoenix, the primary objective is to create a hydraulic barrier wall, and as such, the freeze criteria were developed to ensure there is frozen wall thickness to prohibit passage of water. Based on the fact that there will be no excavations adjacent to the freeze cap and no people working near the freeze cap, the freeze cap will be considered formed to isolate the deposit once the cap is at least 3 metres thick and ground temperature colder than -2°C is present.

The COMSOL Multi-physics 2D/3D finite element program was used to optimize the freeze cap design. The freeze cap design includes an arched configuration of 30 holes that will run parallel to the mineralized zone, approximately 27 metres above the unconformity. The left and right sides of the arch will be drilled along the strike of the mineralized zone and just inside the basement rock. This serves to key the toes of the arch to the low permeability basement rock. The drill holes will be spaced 5 metres apart along the perimeter of the arch. The holes will be drilled via directional drilling and collared vertically from surface at either end of the deposit. The vertical holes will be curved laterally to follow the strike of the deposit along the arch pattern for 900 metres. The end of the holes will be curved down to key the ends of the freeze cap down into the basement rock (Figure 16-13).

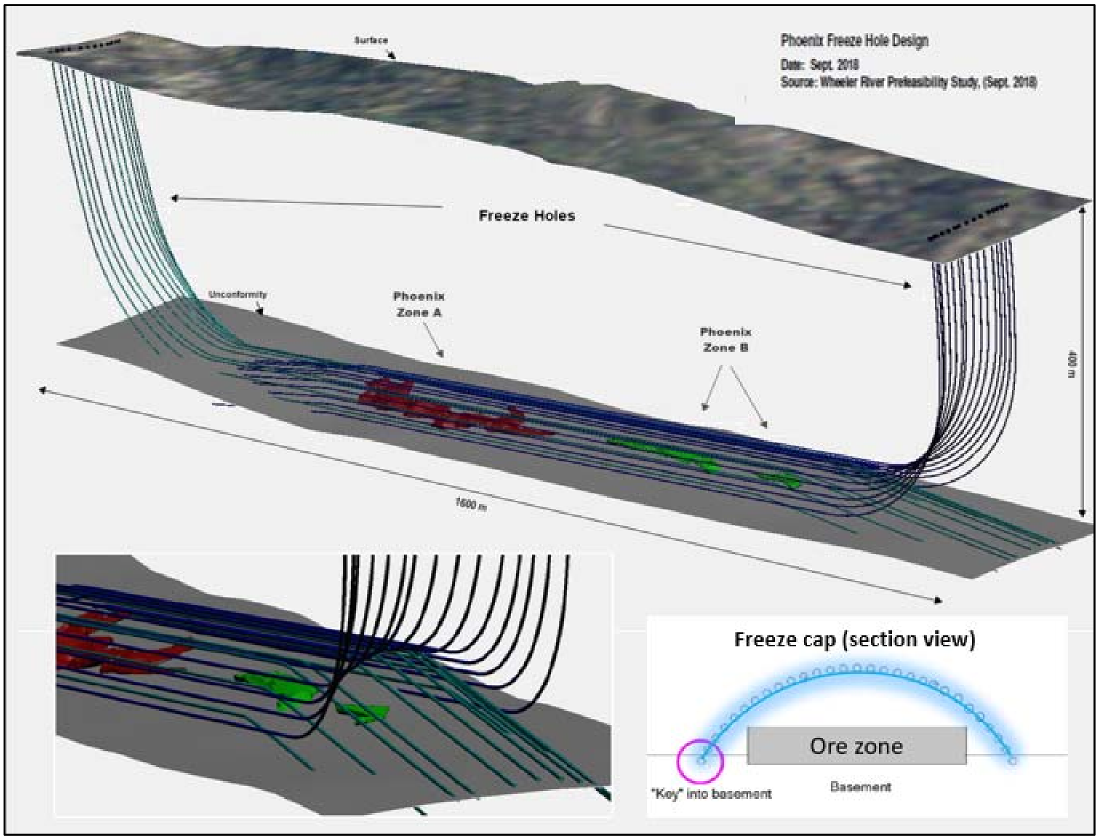
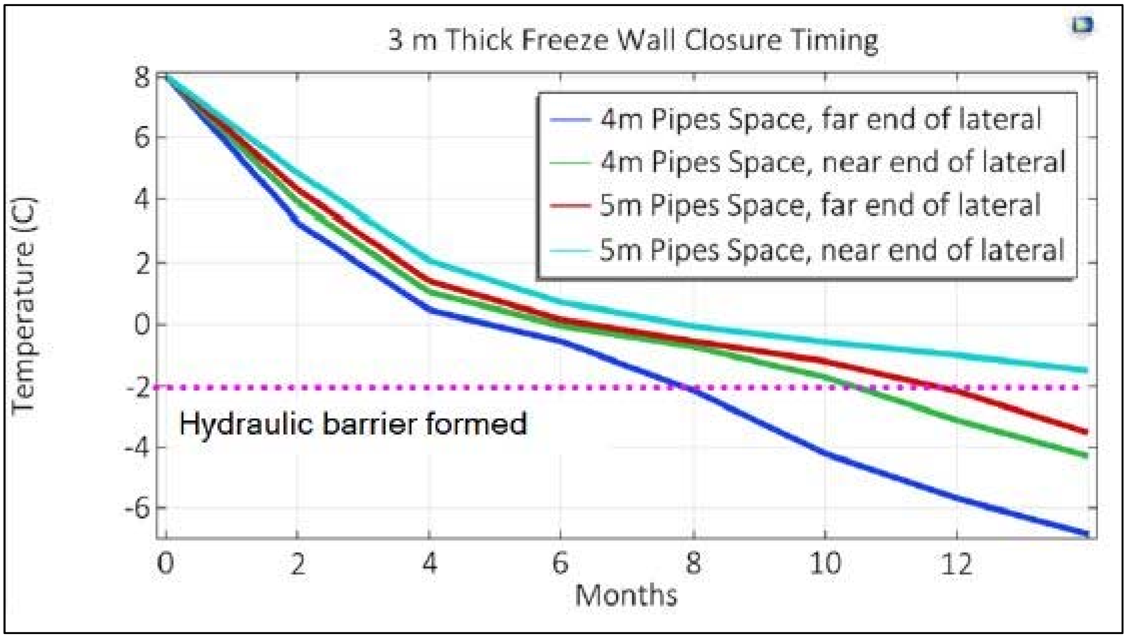


Figure 16-13. Freeze Cap Design

The installed piping within the holes will be at least L80 grade steel with "VAM TOP" couplings to provide 100% efficient joints. The outer casing will be 5-inch diameter and the injection pipe will be 3.5-inch diameter. The brine is assumed to be calcium chloride, 30% by weight, operating at a plant heat exchanger brine cold side temperature of -35°C. The brine flow rate will be 8 cubic metres per hour (35 usgpm), which allows for a higher heat coefficient, an achievable brine fluid velocity in the main header line that services the hole collars and freeze plant, and minimized brine warming along the 900-metre lateral pipe section where heat is removed from the ground. A freeze plant will be required at each end of the deposit where the freeze holes are collared, with the combined capacity of both freeze plants at 1500 tons of refrigeration. Based on this design criteria, modelling predicts a freeze wall will require 14 months to be established (Figure 16-14).



Source: Newmans geotechnique Inc. – Wheeler River In-Situ Leach Freezing Pre-Feasibility, May 17, 2018

Figure 16-14. Freeze Wall Closure Modelling

The freeze plant system included in this design is "modular", which means that a shutdown in any one unit will not result in a complete plant downtime. Once closure is achieved, when the ground is frozen between holes, and the frozen barrier grows in thickness, there is often enough thermal inertia in the system to allow for a mechanical shutdown of an extended duration. This means that natural thawing will be a long, slow process, especially in high water content zones like above the Phoenix deposit where significant amounts of latent heat must be added to facilitate phase change.

Outside the perimeter of the freeze wall (both on the side and above), there will be a series of monitoring wells. These wells will be used to monitor natural groundwater conditions during operations. For the PFS, monitoring wells were designed on a 130-meter perimeter around the freeze wall. The spacing of the wells will need to be evaluated in the future based on hydrogeological assessments.

16.4.4 Drilling Methodology

At present, the drilling of individual wells will be carried out utilizing either air rotary or mud rotary methods. A typical recovery well design was provided by Woodard & Curran Inc. (Figure 16-18) and modified by Denison to suit the project's specific needs. Guidance on drill methodology was provided by ARTISAN Consulting Services Ltd. and subsequently modified by Denison.

No detailed layouts were completed to assess the potential intersection of recovery and injection wells with the freeze holes. However, accuracy will be a key drilling consideration during project execution to avoid operational problems with the freeze cap.

Total depth of all monitoring, recovery, and injection boreholes was set to 430 meters to ensure complete penetration of the Phoenix mineral resource. Future considerations should be taken to customize individual boreholes to tailor the individual depths.

The total number of recovery, injection, and monitoring wells are tabulated and illustrated in Table 16-6 and in Figures 16-15, 16-16 and 16-17 below for the Phoenix Zone A and B mineral resource.

Table 16-6. Cumulative Wells and Meterage for the Phoenix Mineral Resource

Well Type	Number of Wells	Meterage
Zone A Recovery Wells	64	27,520
Zone B Recovery Wells	30	12,900
Total Recovery Wells	94	40,420
Zone A Injection Wells	131	56,330
Zone B Injection Wells	68	29,240
Total Injection Wells	199	85,570
Monitoring Wells	17	7,310
Total	**310**	133,300

Source: Denison Mines – Phoenix – Wellfield Design Memorandum, July 6, 2018

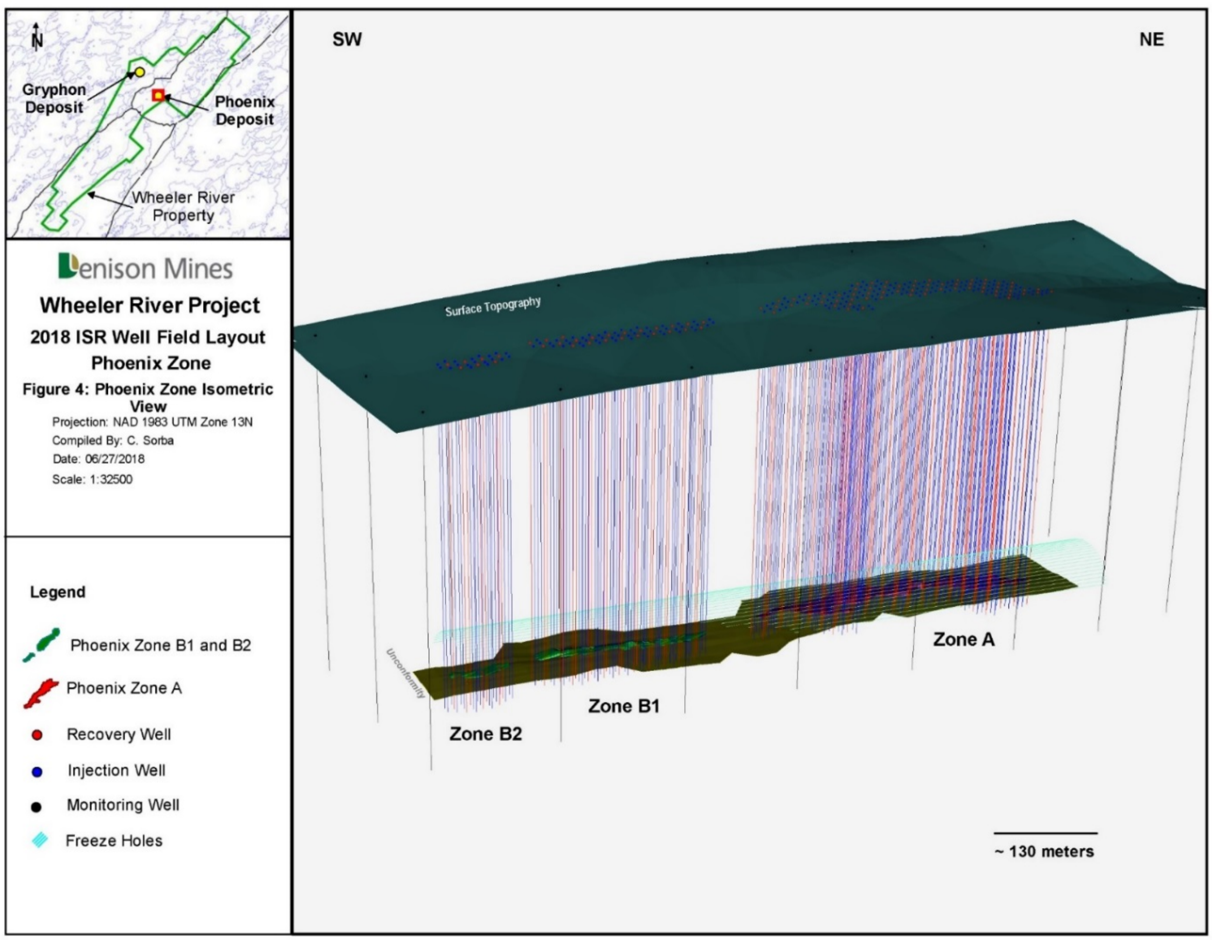


Figure 16-15. Isometric View of Phoenix Wellfield Layout

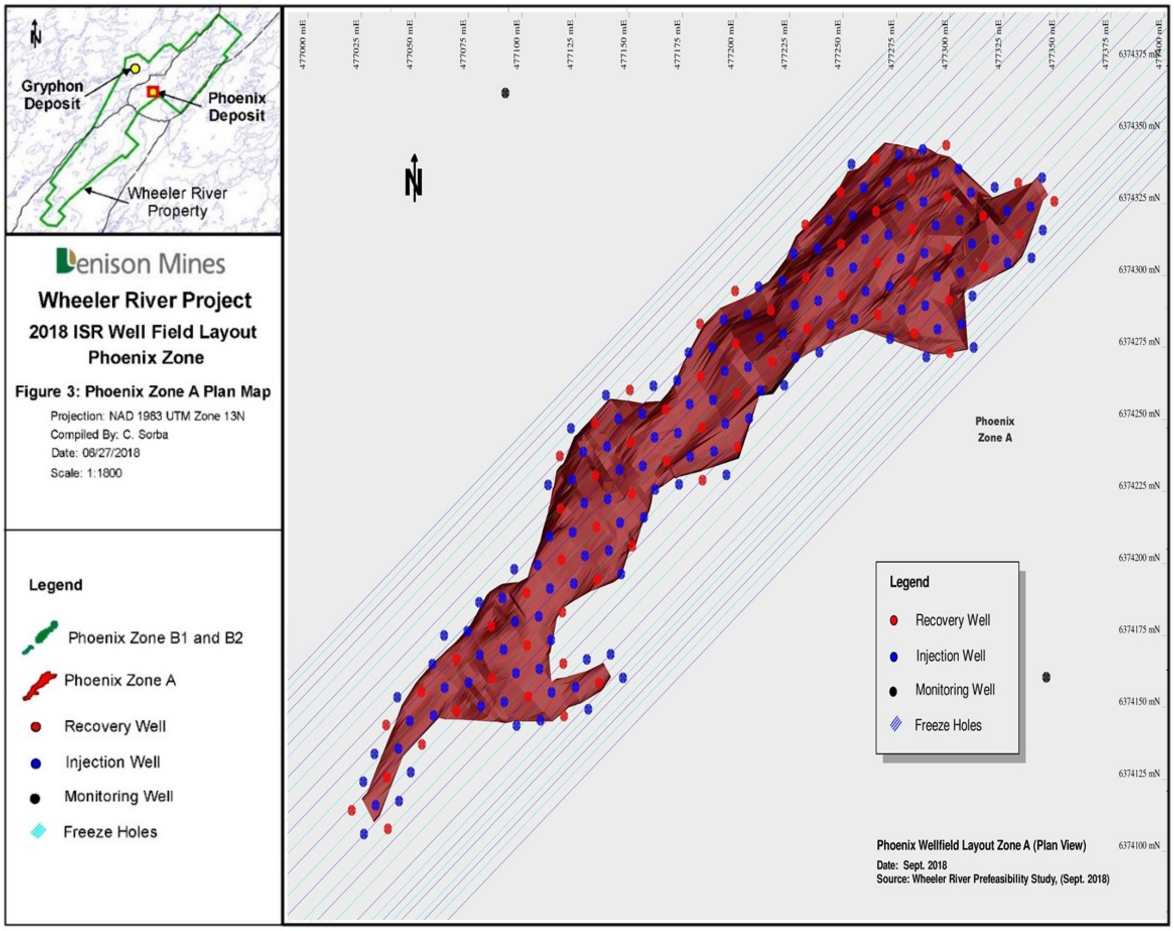


Figure 16-16. Phoenix Zone A Wellfield Layout

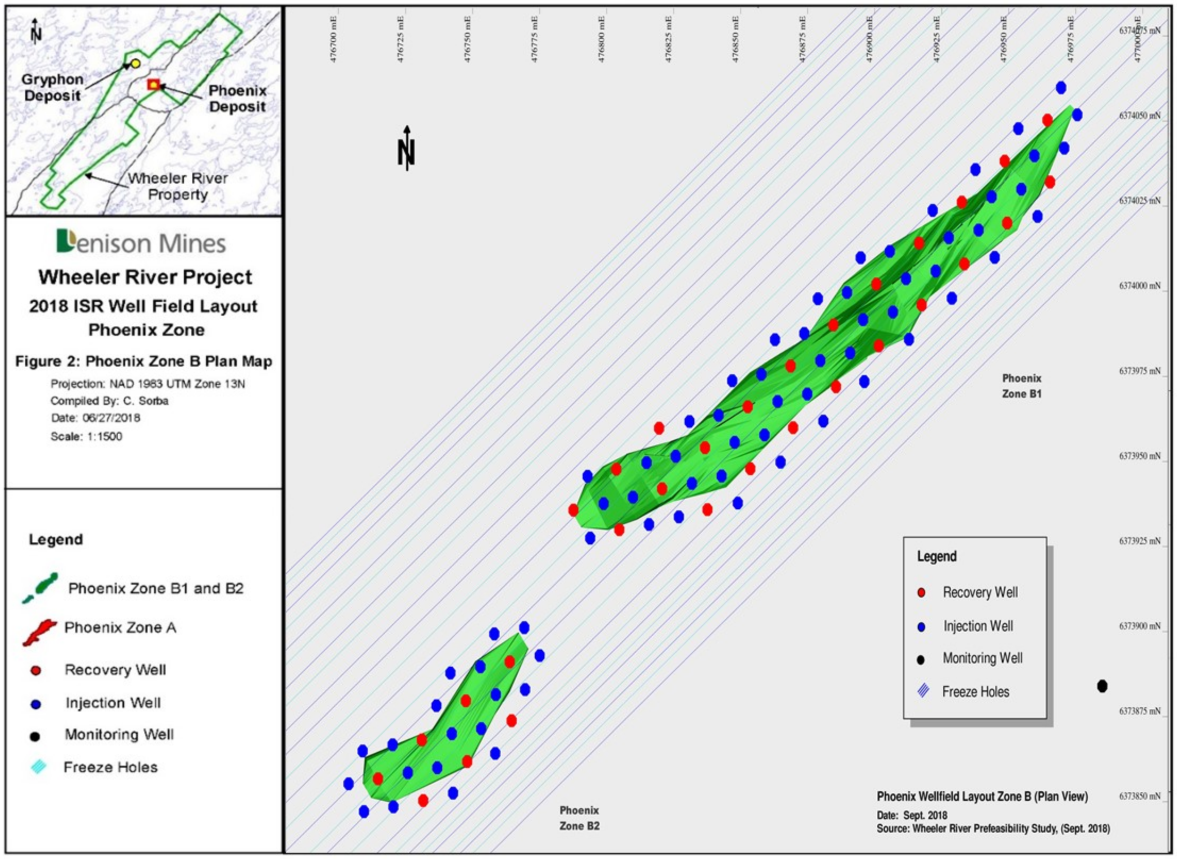


Figure 16-17. Phoenix Zone B Wellfield Layout

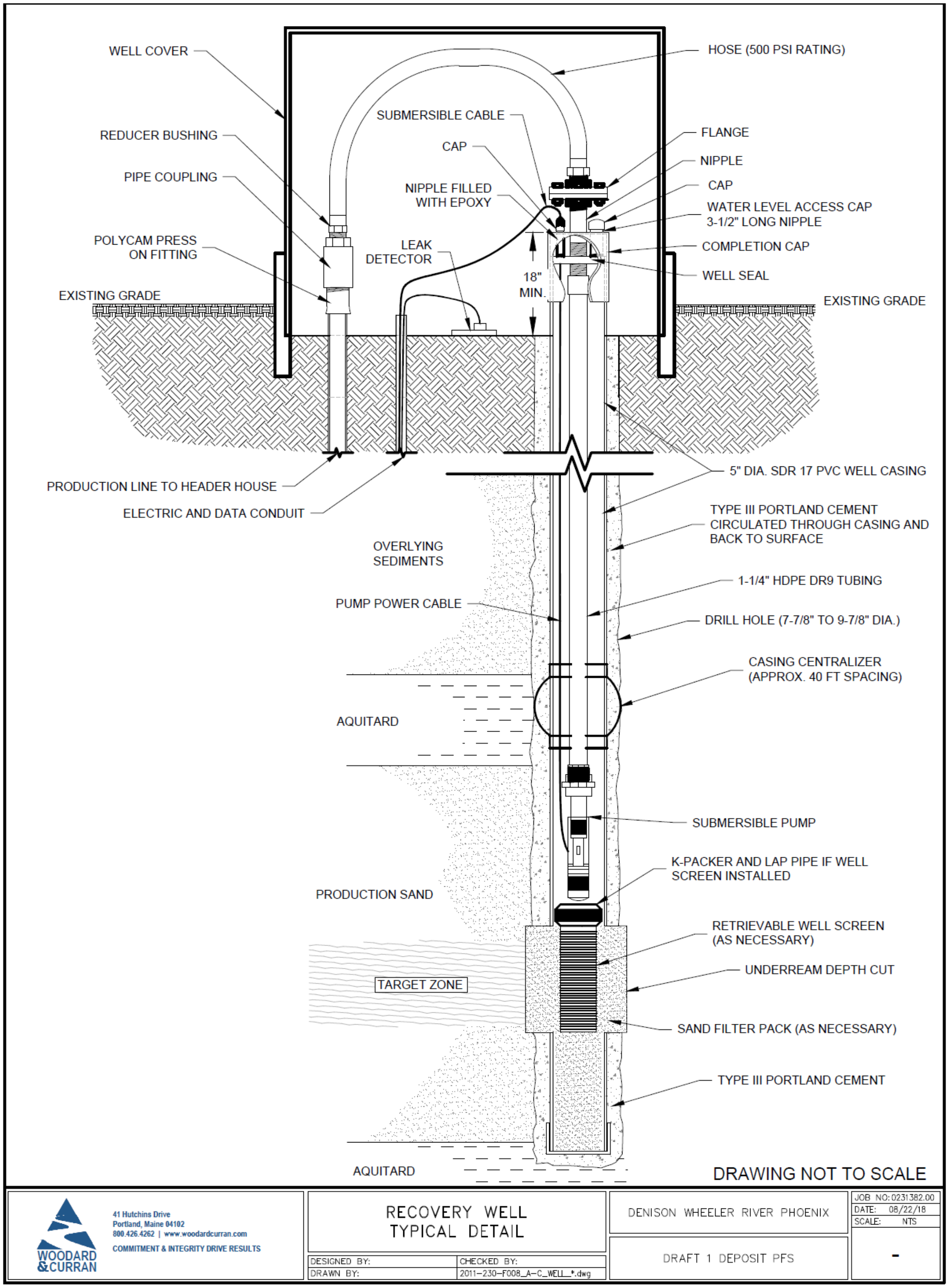


Figure 16-18. Typical Recovery Well Detail

16.4.5 Mechanical Integrity Testing

After an injection, recovery, or monitoring well has been completed, and before it is made operational, a mechanical integrity test (MIT) of the well casing is completed. In the integrity test, the bottom of the casing adjacent to or below the confining layer above the zone of interest is sealed, as is the top of the casing, and a pressure gauge is installed to monitor the pressure inside the casing. The pressure in the sealed casing is then increased to a specified test pressure and must maintain 95% of this pressure for a specific duration of time to pass the test. Well casings that fail the integrity test will be repaired.

16.4.6 Production

The uranium ISR process proposed in this PFS will involve the dissolution of the water-soluble uranium compound from the mineralized host sands at low pH ranges using acidic solutions. The acidic solution will dissolve and mobilize the uranium, allowing the dissolved uranium to be pumped to the surface. The uranium-rich solution will be transferred from the production wells to the nearby precipitation plant for uranium removal, drying, and packaging (Section 17.1). As the entire mineral resource will be isolated from the surrounding aquifer by the freeze wall, production flow rates are anticipated to be equal to injection rates.

16.4.7 Wellfield Piping System

Pipelines will transport the wellfield solutions to and from the precipitation plant. The flow rates and pressures of the individual well lines will be monitored in headerhouses. This data will be transmitted to the precipitation plant for remote monitoring through a master control system. Through the master control system, the user will be capable of controlling headerhouse production lines remotely. Double contained high density polyethylene (HDPE) piping (or equivalent) will be used in the wellfields and will be designed and selected to meet design operating and environmental conditions.

The lines from the processing plant, headerhouses, and individual well lines will be freeze protected and secured to minimize pipe movement. Figure 16-19 illustrates the approximate locations for the processing plant, trunkline, and headerhouses.

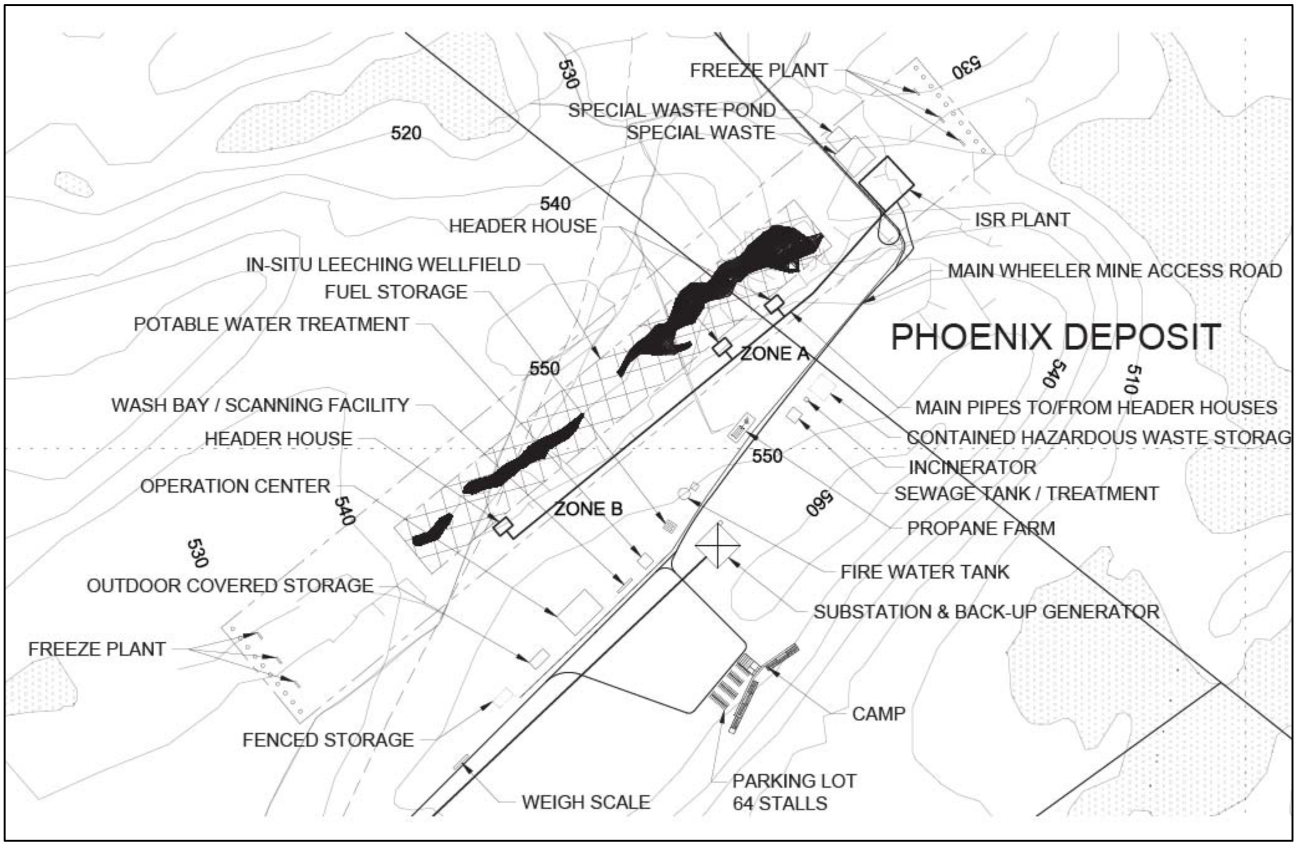


Figure 16-19. Phoenix Plan View Showing Plant, Headerhouses, and Well Lines

16.4.8 Headerhouses

Headerhouse buildings (headerhouses) will be used to distribute the mining solution to injection wells and collect the pregnant solution from recovery wells. Each headerhouse will be connected to two production trunk lines. One of the trunk lines will be used for receiving barren mining solution from the precipitation plant and the other will be used for conveying pregnant solution back to the precipitation plant. The headerhouses will include manifolds, valves, flow meters, pressure meters, and other instrumentation, as required, to fully operate and control the process. This monitoring and control of the system allows the operators to individually adjust each recovery or injection well.

Each headerhouse will service approximately 80 wells (injection and recovery), depending on resource and pattern configuration. Table 16-7 presents the current anticipated headerhouse and well summary by zone.

Table 16-7. Phoenix Deposit Headerhouse and Well Inventory by Production Unit

Item	Production Unit	
	Zone A	Zone B
Headerhouses	2	1
Injection Wells	64	30
Recovery Wells	131	68
Monitoring Wells	12	5

Source: W & C – Past Experience from other ISR Projects

16.4.9 Wellfield Reagents, Electricity, and Other Consumables

The wellfield production has been targeted at 6 Mlbs/year. Due to the consistent production rate and consistent nature of the orebody, wellfield reagents, electricity, and other consumable costs are expected to be consistent each year. Reagents, electricity, and other consumables have been estimated based on this production rate and have been included in the annual operating costs.

16.4.10 Mining Unit Design, Production Rates, and Mine Life

The mining approach is governed by the rate of mineral extraction and the duration of the mine development, mineral extraction, processing, and closure. The following describes each of these mine development and operation components.

The Phoenix deposit is divided into two areas – Zone A and Zone B. A combination of wells will be operated concurrently to provide the overall flow to the processing plant. Initially, this flow rate will be up to 500 L/min.

The development plan is subject to change due to extraction schedules, variations with production area recoveries, plant issues, economic conditions, etc. Uranium recovery head grade, or concentration, of the uranium bearing solution is assumed to average 10 g/L over the entire production schedule.

Column leach testing showed substantial uranium recovery within the first two pore volumes of lixiviant solution. Due to this, it is estimated that uranium recovery will begin immediately upon injection of lixiviant solutions into the mineralized zone. Once the mining solution passes through the mineralized zone and reaches the recovery wells, production of uranium in the precipitation plant will begin. Low pH ISR operations will sometimes circulate a more concentrated mining solution to pre-condition a wellfield prior to flows from that area being directed to the precipitation plant. However, at the Phoenix Deposit, due to the low flow rate required, it is anticipated that these initial flows will be directed to the precipitation plant, allowing for the quick ramp-up of production.

Production is expected to begin at 2.4 Mlbs in year one, ramping up to full production in years 2 through 11, with total production expected to be 59.7 Mlbs. Figure 16-20 shows total annual production over the life of mine for Phoenix.

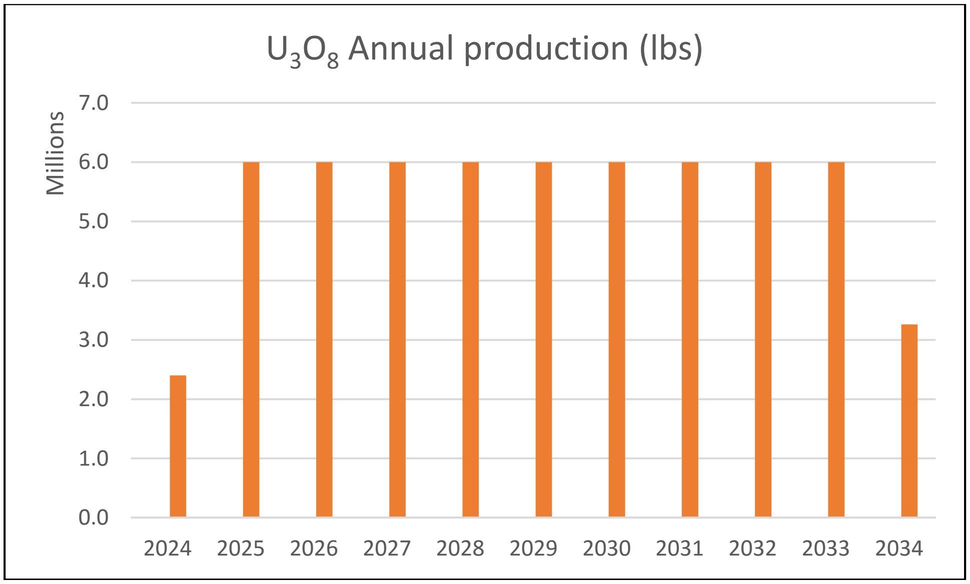


Figure 16-20. Phoenix Production by Year

The authors have estimated the mine life based on head grade, estimated resource, flow rates, and closure requirements for the Phoenix deposit. Production will occur across the Phoenix deposit over a period of approximately 11 years. Restoration and reclamation will be implemented following production and will continue approximately 5 years beyond the production period. The overall mine life of the Phoenix deposit is approximately 18 years from initiation of construction activities to completion of restoration and decommissioning/reclamation.

16.4.11 Mine Development

The first step in the development of the Phoenix deposit will be the drilling and installation of the freeze cap piping system that surrounds the deposit. Concurrent to drilling operations, the freeze plant will be constructed. The freeze plant and piping system will need to be in operation for approximately 14 months to develop a sufficient frozen barrier between the Phoenix deposit and the surrounding sandstone. All infrastructure for mining the Phoenix deposit will be established during the initial construction period with the exception of the wellfield. Wells will be established on an annual basis as required. The physical size and nature of the Phoenix deposit allows all infrastructure to be installed early in the project.

As patterns across the Phoenix deposit reach their economic limit, the production flow in that pattern will be terminated and other patterns will be brought into operation. It is also possible that flow will be adjusted throughout the deposit to meet production targets, and some areas of the deposit may need to have flow shut in temporarily to allow a longer period of time for reactions between the uranium and the mining solutions to occur.

16.4.12 Mining Equipment

Equipment for establishing the wellfield and drilling the wells are standard rotary drill rigs. In addition to drilling equipment, wellfield operations will also utilize submersible pumps, and each well will be equipped with a wellhead assembly, with appropriate valves and other instrumentation to facilitate flow in either direction and for operations monitoring.

Ongoing operations and maintenance activities will use moderately sized mobile equipment for testing and maintenance, such as a light duty crane and a front-end loader.

16.5 Gryphon Deposit Mining Method – Underground

Primary design considerations for mining the Gryphon deposit are provided below.

- In the deposit area, the surface overburden layer ranges in thickness from 20 to 30 m.
- Production from Gryphon will target 9.0 Mlbs of U_3O_8 per year based on expected mill capacity limitations. Production gaps during ramp up and down at Gryphon will be filled with low-grade development material (marginal ore).
- True thickness of individual resource wireframes ranges from 1.8 m to 4.0 m, averaging about 3 m. In some areas, the lenses are close enough together to be impacted by adjacent lens mining. Minimum mining width will be 4 m.
- The moderate grade of the mineralization will not require special mining methods (i.e. conventional mining methods can be used).
- Mineralization/waste contacts are easily visible.
- Mineralization continuity is good at the cut-off grade.
- Minimize the risk of uncontrolled water inflows to the underground mine. Design the system to contain and remove the peak water inflow scenario. The deposit is located well below the unconformity and high-pressure groundwater will not likely be encountered.
- Minimize up-front and total underground mine capital costs and minimize hoisting of waste.
- Maximize recovery of the economically mineable ore resource while maintaining regional ground stability. Geotechnical assessment indicates generally fair to good rock mass conditions. Minimize the requirements for sill pillars.
- Design the ventilation system to support the underground mobile equipment fleet, with capacity to ventilate areas with deferred production and facilities with potential radiation exposure.
- Waste rock generated underground will fall into two categories: clean waste and special waste. The special waste will contain low level radiation but must be treated if brought to surface. The mine will be designed to minimize the amount of special waste hoisted to surface.
- Meet or exceed legislative requirements with respect to safety and the work environment.

The Gryphon deposit will be a shaft access underground mine using the longhole stoping mining method to extract the resource. Opportunities to share common site facilities, services, and personnel with Phoenix have been utilized in this study and are further discussed in other sections of this report.

Gryphon will be accessed via two shafts from surface, namely a production shaft and a ventilation shaft. Blind boring was determined to be the most advantageous shaft excavation method due to favourable cost and schedule implications, safety performance, and low overall risk profile.

The Gryphon mine design includes a full-service 5.0 m finished diameter production shaft and a 4.5 m finished diameter ventilation shaft to support underground development and production. Heated fresh air will be delivered via the production shaft, with return air exhausted up the ventilation shaft. An emergency hoist/conveyance will be installed in the ventilation shaft.

The Gryphon resource extends from the -2 m elevation to the -306 m elevation. A minimum 25 m pillar will be left below the unconformity, with the first underground longhole mining level located at the 567 Level or -32 m elevation (E lens). There are no current plans to recover the resource in the 25 m pillar.

Underground production will be from the longhole stoping mining method, primarily longitudinal retreat. Mined stopes will be backfilled using a combination of rockfill, cemented rockfill, and hydraulic fill. The hydraulic fill will be directed to the empty stopes by means of boreholes and pipelines. Waste rock and cemented rockfill will be directed to the stopes via underground haulage trucks and LHDs.

Ore will be truck hauled to a rockbreaker/grizzly station on the 500 Level near the production shaft and hoisted to surface. To minimize the requirement for hoisting and storing waste rock on surface, where possible, waste rock will be disposed of in available stopes either as rockfill, cemented rockfill, or mixed with the hydraulic fill. Mine operating activities and costs assume that, once sufficient open stopes are available, no waste will be hoisted to surface. The underground mine is expected to produce approximately 605 tonnes per day of ore and an average of 330 tonnes per day of waste rock during the steady state operating period.

The underground mine is considered to include:

- Maintenance facility, with 3 maintenance bays, a wash bay, tire and parts storage, electrical maintenance shop, fuel/lube bay, office, and lunchroom;
- Electrical substations and power distribution;
- Portable and permanent compressed air plants;
- Explosives and detonators storage facilities;
- Shotcrete receiving and mixing station;
- Materials storage;
- Ballast crusher station;
- Rockbreaker and grizzly station, complete with remucks for batching ore and waste during early operations and ore and low-grade material during steady state;
- Dewatering sumps and pumps (designed for average run of mine dewatering requirements and major inflow scenario);
- Mine ventilation facilities, including fresh air and return air raises, fans, and controls; and
- Refuge stations and latrines.

All services to and from the underground mine include:

- Electrical power distribution from a 25 kV power line;
- Process water distribution (process water will be stored in a tank with surge capacity and a reservoir for the fire protection system; note that potable water will be provided as bottled water, and water from the wells that is used for wash water will first be treated with chlorine);
- Mine water discharge lines will include 3 x 300 mm (12 inch) schedule 80 piping installed in the production shaft with 3 additional (redundant) 300mm pipes installed in the ventilation shaft;
- All underground mine capital and operating requirements; and
- Underground mine direct and indirect operating personnel, up to and including the underground mine superintendent (this includes the provision of maintenance,

safety/training, and technical staff, assuming there will be minimal sharing of staff between the Gryphon and Phoenix projects).

16.5.1 Mine Development and Contracting Strategy

In order to maximize Gryphon value while reducing capital requirements, Gryphon has been scheduled to start after one year of production from Phoenix. This strategy minimizes the upfront capital cost of Gryphon and allows Denison to utilize cashflow from Phoenix to fund Gryphon development.

Surface construction, shaft sinking/equipping, and off-shaft excavations will be completed using qualified contractors. Experienced construction personnel, tradesmen, and underground miners are available locally.

Once the shafts and initial off-shaft development is completed by the mining contractor, company development crews will complete the remaining development.

During the initial two years of off-shaft development, there will be a major period of underground construction, mostly on the 500 Level. This work will be completed by qualified contractors. Denison construction crews will complete the remaining ongoing construction.

A requirement for all contractors will be to maximize the use of regional personnel. Denison will attempt to integrate regional personnel into the mining crews to the maximum extent possible. Formal training programs will be implemented during the project period and continue through operations. The training programs will help offset productivity losses due to the traditional high turn-over rates in this region.

16.5.2 Mine Operating Strategy

Denison will operate the mine using management and administration staff, technical support staff, supervision and safety/training staff, and operating and maintenance personnel. The mining contractor will be retained to provide raise development services.

The mine will operate on two 10.5 hour shifts per day, seven days per week, for 362 days per year. Although the crews will be paid based on a 10.5-hour shift, the effective work hours will be somewhat less, as 30 minutes have been allowed at the start and end of each shift for travel time to and from the workplace, and 15 minutes have been allowed each shift for crew safety meetings.

The majority of personnel will rotate to/from site on a two-weeks in, two-weeks out schedule.

16.5.3 Underground 3D Mine Model

A 3D mine model was prepared for the Gryphon stopes and development. An isometric view of the model is shown in Figure 16-21.

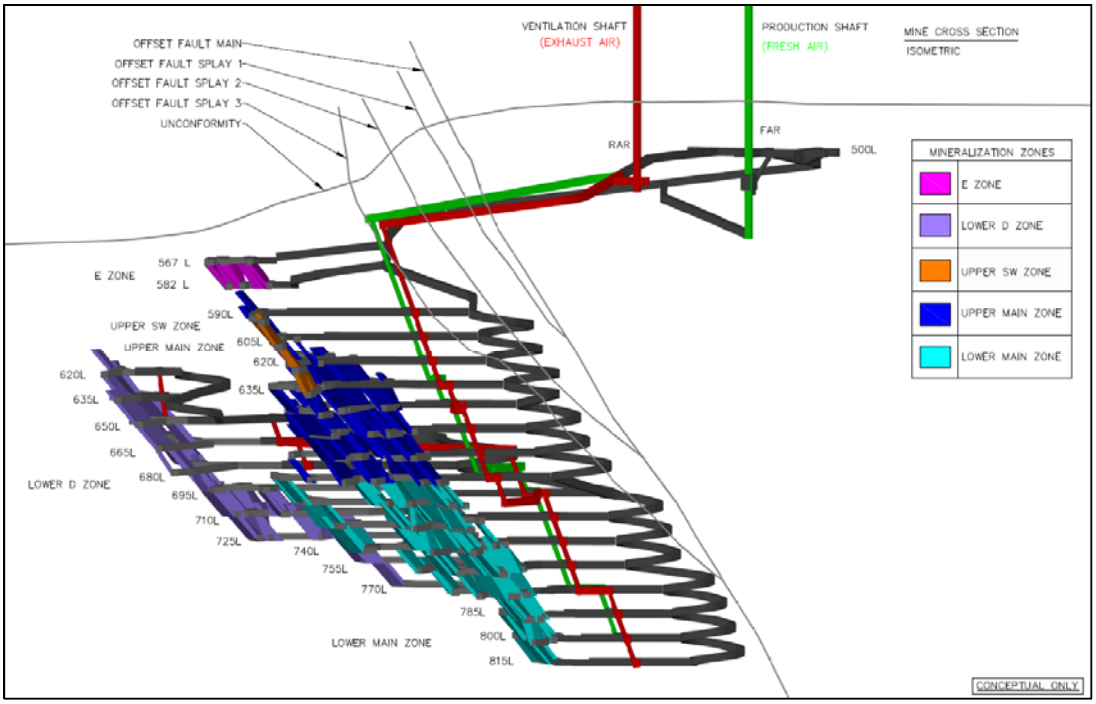


Figure 16-21. Isometric View – Gryphon 3D Mine Model (Looking North)

Access to the underground mine will be via two shafts, namely the production shaft and the ventilation shaft.

Production Shaft

Primary access to the deposit will be by via a production shaft connecting surface to the main shaft station at 500 Level. The shaft will be used to transport personnel and materials underground, to hoist blasted material to surface, and to provide the main fresh air route for the mine. The shaft will be excavated to a depth of 550 meters from surface at a diameter of 6.1 meters using a blind boring method. On completion of a water tight steel/concrete composite liner, the final inside shaft diameter will be 5.0 meters. Refer to Section 18.6 for additional details.

Ventilation Shaft

The ventilation shaft will be excavated using the same blind boring method as the production shaft. In the case of the ventilation shaft; however, the shaft depth will be 500 meters from surface at a diameter of 5.8 meters. On completion of a water tight steel/concrete composite liner, the final inside shaft diameter will be 4.5 meters. Refer to Section 18.7 for additional details.

Underground Design

Access from the shaft to the mine working will be via a single ramp to be developed from the 500L (shaft station) to the 815 Level. The main haulage ramp will be located on the hangingwall (HW) side of the deposit. Each mining sublevel is connected to an internal fresh air raise and an internal exhaust raise. The fresh air raise will serve as a second means of exit from the sublevels. The Gryphon deposit plunges to the northeast and the access ramp is designed to follow the plunge. Short sections of ventilation transfer drifts are included in the design to allow the ventilation raise systems to follow the plunge.

The haulage ramp will be driven 4.5 m wide by 5.0 m high at a typical grade of -15%. The ramp grade will be reduced to -5% at major intersections (access to levels and truck loading stations). The ramp will be used to provide access for personnel and materials from the shaft to the mine workings, movement of mining equipment from level to level within the mine, and ore/waste haulage to the rockbreaker station near the shaft. All ramp development will include the installation of long-term ground support (grouted rebar bolts, screen, and grouted cable bolts).

Mining levels will be located at 15 m vertical intervals with the first level located at the 590 Level. During the initial ramp development to the 590 Level, the E Zone mining block will be accessed to provide early production and a location to dump special waste material, to allow the setup of the ballast crusher station, and to establish an emergency overflow sump and pump station.

On the 500 Level (shaft station), an access drift will be excavated to connect the production shaft to the ventilation shaft, providing early flow through ventilation and secondary egress from the mine. The major underground infrastructure will be located on the 500 Level between the two shafts (Figure 16-22). The access drifts will be extended beyond the ventilation shaft to provide a lateral transfer of fresh air and exhaust air for the mine.

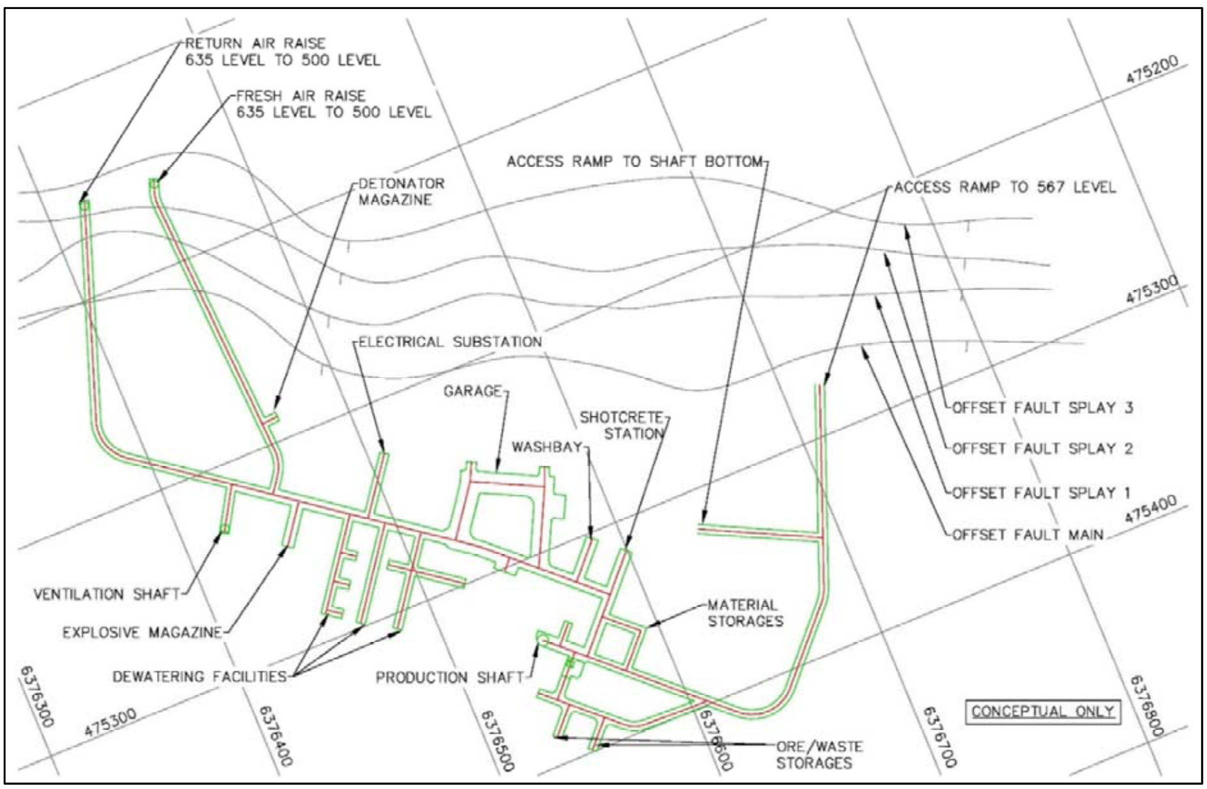


Figure 16-22. 500 Level and Related Infrastructure

Production level access drifts will be driven 4.5 m wide by 5.0 m high, at a nominal grade of 3% from the access ramp to the mineral reserve. All level development will include the installation of permanent ground support (grouted rebar bolts, screen, and cable bolts).

All ramp, lateral level development, and infrastructure excavations will be considered as capital.

The mine has been divided into five mining blocks: E Zone, Lower D, Upper and Lower Main, and Upper SW. Each mining block will be further sub-divided based on early access and geometric constraints, and to reduce the number of sill pillars. Each mining block will be mined from the bottom up with longitudinal retreat to a central access, with any sill pillar at the top being removed after completion of production from the lower levels of the mining block above. This will typically require redevelopment of the sill pillar stope top-cut drifts, as the bottom-cut (mucking) drift for the previously mined stope above will have been backfilled.

Stope over-cut and under-cut drifts will be driven at an average width of 5.9 m wide by 4.0 m high. All drifts will be driven at a grade of 3% to provide positive water drainage from the stopes. The drift width is dictated by the average stope width, to a maximum of 8 m wide. The drift height is required to allow efficient drilling of blastholes and to allow installation of ventilation ducting in the under-cut (mucking) drifts. The stope over-cut and under-cut drifts will include 100% shotcrete coverage and 150 mm of ballast on the floor to reduce the potential for radiation exposure.

Longhole stopes will be backfilled as soon as possible. During the early years of the project phase, sufficient quantities of waste rock will be produced to fill the stopes with cemented rockfill. Later in the mine life, when waste rock generation is reduced, there will no longer be sufficient waste produced underground to satisfy the backfill requirements, and hydraulic backfill will be used. Hydraulic backfill will be delivered to the underground stopes via boreholes and pipelines. The backfill system is reviewed in more detail in Section 18.22.

LHDs will muck broken material from stopes and load 30 tonne capacity haul trucks at truck loadouts established near the ramp. The trucks will haul to the rockbreaker/grizzly station located on the 500 Level. The ore will be hoisted to a surface pad and subsequently hauled to the ore stockpile located near the shaft. A surface contractor will transfer the ore to the mill, approximately 175 km away.

During the early development phase, prior to longhole stopes being available for backfilling, waste rock will be hauled via trucks (30 tonne capacity) to the rockbreaker/grizzly facility near the shaft on 500 Level. During this phase, the waste will be hoisted to surface and stored on the waste stockpile. If any special waste (waste with potential to contain uranium) is hoisted to surface, it will be stockpiled separately. In the current plan, there is no requirement to hoist special waste. Once there are empty stopes available for backfill, waste rock will be hauled to a remuck near the stope, mixed with a cement slurry, and dumped into the stope. Table 16-8 provides the annual hoisting requirements.

Table 16-8. Total Tonnes Hoisted per Year

Item	2029	2030	2031	2032	2033	2034	2035	2036	2037
Ore									
Marginal Ore	0	2,525	4,163	0	0	0	0	6,376	1,123
Silling	0	7,059	21,259	31,492	43,657	40,650	40,650	25,686	5,215
Total Ore	**0**	**9,584**	**25,422**	**31,492**	**43,657**	**40,650**	**40,650**	**32,062**	**6,338**
Production	0	3,599	45,329	104,766	240,315	224,134	173,786	186,180	49,499
Waste	51,972	141,297	95,300	15,905	0	0	0	0	0
Total Hoisted	**51,972**	**154,480**	**166,051**	**152,163**	**283,972**	**264,784**	**214,436**	**218,242**	**55,837**

The mine ventilation system will consist of two ventilation openings to surface. The production shaft will supply the fresh air and the ventilation shaft will exhaust the air. Fresh air will split on the 500 Level down the haulage shaft and across to the 500 Level Fresh Air Raise (FAR). Fresh air will travel down a series of FARs and will be pulled off on the levels and into the active working areas. Rigid ducting will be installed close to the face and will pull the exhaust air from the face back to the Return Air Raises (RAR), which are connected to the ventilation shaft on 500 Level. The ventilation shaft will also serve as a second egress from the mine. The mine ventilation system is reviewed in more detail in Section 16.5.7.

The main mine dewatering system will consist of a clean water pumping system, utilizing decanting sumps on the 500 Level to settle solids. The decanted water will be pumped to surface via piping in the ventilation shaft. A series of boreholes and sumps will stage the water to the 500 Level decanting station.

In the case of a major inflow of water, an emergency sump/pump station will be setup on the 582 Level. Water staged from the underground workings will be directed to the unconsolidated waste filled stopes above the 582 Level Sump. The water will be collected at the bottom of the stopes (using the fill to filter out some of the suspended material) and pumped directly to the main pump station on 500 Level, bypassing the decanting sumps. Bulkheads will be constructed in the ramp at strategic locations to reduce the risk of the inflows overwhelming the dewatering system. The mine dewatering system is reviewed in more detail in Section 16.5.8.

16.5.4 Development and Production Schedule

Development and Production Schedule

A Gantt chart showing the Gryphon mine summary production schedule was prepared to show the surface construction, shaft sinking and construction, development, and production phases of the Gryphon project (Figure 16-23). There will be approximately six years of pre-production period from the time the shaft construction starts in Q1 2026 until production begins in Q2 2031. The production period will be approximately seven years.

LOM production totals 1.26 Mt of mill feed at an average grade of 1.79% U_3O_8 containing 49.7 Mlbs of U_3O_8. These tonnes include marginal tonnes, which are below the cut-off grade, but have enough value to cover the cost of haulage to the mill and processing, and these tonnes need to be mined to access the reserve material. During the peak production years (2032 to 2035), there will not be any marginal tonnes hoisted, to avoid offsetting the higher value tonnes.

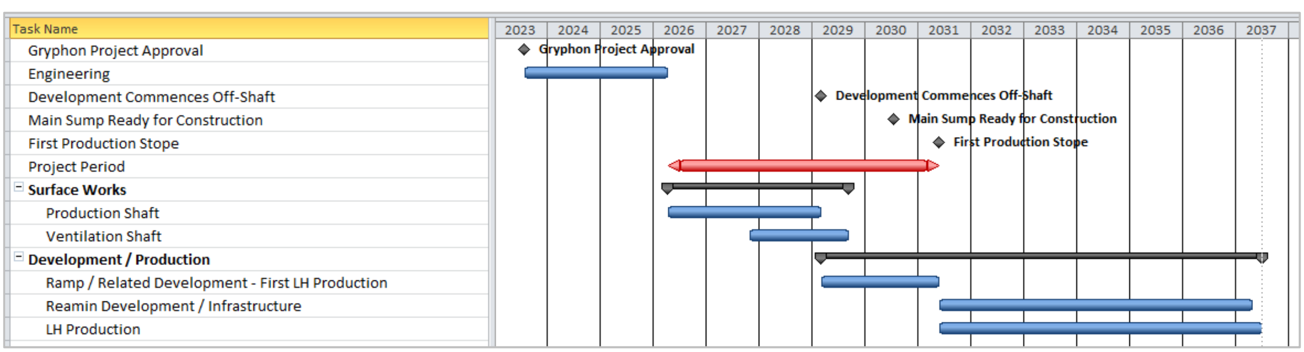


Figure 16-23. Gryphon Mine Summary Project Schedule

Underground Mine Development

Mine development will be completed using traditional drill and blast mining methods. Development jumbos will be equipped to drill 4.1 m long holes, resulting in a 3.8 m round advance. A schematic view of mine development was shown in Figure 16-21.

The development of the haulage ramp will be on the critical path for mine development. As development progresses to the production levels, additional headings will become available. Once development has advanced sufficiently on 500 Level and the haulage ramp to provide adequate space and secondary headings, a second mining crew will be introduced to concentrate on completion of the development on the 500 Level.

Development advance rates will vary depending on the location of the development and additional support/activities required. The early off-shaft development, within 50 m of the unconformity, will require probe holes and cover grouting in areas where the probe holes exhibit potential water inflows. Development in the ore sill will require 100% shotcrete to reduce the radiation exposure in these headings. Table 16-9 summarizes the single, double, and multiple heading advance rates for the various development headings.

Table 16-9. Underground Development Performances

Description	Heading Count	Units	Performance
Ramp Development Off-shaft (shotcreting and grout cover)	Single	m/day	3.0
5.0 m high x 4.5 m wide	Double	m/day	3.9
	Multiple	m/day	5.0
Ramp Development and Capital Access Development	Single	m/day	3.7
5.0 m high x 4.5 m wide	Double	m/day	4.8
	Multiple	m/day	6.1
Operating Access Development (wet shotcreting 15%)	Single	m/day	4.0
4.0 m high x 4.0 m wide	Double	m/day	5.2
	Multiple	m/day	6.6
Ore Sill Development (wet shotcreting 100%)	Single	m/day	2.8
4.0 m high x 5.9 m wide	Double	m/day	3.6
	Multiple	m/day	4.6

The access ramp and off-ramp lateral development will be driven 4.5 m wide by 5.0 m high. This drift size was selected in order to provide sufficient clearance for a 30-tonne truck. The drift height is required in order to provide adequate clearance for the installation of ventilation ducting during ramp development, and to provide sufficient ventilation airflow volumes for truck haulage operation during the mine production period.

A cross-section of the ramp is shown in Figure 16-24. Services to be installed in the ramp include a single 1.1 m (42 inch) ventilation duct (temporary installation during initial ramp development), water supply line, water discharge line, electrical power lines, blasting line, and a communications line. Installation and maintenance of a good quality roadbed will be essential for efficient truck haulage in the ramp. An allowance of 300 mm thickness for roadbed installation is included in the ramp design.

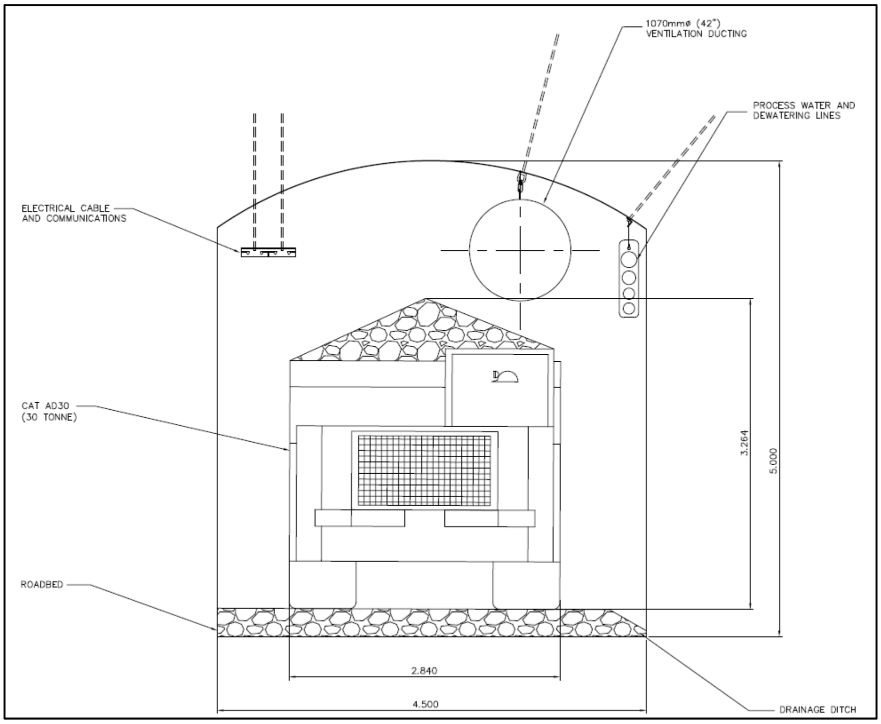


Figure 16-24. Ramp Profile

The silling and operating development will be driven at 4.0 m high, with a width ranging from 4.0 m to 8.0 m, depending on the width of the resource. Ground support will vary depending on mining widths but will include 2.4 m resin grouted rebar, screen, and 100% shotcrete coverage. These drifts will require installation of rigid ventilation duct and fans to pull exhaust air from the face, water supply lines, electrical power and communications cables, blasting lines, and backfill lines. All drifts will be driven at grades of 3% or greater to provide positive drainage for water inflows.

Ventilation raises will be developed using raisebore machines. The upper raises will be 4.0 m diameter while the lower raises will be 3.0 m diameter.

The fresh air raises will be equipped with a ladderway for secondary egress. The raises will be driven in multiple legs, with transfer drifts on the silling horizons. The fresh air raises may also serve as a permanent route for services throughout the mine.

Development of 4.0 m diameter raises will be scheduled at an overall advance rate of 2.4 m/day and 3.0 m/day for the 3.0 m diameter raises. All ventilation raises will be developed using a qualified mine contractor.

Mine Development Sequence

The overall mine development sequence will include:

- Development of the 500 Level access to the internal ventilation raises and the excavation of the infrastructure on the 500 Level. In parallel, development to the first production block on the 582 Level.
- Development of the access drifts on 582 and 567 Levels, preparation of stopes for early production, establishment of the ballast station, and provision of an early location for waste deposition.

- Continue ramp development to the next mining block on the 635 Level, complete the related lateral development, and excavate the ventilation raises to the 500 Level transfer drifts. This will allow production from Upper SW Block to start. Concurrently, the remaining infrastructure development on 500 Level will be completed.
- Development in the haulage ramp will continue to 695 Level, along with the ventilation transfer drifts on 635L and 665L, and the level access drifts to commence production from the Upper Main Block on 695 Level. The ventilation raises will be extended to the 635 Level prior to production commencing.
- This process will be repeated to the 770 Level. After extending the ventilation raises up to the 695 Level, production from the 770 Level can begin in the Lower Main and Lower D blocks.
- The development crews will then focus on sustaining capital and operating waste and ore silling, with two crews for the next two years, and then reduce to one crew for the remainder of the mine life.

The estimated LOM lateral development requirements are summarized in Table 16-10.

Table 16-10. LOM Lateral Development Requirements

Item	Quantity (m)
Capital	**9,658**
Ramp	3,576
Access/Infrastructure	6,082
Operating	**407**
Access	407
Silling	**9,544**
Ore	3,829
Marginal	1,184
Waste	4,531
Total Lateral	**19,609**

Table 16-11 shows the estimated LOM vertical development requirements planned for Gryphon.

Table 16-11. LOM Vertical Development Requirements

Item	Quantity (m)
Vertical Development	
Production Shaft (5.0 m dia. finished)	550
Ventilation Shaft (4.5 m dia. finished)	500
Internal Ventilation Raise (4.0 m dia.)	317
Internal Ventilation Raise (3.0 m dia.)	285
Total Vertical	**1,652**

Waste Rock Broken and Backfill Requirements

Table 16-12 shows estimated LOM quantities of development waste rock broken and rock required for backfilling.

Table 16-12. Waste Rock Broken, Backfill, and Ballast Quantities

Item	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Waste Material Produced										
Marginal Ore	0	0	0	0	7,763	11,749	10,692	11,161	0	0
Special Waste	0	0	11,156	18,668	27,714	47,903	48,612	40,577	23,697	2,893
Other Waste	0	57,579	153,896	136,394	83,344	62,899	55,390	22,700	0	0
Consumed Underground										
Backfill	0	0	7,016	43,826	89,676	107,994	101,365	65,099	19,366	2,197
Ballast	0	5,606	16,739	15,936	13,240	14,558	13,330	9,338	4,331	696
Hoisted (waste only)	**0**	**51,972**	**141,297**	**95,300**	**15,905**	**0**	**0**	**0**	**0**	**0**

There will be a total of 320,000 tonnes of waste hoisted to surface in the first 4 years of development. This waste will be stockpiled on surface. There are no plans to use this waste for construction as it is potentially acid generating. Table 16-13 indicates that there is sufficient waste material produced in years 1 through 4 to keep up with Gryphon backfill requirements without the need for the hydraulic backfill plant, and also shows the amount of backfill required by year and the ratio of waste rock to hydraulic fill.

Table 16-13. Backfill Required and Ratio of Waste Rock to Hydraulic Fill

Backfill (tonnes)	2028	2029	2030	2031	2032	2033	2034	2035	2036
Rock Fill	0	0	7,016	43,826	89,676	107,994	101,365	65,099	19,366
Hydraulic Fill	0	0	0	0	0	78,890	72,891	76,025	120,062
Total	**0**	**0**	**7,016**	**43,826**	**89,676**	**186,884**	**174,256**	**141,125**	**139,428**
Rock Fill (%)	0	0	100%	100%	100%	58%	58%	46%	14%
Hydraulic Fill (%)	0	0	0%	0%	0%	42%	42%	54%	86%

Underground Mine Production

Production from the Gryphon resource will target the milling capacity of 9.0 Mlbs U_3O_8 per year, equivalent to 605 tonnes per day of mineralization. The deposit has the potential to exceed 605 tonnes per day (or 0.9 Mlbs).

Mine production rates have been selected considering a custom milling scenario for the Wheeler River project. Steady annual uranium production rates should help in negotiating custom milling terms, and relatively high rates of uranium production may be more difficult to accommodate.

Tonnes/pounds produced in excess of the mill targets will be stockpiled at the mill.

Stope mucking will be completed using LHDs. The LHDs will transfer material to a remuck located near the haulage ramp and subsequently load haul trucks. The LHDs will be equipped to allow remote operation in order provide safe working conditions near stope brows and to facilitate final clean-up of the longhole stope floor prior to backfilling.

Production grade control will be active throughout the stope development, drilling, and production periods. It is expected that during initial over-cut and under-cut development, drill hole sampling, chip sampling, and drift mapping will be completed to provide detailed information as to the development ore grade variations along the drifts. Production drill holes will have cutting samples taken by the driller to better determine the production grade estimates for the stope.

The final production grade control will consist of drawpoint sampling during the production mucking. This will confirm predicted production grades from the stopes.

A mine production plan has been generated for the Gryphon underground mine, as shown in Table 16-14. This production plan shows the start of production in 2030. The production rate will ramp up to a steady-state production rate of 9.0 Mlbs U_3O_8 per year starting in 2033. All underground mine production will be completed in 2037.

Table 16-14. Mine Production Plan

Mining Block	Elevation	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
E Zone	582 EZ	0	0	3,599	22,344	0	0	0	0	0	0
Upper SW	590 SW	0	0	0	0	368	0	0	0	0	0
	605 SW	0	0	0	0	7,250	0	0	0	0	0
	620 SW	0	0	0	1,050	9,754	0	0	0	0	0
	635 SW	0	0	0	9,781	0	0	0	0	0	0
Upper Main	590 UM	0	0	0	0	0	0	0	0	4,188	0
	605 UM	0	0	0	0	0	0	0	2,485	5,374	0
	620 UM	0	0	0	0	0	0	0	16,195	0	0
	635 UM	0	0	0	0	0	0	19,380	16,583	0	0
	650 UM	0	0	0	0	0	4,209	51,413	0	0	0
	665 UM	0	0	0	0	0	50,540	0	0	0	0
	680 UM	0	0	0	0	23,567	40,684	0	0	0	0
	695 UM	0	0	0	12,153	40,211	0	0	0	0	0
Lower Main	680 LM	0	0	0	0	0	0	0	0	0	3,602
	695 LM	0	0	0	0	0	0	0	0	2,908	7,010
	710 LM	0	0	0	0	0	0	0	0	69,806	0
	725 LM	0	0	0	0	0	0	0	45,273	33,049	0
	740 LM	0	0	0	0	0	0	30,245	72,863	0	0
	755 LM	0	0	0	0	0	35,263	62,015	0	0	0
	770 LM	0	0	0	0	18,951	59,600	0	0	0	0
	785 LM	0	0	0	0	0	0	0	0	0	34,016
	800 LM	0	0	0	0	0	0	0	0	15,346	4,212
	815 LM	0	0	0	0	0	0	0	0	8,595	0

Mining Block	Elevation	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Lower D	635 LD	0	0	0	0	0	0	0	0	658	658
	650 LD	0	0	0	0	0	0	0	0	7,467	0
	665 LD	0	0	0	0	0	0	0	0	10,175	0
	680 LD	0	0	0	0	0	0	0	0	6,437	0
	695 LD	0	0	0	0	0	0	0	1,686	22,177	0
	710 LD	0	0	0	0	0	0	0	18,700	0	0
	725 LD	0	0	0	0	0	0	49,116	0	0	0
	740 LD	0	0	0	0	0	36,696	11,966	0	0	0
	755 LD	0	0	0	0	599	13,322	0	0	0	0
	770 LD	0	0	0	0	4,065	0	0	0	0	0
Total Longhole Stoping		**0**	**0**	**3,599**	**45,329**	**104,766**	**240,315**	**224,134**	**173,786**	**186,180**	**49,499**
Silling		0	0	7,059	21,259	31,492	43,657	40,650	40,650	25,686	5,215
Total Production (tonnes)		0	0	10,658	66,588	136,258	283,972	264,784	214,436	211,866	54,714
Total Production (pounds)		0	0	432,132	2,741,670	5,850,109	12,038,266	10,927,836	8,381,563	7,573,549	1,767,095
Total Milled (pounds)		0	0	432,132	2,741,670	5,850,109	9,000,000	9,000,000	9,000,000	9,000,000	4,688,310

Longhole Mining

A stope optimizer program, Mineable Shape Optimizer (MSO), was used to generate and evaluate stopes at 15, 20, and 25 m level spacing. The level spacing at 15 m provided the most favourable results for dilution and recovery. The 15 m spacing was selected based on the MSO results and due to the narrow vein nature of the deposit, the blast hole deviation in the down dip dimension, and the variable shapes of the mining wireframes as viewed in a vertical projection.

Longhole drilling will be completed using ITH drills suited to narrow vein mining. Blast holes will be 100 mm (4 inch) in diameter and approximately 16 m in length. Blastholes will be loaded and blasted using cartridge emulsions. A dedicated loading crew will be utilized for stope loading and blasting.

A drop raise will be drilled and blasted to create an initial void for production blasting. The drop raise will consist of 14 holes in total, with 5 reamed to a 200 mm diameter. Slot rings surrounding the raise will be blasted across the full width of the longhole stope. The drill pattern will consist of a 2.0 m ring spacing and 2.4 m hole burden.

Stope dimensions will be 15 m vertical height, up to approximately 20 m along strike, and with widths varying depending on the local lens thickness. The average stope will be approximately 3,400 tonnes. Although ore may be produced at a rate of approximately 700 tonnes per day during the mucking cycle, the other stope activities will add significantly to the complete stope cycle time. Excluding the completion of over-cut and under-cut access development, a stope cycle time of 52 days has been calculated, as shown in Table 16-15.

Table 16-15. Summary of Stope Cycle Time

Stope production drilling (including slot)	8 days
Stope blasting/production	8 days
Preparation for backfilling	2 days
Hydraulic backfilling	6 days
Hydraulic backfill cure	28 days
Total	**52 days**

This equates to approximately 67 tonnes per day over the full stope cycle, therefore a minimum of 8 stopes will need to be available at any given time (active drilling, blasting, mucking, filling, or fill curing) in order to provide a steady-state production rate of 500 tonnes per day from stopes. The three main mining blocks will need to be in production concurrently to provide the required number of production stopes.

Over-cut (drill) drift and under-cut (mucking) drift will be driven for the production stope. Each drift may serve several stopes, starting at the "east or west" abutment and retreating toward the central access drift. The resource consists of several stacked lenses, and therefore each level may have several over-cut drifts. Mining generally will progress from footwall to hangingwall and from the extremities to the central access drift. Figure 16-25 illustrates a typical production level and the anticipated longhole stope outlines.

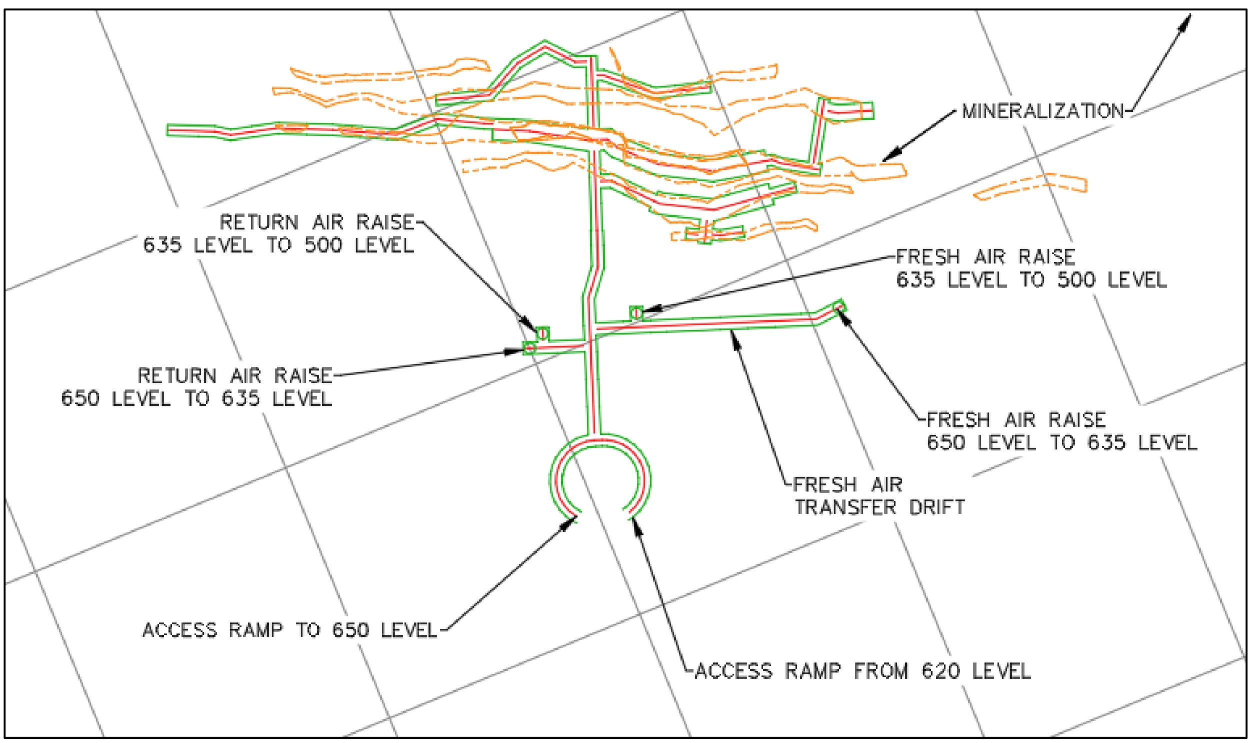


Figure 16-25. Typical Production Level

16.5.5 Underground Mine Backfill

Stope backfilling will be completed using a combination of rockfill, cemented rockfill (CRF), or hydraulic fill delivered by pipeline to the over-cut drift, with waste co-deposited by LHD.

The hydraulic fill plant will be constructed on surface and will produce fill from the surface overburden material (sand). The in situ sand material will be too coarse to produce a suitable backfill and will require additional processing to produce a material with additional fines.

The fill will be directed down 100 mm schedule 80 pipe installed in the ventilation shaft to the 500 Level, and then through a series of boreholes to active mine production levels below. Lateral pipelines will be installed on the levels to reduce line pressures in the distribution system and to deliver the backfill to the individual production stopes as required.

An underground fill crew will be designated to complete piping installations, construct fill barricades, and monitor the filling process as required. It is expected that these crews will be able to use existing mine service vehicles (primarily scissor lifts and Hiab trucks) to complete this work.

Backfill activities will be completed on all shifts, with fill pours running through shift change if required. All filling activities will be monitored at the backfill plant using remote cameras and pipeline pressure and flow monitors.

Work required to complete filling of a stope (hydraulic fill) will include:

- Installation of a pipeline from the junction on the level to the stope. Construction of a dump wall near the open stope, in the over-cut, when waste rock is to be dumped into the filled stopes.
- Construction of a backfill bulkhead near the stope at the stope under-cut drift.
- Filling of the first ±8 meters of stope height and allowing the fill to cure for 4 days to form a solid plug at the bottom of the stope.
- Fill the remainder of the stope to the level of the floor of the over-cut drift. When waste is available, it will be co-deposited into the backfill from the over-cut via LHD. Allow 28 days for the fill to cure before blasting a stope beside or below a filled stope.

Backfill Test Work

Backfill test work was completed as part of the PFS to determine the applicability of hydraulic backfill using material sourced from overburden at the Wheeler River site. The test work included material characterization, permeability, rheology, and strength testing. Based on initial observations, the overburden sample was screened of its fraction of particles above 1.0 millimeter and blended with an external source of material to increase the amount of fines sized below 500 micrometers. Strength test cylinders were cast with the screened and blended overburden over a range of binder content ratios and UCS tests were completed following 14-day and 28-day cure cycles. It was determined that the hydraulic backfill with a binder portion of 11% by mass achieved the targeted strength of 800 kilopascals within the required 28-day cure cycle time. Though the hydraulic backfill test work was successful at achieving the target strength requirement, future field work and test work is required to locate a suitable material on site that does not require blending to optimize the backfill recipe.

During the hydraulic backfill testing, test work on the applicability of cemented rock backfill and tailings paste backfill were also completed in case these options would be explored during future phases of the project. Though no fatal flaws were identified, the test work was preliminary and additional testing is required to better understand optimal operating parameters for these backfill options.

16.5.6 Ore and Waste Handling

During the initial development phase, 6-yard LHDs will load waste rock (and limited ore) into 30 tonne capacity haul trucks to be trucked to the 500 Level rockbreaker station. Ore and waste remucks will be located near the rockbreaker station to enable batching of the material as required.

Once sufficient open stopes are available, the bulk of the waste will be hauled to remucks near the open stopes, trammed by LHDs, and dumped into the nearest open stope. The remaining waste rock will be trucked to the 500 Level rockbreaker station. By 2033, it is estimated that all waste rock generated underground will be used for backfill and will not be hoisted to surface.

Ore produced from development or production will be trammed to a remuck on the level and subsequently loaded into trucks at a truck loading area near the ramp for hauling to the 500 Level rockbreaker station.

There will be three 30 tonne capacity trucks in the fleet to meet ore and waste rock handling requirements. One LHD will be dedicated to loading ore and waste rock into the haulage trucks, and to re-handling the waste as backfill.

The 30 tonne haulage trucks will transport ore to the 500 Level rockbreaker station located in the proximity of the production shaft station. Each truck will carry an average load of 19.5 tonnes per trip. Truck cycle times (round trip) have been calculated for each mining block, as shown in Table 16-16.

Table 16-16. Truck Cycle Times (Round Trip) by Mining Block

From E Zone	20 mins
From Upper Main and Upper SW	33 mins
From Lower Main and Lower D	43 mins
From Lower D FW	45 mins

A total of four trucks will be purchased to have up to three trucks available for haulage as the centre of mine production moves deeper over time. Keeping the availability of these ore haulage trucks as high as possible will be a priority for maintenance crews.

Roadbed maintenance in the haulage ramp will be critical to establishing an efficient truck haulage system. The haulage trucks will have priority for travel in the haulage ramp. A dedicated communication channel will be set up for all vehicle traffic in the ramp and haulage trucks will regularly report position and direction of travel by radio in order to notify other ramp traffic. A grader will be used to complete regular roadbed maintenance in the haulage ramp. This grader will typically operate on night shift when ramp traffic is reduced.

16.5.7 Underground Mine Ventilation and Heating

Design Criteria

The design of the ventilation system components and structures is based on the criteria outlined in Table 16-17.

Table 16-17. Ventilation Design Criteria

Dimensions	Units	Value
Production shaft diameter	m (ft)	5.0 (16.4)
Exhaust shaft diameter	m (ft)	4.5 (14.8)
Internal raise diameter	m (ft)	3.0 (9.8) or 4.0 (13.1)
Ramp/lateral development drift size	m x m (ft x ft)	5.0 x 4.5 (16.4 x 14.8)
Production shaft duct for ramp development; steel	mm (in)	1,829 (60)
Ramp development duct; PVC duct	mm (in)	1,220 (48)
Ore sill development duct; PVC duct	mm (in)	1,067 (42)
Overlap ventilation duct; fabric	mm (in)	915 (36)
Friction Factors / Resistances	**Units**	**Value**
Average blasted drift	kg/m^3 ($lb\text{-}min^2/ft^4 \times 10^{10}$)	0.0120 (65)
Concrete lined shaft with rope guides and services	kg/m^3 ($lb\text{-}min^2/ft^4 \times 10^{10}$)	0.0065 (35)
Raise bore raises	kg/m^3 ($lb\text{-}min^2/ft^4 \times 10^{10}$)	0.0050 (27)
Internal raises with manways	kg/m^3 ($lb\text{-}min^2/ft^4 \times 10^{10}$)	0.0241 (130)
Steel duct	kg/m^3 ($lb\text{-}min^2/ft^4 \times 10^{10}$)	0.0033 (18)
Fabric duct	kg/m^3 ($lb\text{-}min^2/ft^4 \times 10^{10}$)	0.0037 (20)
PVC duct	kg/m^3 ($lb\text{-}min^2/ft^4 \times 10^{10}$)	0.0018 (10)
Single door	PU	20
Bulkhead	PU	250
Airlock	PU	40
Velocity Thresholds	**Units**	**Value**
Production shaft	m/s (fpm)	>12.0 (2,360)
Exhaust shaft	m/s (fpm)	>24.0 (4,724)
		8.0 - 12 (1,575 - 2,360)[1]
Airways with haulage	m/s (fpm)	>6.0 (1,180)

Notes:

1. Velocity at which water blanketing occurs in exhaust shafts.

Description

The mine will be designed to have dedicated fresh and return ventilation raises/drifts, providing flow through ventilation in all production mining areas to avoid recirculation of air. The ventilation system will be designed to operate mainly as a pull system, to reduce leakage throughout the mine, with a push fan at the intake directing the air through the heater and into the production shaft. The fresh air will be split on the 500 Level to the haulage ramp, and to the 500 Infrastructure (garage and main sump) and internal fresh air raise, from which it will be directed to the working areas through a series of internal fresh air raises and transfer drifts. Regulators, dampers, doors, bulkheads, and auxiliary fans will be included to control the air distribution to the various levels as necessary. The air will exhaust through a series of internal return air raises, return air transfer drifts, and the ventilation shaft to surface.

Airflow Requirements

The airflow requirement in Saskatchewan for diesel powered mobile equipment operating in underground mines is 100 cfm/HP (0.063m^3/s). The total airflow requirement is summarized in Table 16-18, taking into consideration the equipment operating underground at full production and airflow required in active levels/facilities without equipment.

Table 16-18. Total Airflow Requirements

	Qty	Engine Rating (kW)	Utilization	Total Power (kW)	Airflow Requirement	
					m^3/s	cfm
Development						
2 boom Jumbo (elec/hyd)	2	110	20%	44	3	5,871
Boom Bolter	3	62	20%	37	2	4,964
6 yd LHD	2	208	75%	312	20	41,631
Scissor lift	2	129	40%	103	7	13,770
Toyota Jeep	2	120	40%	96	6	12,810
Production and Backfill						
ITH with compressor	2	119	20%	48	3	6,351
6 yd LHD	2	208	75%	312	20	41,631
Tractor	1	125	30%	38	2	5,004
Haulage						
30t truck	3	305	100%	915	58	122,092
Miscellaneous						
Toyota Jeep	3	120	30%	108	7	14,411
Scissor lift	1	129	40%	52	3	6,885
Boom truck	1	129	30%	39	2	5,164
Shotcrete equipment	1	125	40%	50	3	6,672
3.5 yd with forks	1	150	30%	45	3	6,005
Kubota forklift	1	125	40%	50	3	6,672
Active levels allocation (10m^3/s)	6				60	126,522
Main sump (6m^3/s)	1				6	12,708
Facilities allocation (14m^3/s)	1				14	29,652
Leakage (10%)					14	29,993
TOTAL AIRFLOW					**236**	**498,808**

A description of the ventilation system as it evolves through the life of mine is given in the following sections.

Pre-production

The production shaft will be the first airway to be established in the ventilation system from which the ramp and drift to the exhaust raise will be developed. The return and fresh paths will be through the production shaft, with the fresh air isolated within an 1,800 mm (60 inch) diameter steel duct. A schematic of the ventilation system at this stage is provided in Figure 16-26. The surface intake fans for the LOM will push the air down the ducting in the shaft to the 500 Level, at which point a wye ducting arrangement will be made with a 75 kW (100 HP) fan at each end pushing air through a 1,220 mm (48 inch) PVC duct. Rigid ducting will be used to reduce the frictional losses, and, if the ramp continues further than planned, prior to establishing flow through ventilation to surface, a second fan could be installed in series to provide the additional push required.

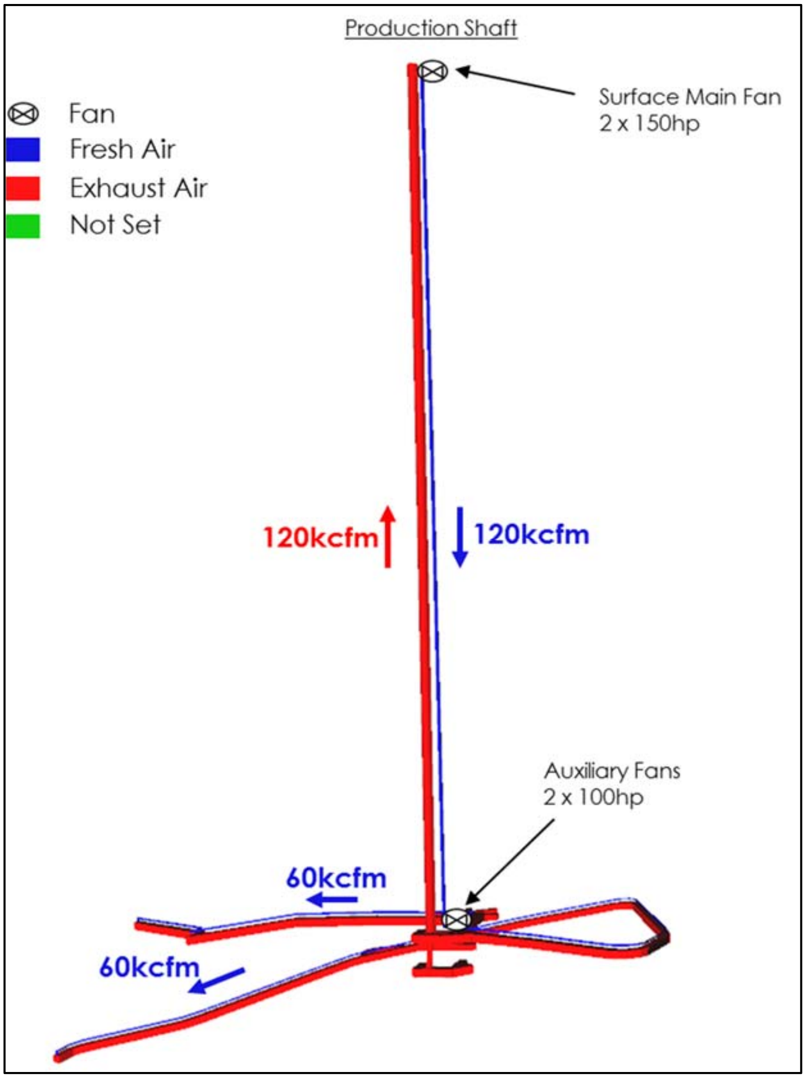


Figure 16-26. Pre-production Ventilation Schematic

Full Production

Once the ventilation shaft is commissioned, a flow through ventilation system can be implemented in a 'push-pull' configuration. The primary movers will be the 'pull' fans on the ventilation shaft, but the 'push' fans will be required at the intake to ensure that the air is drawn through the heaters rather than directly through the headframe. Airlock/regulators will then be used to control the ventilation flow from the fresh air raises, with an exhausting overlap auxiliary ventilation system drawing air onto the active levels. A schematic of the ventilation system at full production is presented in Figure 16-27. While this represents a typical arrangement of auxiliary fans, the exact locations will depend on production schedule. Bulkheads will be required at all the internal fresh and return air raises, and doors and regulators will be installed at the raises to provide fresh air onto the active levels. The internal fresh air raises will be equipped with an escapeway.

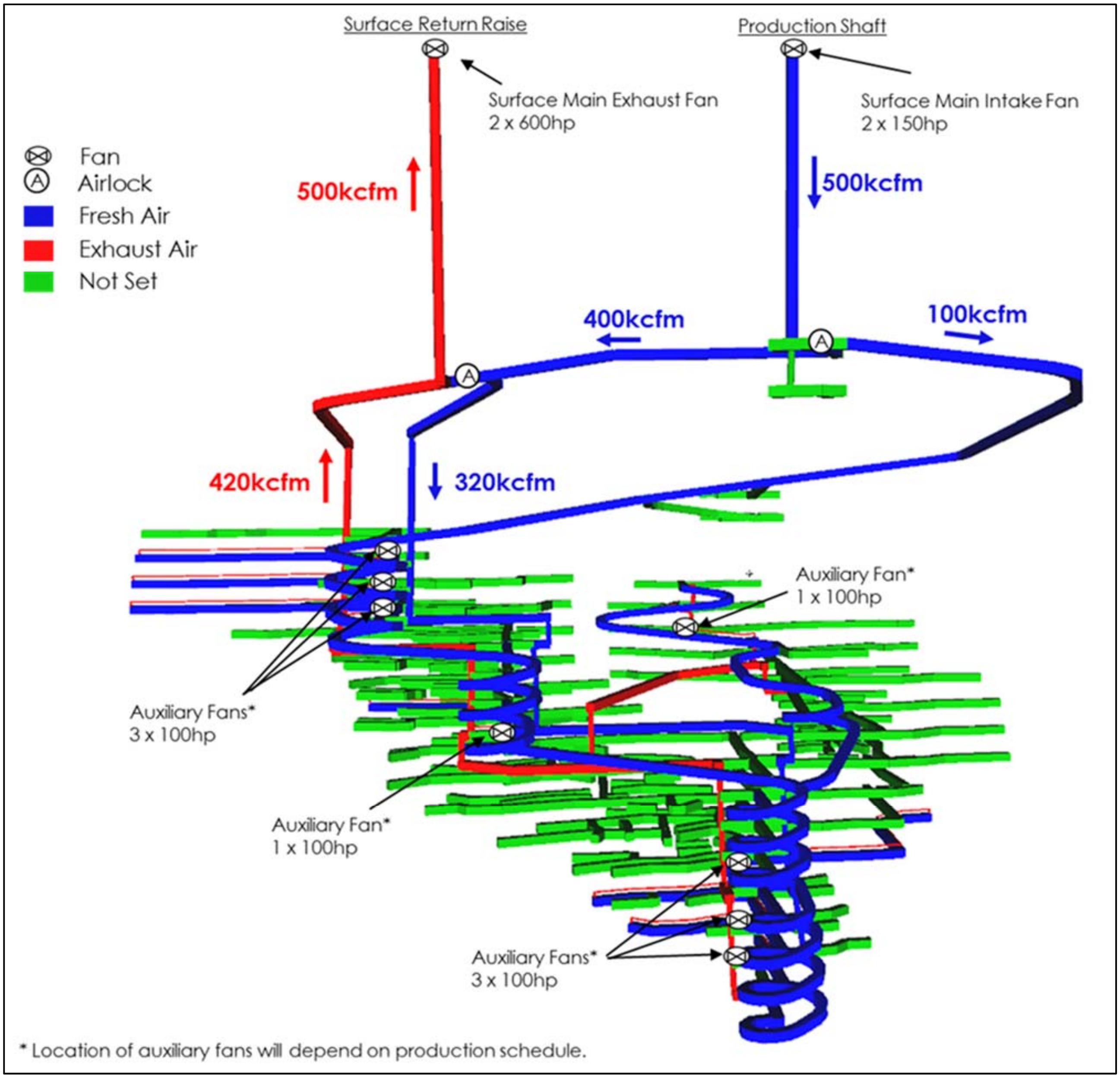


Figure 16-27. Full Production Ventilation Schematic

Main Fans

The main fan motor power requirements were estimated from the ventilation model, taking into consideration pressure losses at the fan from the ducting, silencer, damper, etc. The ventilation system capabilities and fan ratings are summarized in Table 16-19.

Table 16-19. Ventilation Fan Power Requirements (including ducting, heater, pressure losses, etc)

Location	No. Fans	VFD Capable?	Peak Airflow m^3/s (cfm)	Peak Pressure Pa (in. W.G.)	Total Power kW (HP)
Surface Intake Fan (Production Shaft)	2	Yes	260 (550,000)	600 (2.4)	2 x 112 (150)
Surface Return Fan (Ventilation Shaft)	2	Yes	240 (500,000)	3,360 (13.5)	2 x 450 (600)

All main fans will be located on surface and have a horizontal arrangement. The fresh air fan will be provided with a propane heating system for use during the winter months.

Winter design minimum ambient temperature was taken to be -40°C (-40°F), with the mine air heaters sized for a maximum temperature rise of 26.6°C (80°F), thus increasing the air temperature to 4°C (40°F) from the winter design ambient temperature. For the peak fresh air system capacities, a 14,936 kW (51 MMBTUhr) heating system will be required.

Auxiliary System

To reduce exposure time of exhaust air from the face in the production headings, an exhaust overlap system will be implemented. A schematic of this auxiliary ventilation method is illustrated in Figure 16-28. The system will consist of two fans, one providing the primary power to move the fresh air on the level and the smaller overlap fan required to ventilate the face and ensure that no ventilation dead zones are present.

The airflow requirements in active production levels will be 30 m^3/s (60,000 cfm), so the exhausting power can be delivered by a single 75 kW (100 HP) installed at the internal return air raise bulkhead. The 1,067 mm (42 inch) ducting for the exhausting fan will need to be rigid to sustain the negative pressures resulting from the exhausting fan, and PVC ducting was assumed in the design criteria to minimize the friction loss at the duct. For the overlap section of the ventilation system, a smaller fabric duct, 915 mm (36 inch) in diameter, is planned with a 38 kW (50 HP) fan directing the air onto the face. Fabric ducting will be required for this last section, so that it can be retracted prior to blasting and thus avoiding damage to the auxiliary ventilation infrastructure.

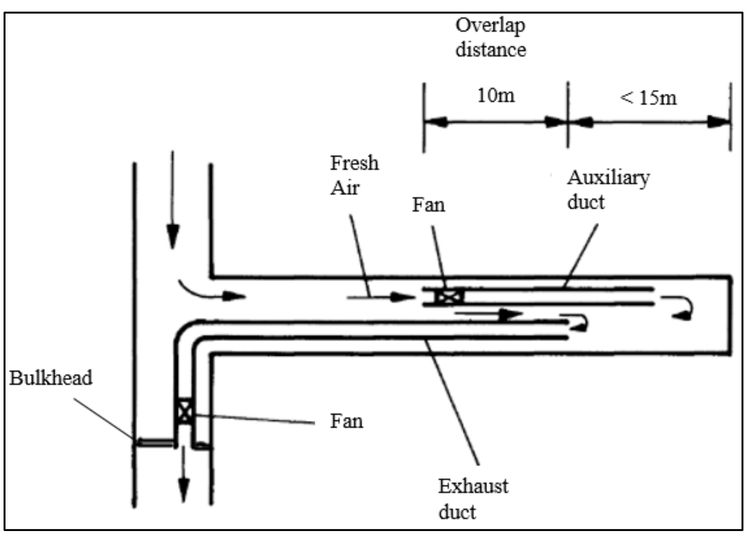


Figure 16-28. Schematic of Exhaust Overlap Auxiliary Ventilation System

16.5.8 Underground Mine Dewatering

The dewatering system will be designed to handle emergency inflows of 1,650 m^3/hr and daily mining activity inflows of 258 m^3/hr (58 m^3/hr from process water and 200 m^3/hr from groundwater). The system is designed to pump mine waters from underground up to surface for treatment and discharge. For normal operations, the dewatering system will consist of a main pump station at the 500 Level, a decanting station at the 500 Level, and cascading pump stations located at the 582 Level, 635 Level, 695 Level, 725 Level, 755 Level, and 815 Level. The main pump station and the 582 Level and 635 Level cascading sumps are designed to handle emergency inflows. In the event of an emergency inflow, which is assumed most likely to occur above the 635 Level, water will gravity drain to the 635 Level, be transferred to the top of a rock-filled stope at the 567 Level, referred to below as the emergency catchment sump, recollected at the 582 Level cascading station (also referred to as the booster station), and transferred to the main pump station on 500 Level. The purpose of the transfer to the rockfilled stope is to allow for settling and filtration of silt which could potentially enter with the groundwater. The dewatering system is not designed to prevent the mine from flooding below 635 Level, but to maintain accessibility and integrity of infrastructure above that level to facilitate remediation of the inflow source. It is assumed that, in such an event, all other mine operations will cease, and efforts will be dedicated to remediation of the inflow (Figure 16-29). A negative pressure exhaust system is also tied in to the dewatering design to exhaust radon gas from the water.

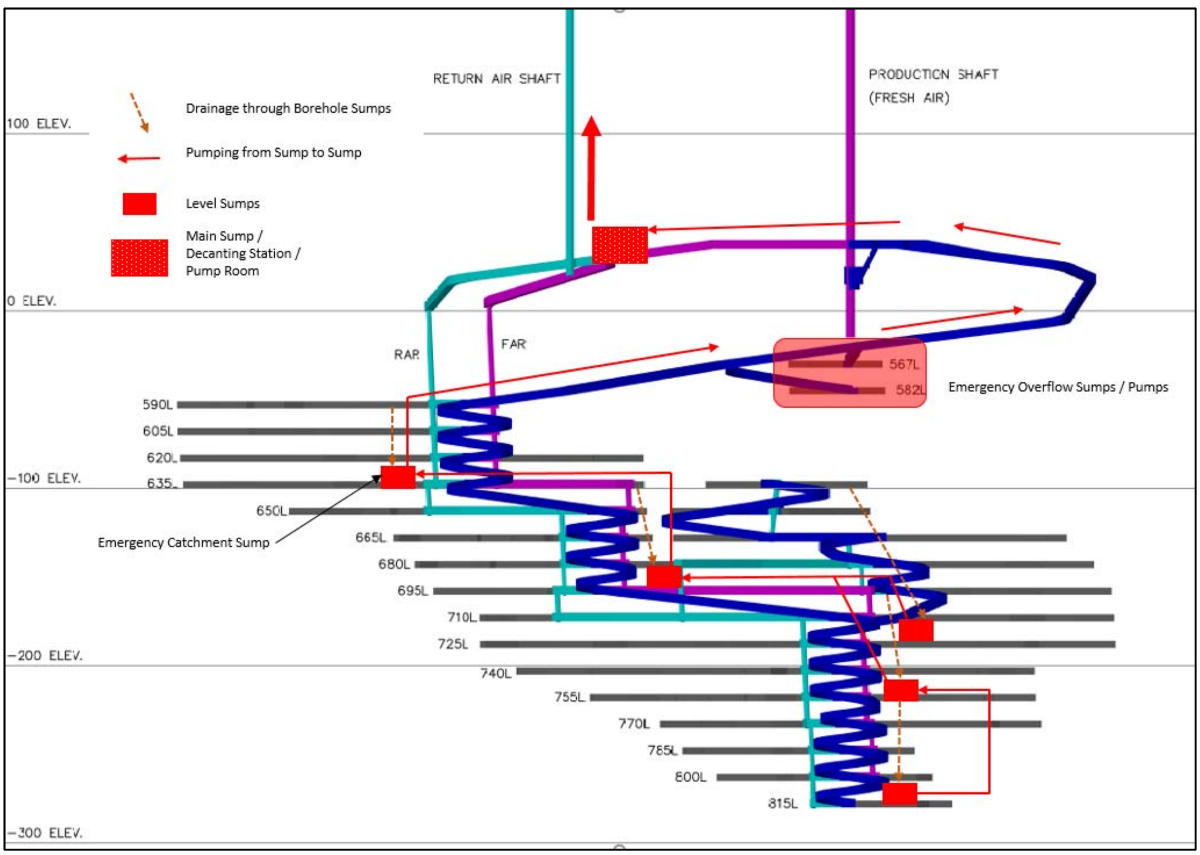


Figure 16-29. Dewatering Schematic

Main Pump Station

The main pump station will consist of three sets of two multi-stage centrifugal pumps in parallel. One pump in one of the sets will operate to handle the daily average inflows within the Gryphon orebody. Water from the pumps will be transferred to surface via three 300 mm pipes in the return air raise shaft or three 300 mm pipes in the production shaft. Only one set of three pipes is required at any time, but both shafts are equipped with dewatering pipes for redundancy. The main pump station will also contain a large storage sump that runs parallel to the pumps. To handle emergency inflows, 5 out of the total 6 pumps will run, with an additional pump on standby. The main pump station will be closed off with a rubber curtain and contain an exhaust pipe for radon gas, which will be exhausted to surface via the return air raise. Table 16-20 summarizes the discharge capacities of the proposed pumps.

Table 16-20. Total Output by Pump

Operation	Pumps Operating	Total Outflow (m^3/hr)
Daily	1/6	453
Emergency	5/6	1,817

Emergency Catchment Sump

The emergency catchment sump will be used in the event of emergency inflows. The stopes located within the 567 Level will serve as the sump. Water from the 635 Level cascading station will be pumped to the stopes at the 567 Level and rockfill within these stopes will allow silt to settle and filter through the water prior to entering the booster station at the 582 Level. From there, water will bypass the decanting station and enter the main pump station sump, from which it will be pumped to surface.

Decanting Station

The decanting station will consist of two decanting drifts and one collection sump. Water from the 582 Level will be pumped to one of the decanting drifts until it is completely full of solids. Water will then be pumped to the secondary decanting drift while the solids in the primary drift are dewatering or being cleaned out. Once the secondary drift is full, water will then be pumped back to the first decanting drift, thus repeating the process. The decanting drifts will consist of a permeable wall to filter solids out of the water. During normal operations, water will either permeate through the wall, or overflow the wall, while the majority of solids will remain in the decanting drift. Once the decanting drift is taken offline, water will permeate the wall as the solids dewater. Water from the decanting drifts will gravity drain to a collection sump. This collection sump will drain to the main pump station sump via boreholes, from which it will be pumped to surface. The decanting station will be closed off with a rubber curtain and contain an exhaust pipe for radon gas, which will be exhausted to surface via the return air raise.

Cascading Pump Stations

Cascading pump stations will be placed within the Gryphon orebody to pump dirty water to the decanting station. The cascading pump stations can be found at the 582 Level, 635 Level, 695 Level, 725 Level, 755 Level, and the 815 Level. The cascading pump stations at the 635 Level and 582 Level will be fitted with five submersible pumps, a sump, and a cat walk. Cascading pump stations on every other level will be fitted with one submersible pump and a sump. Each station will be closed off with a rubber curtain and contain an exhaust pipe for radon gas, which will be exhausted to surface via the return air raise.

Daily Inflows

To meet daily inflow requirements, the cascading pump station pumps and sumps will be designed to meet the daily average dewatering rates of 258 m^3/hr. During development, a moveable diaphragm pump at the development face will pump water to borehole sumps via the ramp. These borehole sumps will be located directly above the cascading pump stations. Water will then enter the cascading pump station, from which it will be pumped to the next pumping station above, via the ramp. Once water reaches the cascading pump station at the 582 Level, it will be pumped to the decanting station, where it will settle prior to entering the main pump station at the 500 Level.

Emergency Inflows

In the event of emergency inflow, the cascading pump stations below the 635 Level will flood. The cascading pump stations at the 635 Level and 582 Level will be adequately sized to meet these emergency inflows. From the 635 Level, water will be pumped to the stopes at the 567 Level, where silts will settle out of the water prior to entering the cascading pump station at the 582 Level. From the 582 Level, water will bypass the decanting station and enter the main pump station at the 500 Level, from which it will be pumped to surface. Bulkheads will be constructed in the ramp at strategic locations to reduce the risk of the inflows overwhelming the dewatering system. Once the bulkheads are cured, the pumps will shut off and the mine will flood to the bulkheads.

16.5.9 Other Underground Mine Infrastructure

Underground Maintenance Facilities

Underground mobile equipment will be lowered down the production shaft to 500 Level. No ramp access to surface is planned. A fully serviced, multi-bay underground maintenance shop will be constructed on the 500 Level, within walking distance of the production shaft. It will be sized to accommodate ~10% of the total equipment fleet. The maintenance shop will consist of 3 bays, each with an overhead crane. The bays will be sized to fit the largest pieces of underground equipment. The maintenance facility will also contain an electrical maintenance shop, a welding bay, distribution of lubricants, an office, and limited parts storage. A tire storage bay and wash bay will be located nearby. The shop will be designed for flow through traffic, when possible, and will be tied into an exhaust duct, which exhausts directly into the ventilation shaft. A 3-bay underground maintenance facility is shown in Figure 16-30.

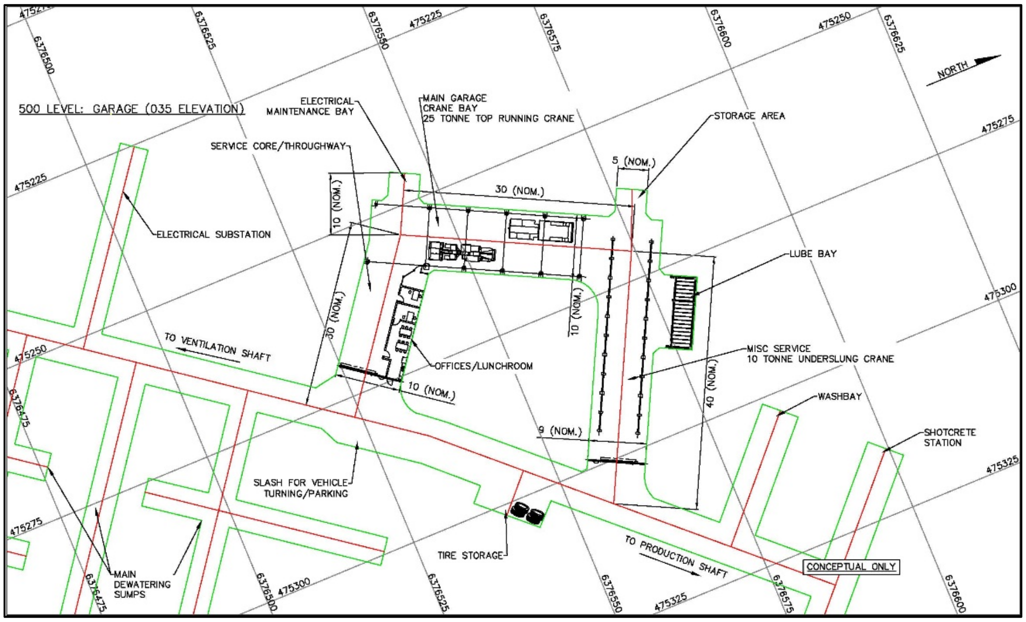


Figure 16-30. Underground Maintenance Facility

Fuel and Lubricant Storage

Fuels and lubricants will be transported underground in bladders via the cage and transported with a forklift to the fuel and lube stations. Fuel and lubricants will be stored at two fueling and lube stations, located on the 710 Level and near the main 500 Level shop. The fuel and lubricants, including hydraulic oils, will be stored in appropriate self-contained modules (Lube-cubes or SatStats). The fuel and lube storage bays will be equipped with a fire suppression system and fire-resistant doors, as per regulated requirements.

Explosives and Detonator Storage

Underground explosives storage will be established on the 500 Level. Underground storage capacity will be up to 10 days supply of development and stope blasting materials. A small storage facility for detonators will also be constructed on the 500 Level.

Second Egress

A system for providing a second egress from the mine is required to be in place prior to the start of production. This second egress will consist of a small cage installed in the ventilation shaft and ladderways installed in the internal fresh air raises.

Refuge Stations

A permanent refuge station will be located on the 500 Level near the maintenance shop and on the 700 Level central to the production areas. The refuge stations will serve the development and production crews working in the area, as well as the maintenance crews working in the underground shop.

Two portable refuge stations will be purchased. As mine development and production moves into new mining areas, an appropriate existing cut-out will be used to set up a temporary refuge station.

16.5.10 Underground Mine Services

Roadbed Material Delivery

Maintaining good roadbed conditions in all travel ways is important to overall productivity and mobile vehicle maintenance and will be essential in allowing efficient vehicle travel on the ramp.

Roadbed material for the underground mine will be obtained by crushing waste rock underground on the 567 Level. As required, the haulage trucks will haul the roadbed material to an area where road maintenance is currently underway.

A grader is included in the mobile equipment fleet. As it is anticipated that most material movement and personnel traffic in the ramp will take place during day shift, the grader will operate on night shift to maintain good operating condition on the access ramp travel surface. This will include the placement and spreading of additional roadbed ballast as required.

Materials Handling

All materials brought into the mine, including explosives, shotcrete, pipe, ground support materials, ventilation duct, fans, etc, will be transported underground via the shaft to the 500 Level storage bays and subsequently moved to the various storage locations throughout the mine. This will be accomplished primarily using flat-deck trucks equipped with Hiab lift booms. It will be important that sufficient materials storage is maintained underground for materials transport to be done in an efficient manner, with each load of materials being maximized in order to reduce the total number of cage trips required.

Underground Electrical Power Distribution

Four main 25 kV feeders will be supplied from the surface pre-fabricated electrical building, or “e-house”, to provide the surface and underground power for the Gryphon deposit. These will include two surface feeders and two redundant underground feeders, ‘A’ and ‘B’.

The ventilation and hoisting systems will be supplied via a buried 25 kV Teck cable routed from the surface e-house to the hoist/FAR e-house. At the hoist/FAR e-house, two transformers will be installed. A 25 kV to 4,160 V step-down transformer, sized to supply power to the RAR fans and hoist motor (via VFDs), and a second transformer, 4,160 V to 600 V for auxiliary loads and FAR fans. From the hoist/FAR e-house, three buried cables, including two 5 kV feeders and one 1 kV feeder, will be run to the RAR e-house.

The total surface ventilation power requirement will be approximately 1,500 kVA, based on the operation of two 600 HP return air and two 150 HP fresh air surface ventilation fans.

The total hoisting power requirement will be approximately 3,500 kVA, based on the operation of one 2,000 HP hoist and a 300 HP egress hoisting system.

In the event of an emergency, power will be required for ventilation, dewatering, egress hoisting, and auxiliary surface power. Emergency power will be provided by three 13.8 kV, 3.5 MW gensets connected in parallel. These gensets will provide power to a 13.8 kV switchgear, which will be connected to a 13.8 kV / 25 kV step-up transformer. The generator system will be connected to the main system via a transfer switch.

The total surface emergency power requirement will be approximately 9,000 kVA, based on the operation of one 600 HP return air fan and one 150 HP fresh air surface ventilation fan, five 1500 HP dewatering pumps, two 100 HP egress hoists, and surface auxiliary loads. The emergency related electrical power loads are summarized in Table 16-21.

Table 16-21. Emergency Related Electrical Power Load Summary

Item	Connected Load (kVA)
Generator Switchgear	8,974
Gryphon Aux. CDP	1,000
Surface Fresh Air Fans	142
Surface Return Air Fans	566
Surface Hoisting	188
Mine Dewatering Switchroom	7,078

The two underground feeders, 'A' and 'B', will run from the surface e-house down the shaft to a switchgear in the main shaft substation at 500 Level. This switchgear will provide a tie-in point between underground feeders ‘A’ and ‘B’ and will also provide an isolation point for the main dewatering switch room, 500 Level infrastructure, and the development in the main ramp. The tie-in point in the switchgear will provide redundancy for the critical main dewatering pumps and for the development phase to minimize down time. As initial development occurs, isolation will be done from the main shaft substation switchgear until the underground switchrooms are installed. There will be 25 kV junction boxes installed at every two to three mining levels as the ramp is advanced to maintain the integrity of the main 25 kV distribution cables, reduce splices, and facilitate tie-ins for the sill development and production crew as they approach the area.

The 710L switch room will be installed once the ramp accesses this level and will serve as the primary isolation and tie-in point for the 710 Level infrastructure and the production/development in the Lower Main and Lower D.

In addition to the above-mentioned switchrooms, there will be three additional types of switchrooms; a development switchroom (DSR), a primary switchroom (PSR), and a secondary switchroom (SSR).

DSRs will be supplied from 25 kV power and will include two 25 kV junction boxes and a mine power centre (MPC) for 600 V power. These facilities will provide power to development crews for equipment, ventilation, and pumping. The high voltage junction boxes will be permanently installed and the MPC will have the ability to be relocated as required.

PSRs will be supplied from 25 kV power and will include a 25 kV disconnect switch, a 25 kV junction box, and an MPC for 600 V power distribution. These facilities will be placed in new excavations with room for an optional additional MPC in areas of high power demand (if needed).

PSRs will feed 600 V power to SSRs on other levels via boreholes (or manways) and will have capacity to provide power to level infrastructure facilities (i.e. dewatering pump sumps, refuge stations, storage bays) as required.

It is assumed that no PSR will need to provide development and production power to more than four SSRs at a time. This is based on a mining plan to have two development crews and one production crew located in each area.

SSRs will typically include a 600 V rack with associated equipment and will be in a small cut out excavation. SSRs provide power to production and development crews and local infrastructure facilities.

The distribution system offers flexibility and ensures that MPCs can be placed wherever infrastructure loads are deemed required, such as ventilation or dewatering.

The total underground power requirement will be approximately 15,000 kVA, based on one production crew, two development crews, and supporting infrastructure, including pumps and fans.

The surface and underground related electrical power loads are summarized in Table 16-22.

Table 16-22. Surface and Underground Electrical Connected Loads

Item	Connected Load (kVA)
Surface E-House Switchgear	**20,822**
Surface Fresh Air Fans	**1,000**
Fan No. 1	382
Fan No. 2	142
Auxiliary Loads	99
Surface Return Air Fans	**1,232**
Fan No. 1	566
Fan No. 2	566
Auxiliary Loads	99
Surface Hoisting	**3,661**
Hoist	1,887
Hoist Auxiliary Loads	698
Headframe Auxiliary Loads	698
Egress Loads	378
Mine Dewatering Switchroom	**7,078**
Feeder A	**3,767**
U/G Auxiliary Fans	944
Compressors	755
Cascading Pumps/Booster Pumps	783
Development/Production Crew	900
Misc. (lighting, etc)	195
U/G Infrastructure (shop, refuge station, etc)	190
Feeder B	**3,702**
U/G Fans	783
Compressors	1,132
Misc. (lighting, etc)	210
Cascading Pumps	878
U/G Infrastructure (shop, refuge station, etc)	699

Underground Communication, Automation, and Instrumentation

The primary underground communications will be via a wireless VOIP telephone (over fibre network) system. A fibre optic network, including programmable logic controller remote input/output, has been included in the design for monitoring and control of ventilation and dewatering systems. The fibre optic backbone has been suitably sized to accommodate expansion and additional systems that may be required.

Instrumentation and automation initiatives will be taken where suitably advanced technology can be utilized. Applicable systems will function via the installed fibre optic network. The technology will apply to the ventilation network, mine dewatering system, and additional systems that may evolve over the life of the property.

Mine Process Water Distribution

Water will be distributed throughout the mine via Victaulic water lines (50 mm diameter) installed in the access ramp and lateral development headings. Pressure reducers will be installed in the water supply line in order to maintain water pressures at a maximum of 100 psi working pressure.

Definition Drilling

Detailed delineation diamond drilling will be completed from underground drill bays situated over a strike length of approximately 300 m in order to evaluate the continuity of the deposit lenses. Underground drilling, when combined with existing surface exploration drill holes (spaced approximately on 25 m centres), is designed to achieve a spacing of approximately 12.5 m centres at the expected mineralized horizons.

Approximately 30,000 m of underground delineation drilling in 180 holes will be required to achieve the appropriate spacing across the deposit lenses. Each hole will average approximately 160 m in length.

16.5.11 Underground Mobile Equipment

Mobile mine equipment will be purchased in stages as mine development advances and the second development crew is added. The production truck haulage fleet will be increased as mine production rates increase and the average depth of production increases.

For the initial ramp development starting in year 1, a single development jumbo, an LHD, two 30 tonne trucks, a scissor lift, and a Hiab boom truck will be purchased.

During year 2, the second development crew will begin working. In year 3, the production crew will begin production on 582 Level. Additional development equipment and most of the mine production equipment will be purchased during this period. A total of 30 pieces of mobile equipment will be required underground during the peak production period. Table 16-23 provides a breakdown of the equipment fleet at peak requirement.

Table 16-23. Mobile Equipment List

Description	Peak Count
Development	
2 boom Jumbo (elec/hyd)	2
Boom Bolter	3
6 yd LHD	2
Scissor lift	2
Toyota Jeep	2
Production and Backfill	
ITH with compressor	2
6 yd LHD	2
Tractor	1

Description	Peak Count
Haulage/Construction and Misc.	
Toyota Jeep	3
30 t truck	3
Scissor lift	1
Grader	1
Boom truck	1
Shotcrete sprayer (Encreter - dry or wet applications)	1
Transmixer	1
Kubota forklift	2
3.5 yd with forks (load roadbed crusher/transport materials/compressors)	1
TOTAL UNDERGROUND MOBILE EQUIPMENT	**30**

16.6 Radiological Dose Assessments

A screening level evaluation was completed for both Phoenix ISR operations and Gryphon underground mining operations. The aim of the assessment was to provide input into the technical and operational designs for these facilities in order to ensure a safe and efficient operation.

The evaluation focused on key personnel in various roles that are likely to receive the highest exposures. For the purpose of current screening assessment, exposure to radon/radon decay products (RDP) and gamma radiation are the key considerations based on experience in the Basin at similar operations. While there will be contributions to dose from LLRD and from general time in the native mine rock exposure, these are expected to be minimal relative to that from RDP and gamma radiation and were therefore not included in the assessment. In future work a more detailed assessment of all workers from all sources of exposure will need to be completed.

16.6.1 Regulatory Setting

Maintaining radiation doses to workers within acceptable limits will be required at all times during the operation. In general, the ALARA principle (As Low As Reasonably Achievable) is applied in practice with formal regulations on radiation doses come from four sources:

- CNSC;
- Denison Safety Standards;
- Future Environmental Impact Assessment; and
- Future Licensing Codes of Practice.

Canadian National Authority - The Radiation Protection Regulations (2000), issued by the Canadian Nuclear Safety Commission, state effective dose limits over specific time periods which a licensee must not exceed. Doses to Nuclear Energy Workers (NEWs) must be below 50 milli-sieverts (mSv) per year and 100m Sv/per 5 years. Effectively, this means the annual exposure of a particular NEW should be less than 20 mSv/a on average. In addition, pregnant NEWs must not exceed 4 mSv during the balance of a pregnancy. Proposing to the Commission any mining activity where a planned annual dose exceeds 20 mSv would be unlikely to receive approval.

In application of the ALARA principle, Denison will be targeting to not exceed a dose rating of 75% of the maximum allowable (i.e. 15 mSv/a) with planned doses in excess of 10 mSv/a will be highly scrutinized. Codes of practice (COP) will be developed based on more detailed assessments and will help guide the operations to achieve annual doses target with shorter time internal targets (i.e. daily, weekly targets). The COP will establish operational controls (i.e. administrative and action levels) and are to be proposed by the licensee and intended to signify the point at which the process is out of control. Industry action levels values are typically set at 1 mSv/week to ensure dose limits of 50 mSv/a are not exceeded

16.6.2 Sources of Exposure

Mine workers may be exposed to workplace radiation hazards including radon & radon progeny, long lived radioactive dust and external gamma radiation. It is important to evaluate the potential workplace radiation levels and associated worker exposures to ensure that the proposed operations are designed to meet the dose constraints with an appropriate margin of safety. Detailed radiation dose assessments were conducted by Arcadis Design and Consultancy in a report titled “Phoenix ISR Screening” dated July 2016 and “Gryphon Radiation Modelling” dated September 2018. Pertinent content has been summarized into this section.

Gamma Radiation

Gamma radiation exposures are controlled through the use of minimizing time around radioactive materials, maximizing distance from them, and introducing shielding materials between the source and the worker. The gamma radiation emitted from a source can be estimated knowing the characteristics of the source, such as ore grade, composition, size, and shape. The effects of geometry and shielding are then modelled to estimate the gamma radiation exposure rate. Where workers are exposed to multiple sources, the exposure rates are summed to give an overall exposure rate. When combined with occupancy information, worker dose can be estimated for each exposure scenario. Gamma radiation exposure will be the primary contributor to dose, and is the limiting factor in the assessment of worker doses.

Radon and Radon Progeny

Radon is a gas that is present with uranium and is released from rock and from water during the mining operation. Radon is a noble gas and generally is not itself a hazard, except in very high concentrations. It is the radon decay products, referred to as radon progeny, which are of concern. The amount of radon that is released into the mine atmosphere not only depends on the ore grade but also on other factors, such as the porosity of the rock, amount of water inflow, air flow patterns, residence time, etc.

It is primarily controlled using dilution of air and continuous air exchanges in a similar way that an underground mine provides ventilation and fresh air to mitigate exposure to exhaust fumes from diesel equipment.

Other

In underground mining, exposure to long lived radioactive dusts (LLRD) is primarily controlled by using water sprays, enclosing workers, and allowing time for dusts to settle after dust generating activities (i.e. blasts). Past experience has shown that, with good dust control measures, LLRD exposures can be maintained at much lower levels than gamma and radon progeny exposures.

16.6.3 Phoenix ISR Dose Assessment

In comparison with conventional mining and processing facilities, ISR mines are smaller as they have no requirement for ore handling, crushing, grinding and leaching processes. In turn this reduces the risk of some exposure pathways, namely Low-Level Radioactive Dust (LLRD), gamma and RDP in the handling of the raw ore.

The major risks for the ISR operation are:

- Large quantities of radon-222 gas can be dissolved in the lixiviant returning from underground and is brought to the surface. That portion of the total dissolved radon which is above the solution's saturation value is released when encountering atmospheric pressures and temperatures and can also be released during the decay of radium contained in waste products being processed and stored at the surface. As a result, there is potential for radon to be released from the lixiviant in the precipitation circuits and containers exposed to atmospheric pressure. In a modern facility with proper ventilation and safe maintenance procedures, dose from radon can be mitigated.
- Due to the high-grade nature of the lixiviant compared to typical ISR operations, increase gamma radiation may be experienced in the process areas. Simple procedures of minimizing worker time near operating vessels and installation of shielding where appropriate will minimize dose.
- LLRD pathway is exclusively associated with the yellowcake drying and packaging areas since up to the drying step, the ISR process is essentially aqueous and the risk of significant dust generation elsewhere in the process is low.

Overall the above aspects are well understood and in modern facilities, exposures are quite low. With planning, ISR is very feasible from a radiation protection perspective and with design and suitable radiation protection practices, the doses to workers can be maintained well below regulatory limits.

16.6.4 Gryphon Underground Operations Dose Assessment

The dose assessment completed at Gryphon focused on four key positions that are likely to be the most exposed due to the length of time and proximity of the work that occurs in the ore zone itself. The four positions include: Longhole driller, development miner, bolter, LHD operator.

A computer model using MicroShield™ was developed to estimate the gamma doses from sources at specific points in the mine. The program is able to model based on different locations of gamma source, variations of ore grade, incorporate shielding, location of workers and a variety of other factors. In completing the dose assessment for Gryphon operations important aspects of the mine design were considered including:

- Ventilation circuits, volumes and other design considerations as specified in the mine design;
- Completed two scenario's using average in-situ ore grade of 2.05% U_3O_8 and a second scenario using 90th percentile of ore grade which is 3.29% U_3O_8;
- Mine opening sizes, lengths, development advance rates, cycle times, ground support designs and use of equipment (including operator cabs and remotely operated equipment) as per the mine design;
- Dry radon emanation rate similar to that of other basin operations; and
- Mine water flow rate based on hydrogeological modelling of the mine.

Exposure rates are combined with duration of exposure at each location to estimate the annual dose. Annual hours are determined using the hours per shift at each location in combination with an assumption of 180 shifts per year. Table 16-24 shows the annual dose by occupation for in average ore grade (2.05% U_3O_8) and at the 90% percentile (3.29% U_3O_8) at the designed ventilation rates.

Table 16-24. Gryphon Estimated Annual Dose

Ore Grade ($\%U_3O_8$)	Ore Grade ($\%U_3O_8$)	Annual Dose (mSv) from RDP Exposure	Annual Gamma Radiation Dose (mSv)		Total Annual Dose (mSv)
			10 cm shotcrete	15 cm shotcrete	
Bolters	2.05	0.54	10.38		10.92
	3.29	0.85		9.23	10.08
Development Miner	2.05	0.77	12.95		13.72
	3.29	1.22		11.58	12.8
LHD Operator	2.05	0.28	3.52		3.8
	3.29	0.32		3.20	3.52
Longhole Driller	2.05	0.80	11.78		12.58
	3.29	1.30		9.75	11.05

Results indicate that the design ventilation rates provide suitable fresh air to workers and maintain a reasonable job of removing RDP sources from the workplace. The advancement of rigid ducting to near the working face combined with the use of remote operation of equipment minimizes RDP exposures. During operations it will be essential to maintain clean work places as radon will emanate from localized point sources of water and or much spillage. Where these occur, they are considered upset conditions and need to be cleaned up to eliminate the hazard.

In all cases gamma radiation is the primary contributor to dose. The use of 10 cm of shotcrete application in ore headings and the use of equipment (including enclosed operator cabins and remote operation) reduces the exposure rating to below regulatory limits. In areas of high grade or where longer cycle times are expected, additional shielding may be required for operator cabins. Provision for such situations can be captured in operational radiation protection planning. It is anticipated that mine planning will be able to provide advance notice when such situations are expected.

In consideration of the As Low As Reasonably Achievable (ALARA) approach, additional efforts will be required to reduce dose rates further. Opportunities to reduce doses include:

- Optimized ventilation plans and including flexibility in the network to increase ventilation on demand to specific areas;
- Incorporate of additional shielding in the operator cabs of equipment;
- Detailed assessment of the dose often indicated that the majority of dose is often received during setup and tear down of operations and/or are often received in specific locations in the drift. Customizing equipment for more efficient setup times, development of operational procedures and education of workers to minimize time in the high exposure areas will significantly reduce exposure rates; and
- Evaluation denser shotcrete might be applicable as an alternative to adding additional layers of shotcrete. A further reduction in gamma radiation dose of about a factor of two is provided by each additional 5 cm layer of shotcrete.

Overall, the Gryphon deposit can be mined working in ore as currently proposed; however, as previously indicated, further optimization is possible and additional ventilation and shielding would be required to reduce worker exposures to meet ALARA principles.

17 Recovery Methods

17.1 Phoenix Deposit Processing at Wheeler River

The uranium recovery or precipitation plant will house most of the process equipment in a 46,500 square foot pre-fabricated metal building. The plant will have four major circuits: impurities removal, yellowcake precipitation, dewatering/drying and packaging. An overview of the ISR process is outlined in Figure 17-1 below.

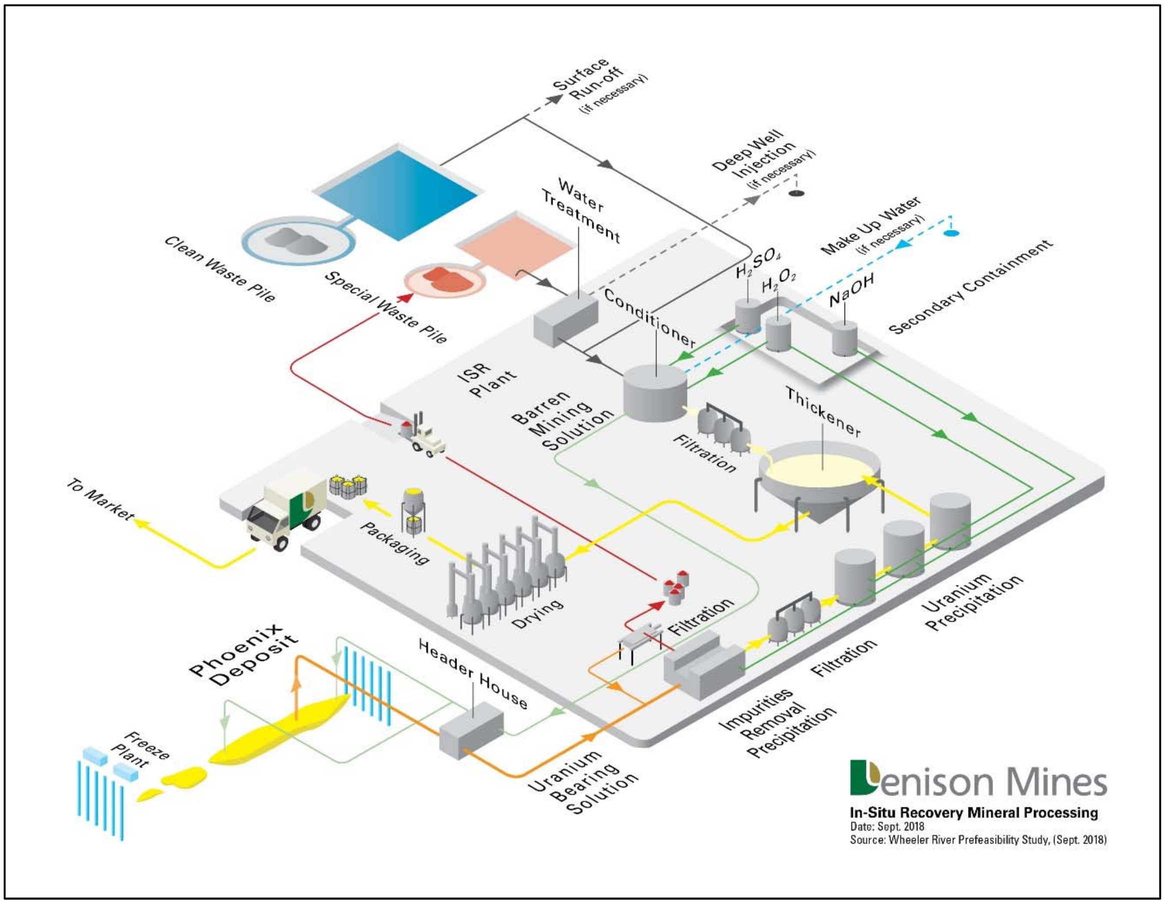


Figure 17-1. ISR Process Overview

Auxiliary equipment consists of filtration systems, bulk chemical storage, process solution storage tanks, water treatment, and a control room.

This evaluation assumes an average uranium head grade of 10 g/L based on the metallurgical test work.

17.1.1 Plant Design

The major process components of the precipitation plant are listed in Table 17-1.

Table 17-1. ISR Processing Plant Equipment List

Equipment Number(s)	Equipment Description	Quantity	Capacity
DR-4001 to DR-4006	Vacuum Drying System	6	3.5 m^3/batch
K-3001	Yellowcake Thickener	1	1,135 m^3
IR-1001 and IR-1002	Iron/Radium Precipitation Skid	2	200 gal/min
FP-1001 and FP-1002	Iron/Radium Filter Press	2	80 ft^3

Source: Engcomp – Wheeler Site Infrastructure & Mining Pre-Feasibility Study Process Mechanical Equipment List, May 9, 2018

These systems have been designed to efficiently recover uranium and to reduce operating costs by recycling and re-using most of the solutions inside each circuit. Removal of lixiviant from the system will not be necessary as the freeze cap maintains containment of the solution in the mining zone. Test work results indicate that constantly removing (or 'bleeding') a small volume of lixiviant to control the level of contaminants in the leach solution is not expected to be necessary. Removal of metals, radium, and other impurities through the impurities removal process is expected to be sufficient to control contaminants buildup in the lixiviant. On a contingency basis, provisions have been made to route a bleed solution to a low-capacity water treatment unit where the contaminants would be removed from the system and resultant water would be re-used as process make-up water. Any excess water from the water treatment system will be reintroduced to the leach refortification stream. Finally, water make-up for the plant will be provided by a water well or from captured surface run-off or contaminated surface facility water sources.

Uranium bearing solution containing dissolved uranium from the wellfields will be pumped to the precipitation plant for beneficiation as described below:

- pH adjustment – The pH of the incoming solution to the plant is constantly monitored and maintained at a specific value to ensure the uranium is fully dissolved through the addition of acid.
- Impurities Removal – the uranium bearing solution is pumped to a series of agitated tanks where sodium hydroxide, commonly known as "caustic soda", and barium chloride are progressively injected, along with a flocculant. The resulting increase in pH and the addition of barium chloride and flocculant promote the formation of metal hydroxides and radium precipitates. The solution flows by gravity to a decanter/settler, allowing the precipitates to sink to the bottom and the clear solution to rise to the top. Metal hydroxides and radium complex precipitates will be directed to a filter press, where 90% of the moisture containing uranium rich solution is recovered. The filtered cake with a 10% moisture content is disposed of in tote bags and stored on the special waste pile on surface in a lined area. The solution overflowing from the decanter/settler is filtered in a series of sand filters, where entrained precipitate is pumped back to the decanter/settler unit. The clear uranium bearing solution is forwarded to the next processing step.
- Yellowcake precipitation – Uranium oxide (U_3O_8), referred in the industry as "yellowcake", is recovered from the solution following the iron/radium removal process. Hydrogen peroxide is injected in a 3-stage series of agitated tanks to precipitate uranium. Additional pH adjustment is provided, if required, by further addition of sodium hydroxide. A thickener provides time for growth of the uranium oxide crystals. The precipitate accumulates at the bottom of the thickener and the barren leach solution (BLS), depleted of uranium, rises to

the top. The BLS is cleaned through a series of sand filters prior to refortification. The precipitated yellowcake product accumulated at the bottom of the thickener is withdrawn at the underflow of the thickener and pumped through a filter press, where excess liquid is removed and circulated back to the thickener.

- Yellowcake dewatering/drying and packaging – Entrained solid particles exiting the filter press are collected and packaged. Fresh water is sprayed on the surface of the cake displaying trapped BLS within the cake, reducing the entrainment of contaminants to the dryer. The remaining moisture is evaporated in a low-temperature dryer (approximately 400^{o}F). Water released from the drying process is condensed, collected, and reused in the plant for reagents preparation purposes. The product drying activity is a batch process, where a specific volume of dewatered yellowcake product is accumulated in a vessel surrounded with a jacket of circulating oil from an oil bath heated at high temperature. The drying circuit is design with 6 dryers capable of producing 1 M lbs/a. Once the moisture is removed from the yellowcake product, the material is transferred into 400 L steel drums by gravity, where it is allowed to cool prior to the installation of covers.
- BLS refortification – The ISR recovery process circulates lixiviant through the mineralized zone and it is expected that, over time, some contaminants may accumulate in the recycled solution. As described previously, it will be possible for a certain volume of BLS solution to be removed and replaced by fresh solution. Any treated effluent generated during treatment of contaminated site run-off will be re-used where possible. Otherwise a groundwater well will provide fresh water to the process to offset any process consumption. Sulfuric acid and hydrogen peroxide are then added to the volume of make-up water. The solution is then mixed with the recycled BLS and re-injected in the wellfield.
- Reagents storage and preparation – Sulfuric acid, hydrogen peroxide, sodium hydroxide, and barium chloride are the main chemicals used in the uranium recovery plant. Acid and caustic serve to adjust pH, an oxidant enables the dissolution of uranium in the ground, and barium chloride co-precipitates radium generated during the uranium leaching process. Other chemicals, such as flocculants, are needed to settle precipitates at different stages of the process.
- Water treatment – It is expected that water treatment is required for contact areas and BLS bleed stream if needed.
    - Runoff from contact areas, such as the special waste facility (SWF), is collected in a contact water management pond, from which it is pumped to the operational phase wastewater treatment plant. It is expected that treatment is required for molybdenum and radium-226 to meet end-of-pipe limits (SRK 2018a). Chloride, aluminum, arsenic, cadmium, copper, iron, lead, selenium, silver, and uranium concentrations in contact area runoff are also expected to exceed provincial surface water quality objectives. Thus, conventional treatment for metals and radionuclides is proposed in the water treatment process. Generated sludge is pumped to the iron/radium removal filtration unit.
    - Process liquid wastes will be transferred to the water treatment plant. There is potential that a BLS bleed stream solution might require treatment. Such stream would be fed directly to a crystallizer. Moisture generated from the evaporation process would be condensed and re-injected in the plant. Dry salts would be collected in tote bags and ultimately disposed of in underground stopes at Gryphon.
- Waste disposal – Waste is generated from the iron/radium removal process, contact areas in the water treatment plant, and potentially the salts generated from the evaporation process. Filter presses remove moisture from slurry streams and collect damp residues

containing 80% solids by weight in tote bags, which are then stored in a dedicated area before permanent disposal in underground stopes at Gryphon. The crystallized salts would also be captured and sealed in tote bags.

17.1.2 Energy, Water, and Process Material Requirements

Chemicals that are anticipated to be used during processing and the assumed annual peak production consumption rates are listed in Table 17-2 below. Based on available radionuclides assay results, only 1% of the radium contained in the tested material dissolves. As such, it is anticipated that barium chloride consumption will be very low. Test work completed to date has not defined an application rate for this chemical. It will be further studied in the next phase of the project. There may be small quantities of other chemicals used at the site which are not listed in the table below.

Table 17-2. Chemical Consumption Rates

Chemical	Consumption Rate
Sulphuric Acid (93%)	1.34 kg/kg U_3O_8
Hydrogen Peroxide (99%)	0.91 kg/kg U_3O_8
Sodium Hydroxide (aqueous - 55%)	1.98 kg/kg U_3O_8

Source: W & C – Process Calculations and Equipment Sizing Wheeler River Uranium ISR Project Revision G August 30, 2018

The different types of chemicals will be stored, used, and managed so as to ensure worker and environmental safety, in accordance with standards developed by regulatory agencies and vendors. The sulfuric acid, hydrogen peroxide, and sodium hydroxide storage areas will include secondary containment. The various acid and caustic chemicals are of potential concern and will be stored and handled with care. To prevent unintentional releases of hazardous chemicals and limit potential impacts to the public and environment, Denison will implement its internal operating procedures consistent with federal, provincial, and local requirements.

Estimates used in the evaluation presented in this document assume the consumption of approximately 46,252,800 kWh annually of electricity to heat and light the precipitation plant and operate the process equipment.

17.2 Gryphon Deposit Processing at McClean Lake Mill

Mineral processing for Gryphon production is based on processing at the McClean Lake mill. The mill is owned by Orano (70%), Denison (22.5%), and OURD Canada (7.5%). The mill is currently processing material from the Cigar Lake mine, however, it has additional licenced processing capacity up to 24 Mlbs U_3O_8 per annum.

Toll milling agreement terms have not been assessed as part of this study. Hatch has relied upon Denison's information from their McClean Lake Joint Venture that mill capacity is available in the timeframes aligning with the Wheeler River prefeasibility production plans, and that a toll milling agreement can be established for Wheeler River deposit materials.

17.2.1 Transportation

Delivery of the Gryphon mill feed to the McClean Lake mill will require construction of a 50 km section of haul road between the McArthur River mine and the Cigar Lake mine. It is expected that this road will be a joint effort with the province of Saskatchewan, and as such, 50% of the total cost for this road has been included in the capital cost estimate herein. Gryphon mill feed will consist primarily of lower grade, coarse, dry muck from the Gryphon deposit. Life-of-mine quantities to be trucked are estimated to be 1,243 kt (dry), hauled conventionally in covered trucks (Figure 17-2).

Figure 17-2. Proposed Gryphon Covered Haulage

17.2.2 Mill History

The mill was designed as a typical acid leach uranium mill. During the design of the mill, allowances were made for potential future mill expansion and for the ability to process high-grade uranium ores, as it was thought that higher grade feed, and feed from other off-site sources, may be processed during its life.

The mill was constructed between 1995-1999 and commissioned in June 1999. The mill commenced production at a rate of 6 Mlbs of U_3O_8 per annum, processing grades of up to 4% U (Remple, 2000) (Schwartz, 2000).

In 2005, regulatory approval was received to modify the mill to receive Cigar Lake ore, including the construction of an ore receiving facility. The mill entered a period of care and maintenance in 2010 when on-site mining was complete and Cigar Lake feeds were not yet available.

A significant mill upgrade project was initially considered in 2009, and implemented in 2012-2014, considering a production rate of up to 22,300,000 lbs U_3O_8 per annum. The mill restarted in October 2014 (AREVA (Orano), 2013).

The mill operating licence has been updated and expanded multiple times during the mill's life, and most recently was approved to process 24,000,000 lbs U_3O_8 per annum, with a 10-year licence renewal to June 30, 2027 (CNSC, 2017). In 2016, the McClean Lake mill produced 17,300,000 lbs U_3O_8, and in 2017, it produced over 18,000,000 lbs U_3O_8.

A process overview of the McClean Lake mill is provided in Figure 17-3.

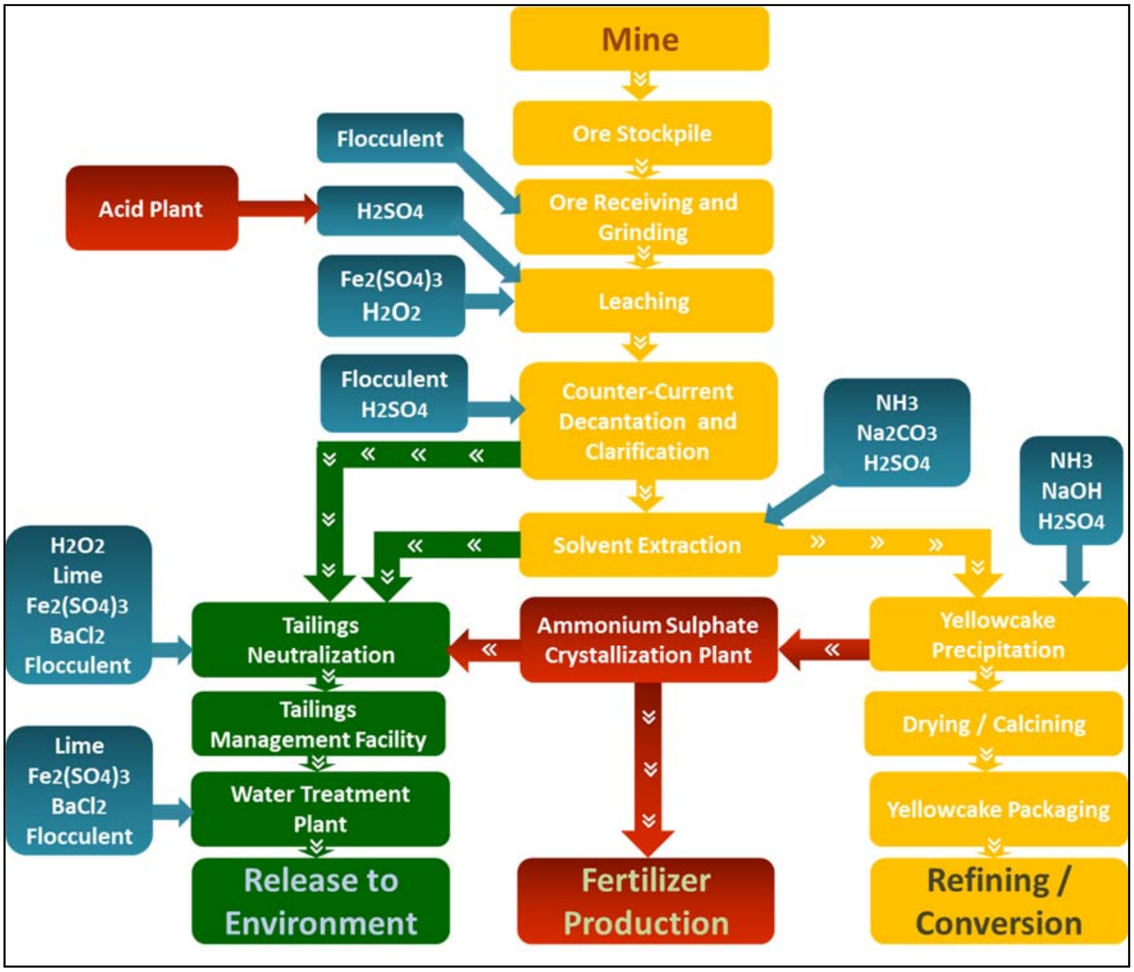


Figure 17-3. Mill Process Overview (courtesy of Orano McClean Lake site guide)

It should be noted that the front end of the mill was originally configured as a two-stage leach circuit. The circuit consisted of three primary leach tanks (60 m^3) which fed a primary thickener. The thickener overflow reported to the clarification circuit and solvent extraction for further uranium recovery, and the underflow reported to the secondary leach circuit, consisting of seven secondary leach tanks (24 m^3). The secondary leach tanks discharged to the CCD circuit where the slurry was washed, with final solids underflow reporting to the tailings neutralization circuit and the overflow reporting to the primary leach circuit. The CCD overflow, containing acid and iron, was used as the leach solution in the primary leach circuit.

During the various mill modifications over the years, the most significant change to the mill design was the configuration of the leaching circuit. Currently, only the seven secondary leach tanks and the CCD circuit are being utilized for leaching and solid/liquid separation. The remainder of the original equipment is still installed in place, although not in service.

In general, no significant modifications have been made to the downstream uranium recovery circuit over the years (from clarification onwards) and only incremental addition of parallel/series equipment has been completed to increase production capacity. This includes the construction of a standalone second SX circuit as part of the 2012-2014 capacity upgrade. Additionally, as part of the same upgrade, a new larger tails neutralization circuit was constructed.

An oxygen plant was constructed on-site in the 2000's in order to utilize gaseous oxygen as the leaching oxidizing agent, however, use of oxygen as the leaching oxidizing agent has been abandoned and the oxygen plant is currently only used to support the ferric sulphate plant. A counter current cyclone circuit was also constructed to support CCD operation; however, it is not currently in service.

17.2.3 Mill Current Configuration and General Process Description

The processing of the Gryphon deposit at McClean Lake is based on the current operating conditions and configuration of the mill. In general, the process conditions required for high recovery of uranium from Gryphon ores align well with and are amenable to current mill operating conditions.

The design basis for Gryphon processing is the co-milling of both Gryphon and Cigar Lake ores through the mill. This will involve separate material receiving circuits for each feed, followed by co-leaching in a single leach circuit. Co-leaching in a single leach circuit will require additional metallurgical accounting activities to assign operating costs and recovery by ore type. It is anticipated that recovery and cost assignment would be accomplished through the use of empirical calculation based on batch test work, however this requires further discussion as part of the overall toll milling agreement.

The mill is currently configured to be fed from either the ore stockpile and grinding circuit or from the ore slurry receiving facility, which is currently used to receive high-grade material from Cigar Lake.

The grinding circuit, consisting of a SAG and ball mill, is currently not in service, and is available to the Gryphon feed.

The existing slurry receiving circuit for Cigar Lake ores consists of a vacuum container unloading system, a neutral thickener for density correction, and slurry storage Pachucas. No change is envisioned to the slurry receiving circuit for Cigar Lake ores.

Over the years, the leach circuit configuration has been modified several times. Currently, the leach circuit operates with only the seven secondary leach vessels in service, which advance to the CCD circuit. The primary leach tanks and the primary thickener are not currently in service. In leaching, sulphuric acid is used to leach the uranium from the ore into solution. Some uranium is not directly leachable and must first be oxidized. Hydrogen peroxide is added to oxidize the uranium to a soluble state and ferric sulphate is added to assist in the oxidation kinetics.

The CCD circuit consists of six thickeners in series and is utilized to separate the uranium containing solution from the barren residual solids. Wash water is added to minimize the aqueous uranium in the final solids from the circuit, which are directed to a tailings neutralization circuit.

The final solids from the CCD circuit, along with waste streams from the SX circuit and other sources throughout the mill, are directed to the tailings neutralization circuit. Ferric sulphate, barium chloride, and lime are added to stabilize any arsenic, molybdenum, radium, and other minor elements that have been solubilized in the process. The final tailings are thickened and pumped to the tailings management facility.

In order to improve SX performance, the uranium bearing solution from CCD is clarified and passed though sand filters to remove any suspended solids from the solution. It is then sent to the two parallel solvent extraction circuits.

In SX, the solution is contacted with an organic solvent, whereby the uranium is selectively transferred to the organic along with any molybdenum. The uranium and molybdenum are then stripped out of the organic phase using anhydrous ammonia into an ammonium sulphate solution, resulting in a purified (with the exception of molybdenum) and concentrated uranium solution.

The pregnant strip solution is passed though molybdenum removal carbon columns, used to remove any molybdenum, which is an impurity in the final uranium product. The further purified solution is then advanced to the yellowcake precipitation circuit, where anhydrous ammonia is used to precipitate ammonium di-urinate (ADU). The ADU is then thickened, densified, and washed though a centrifuge, where it is then advanced to a calciner. The calciner produces a high purity U_3O_8 product that is then packaged for off-site shipment and processing.

Ancillary circuits supporting the uranium recovery process include:

- An acid plant used to produce the necessary acid for leaching.
- A ferric sulphate plant used to produce the necessary ferric sulphate for leaching.
- An oxygen plant to support the ferric sulphate plant.
- An ammonium sulphate crystallization plant, which treats the bleed stream from the uranium precipitation circuit and produces a saleable ammonium sulphate product.
- A tailings management facility (TMF) to safely store the final residues from the process.
- A water treatment plant to treat water reclaimed from the TMF prior to discharge to the environment.
- Reagent receipt and storage facilities, including a lime slaking plant, to support the various mill circuits.
- General plant utilities, including process and fresh water systems, cooling water systems, compressed air systems, and steam.

17.2.4 Production Design Basis

Gryphon ores are expected to be processed at McClean Lake in the same timeframe as Cigar Lake ores. As a result, the processing of the Gryphon deposit at McClean Lake has been approached considering the impact of both feed sources on the mill. Subsequent sections of this report describe the McClean Lake mill modifications required to process both ore sources per the current production plans.

Production Plan and Mill Feed Rate

The basis for production requirements at the McClean Lake mill was established from:

- Wheeler River's Gryphon production plan, as shown in Section 16 of this report.
- Cameco's Cigar Lake NI 43-101 Technical Report, dated March 29, 2016, Section 16.3 (including Table 6-2).
- Gryphon deposit production is anticipated from 2032-2040 at 230 kt/a, containing 9 Mlbs/a U_3O_8.

Cigar Lake:

- Cigar Lake's current production plan (Cameco 2016 NI 43-101) notes published production to 2028 at a rate 18 Mlbs/a. In 2025 through 2028, the grade in Cigar Lake ore begins to decline.

- For the period of Gryphon production, it is assumed that Cigar Lake Phase 2 resources would be processed. It has been assumed that Cigar Lake Phase 2 would operate at a lower production rate of 15 Mlbs/a U_3O_8. The uranium grade for Cigar Lake Phase 2 is unknown, and Hatch has assumed a grade of 15% U_3O_8. Hatch notes that this assumption will need to be validated if/when Cameco declares reserves for Cigar Lake Phase 2.
- Key impurities of arsenic and molybdenum are unknown for Cigar Lake ore. In reviewing the modifications for comingled processing, Hatch has assumed a maximum comingled mill feed of 0.4% arsenic and 0.1% molybdenum during Gryphon ore processing. Hatch notes that this assumption will need to be validated when Cameco declares reserves for Cigar Lake Phase 2.

Table 17-3 identified key processing criteria used in the evaluation.

Table 17-3. Mill Key Processing Criteria

		Cigar Lake	Gryphon	Comingled Feed
Year		2032-2040		
Plant Operating Time	%	89.5	89.5	89.5
Annualized Throughput	t/a	46,123	229,765	275,887
Instantaneous Throughput	kg/h	5,883	29,306	35,189
U_3O_8	wt%	14.98	1.81	4.01
U_3O_8 Annual Production	lb/a	15,000,000	9,000,000	24,000,000
Overall Mill Recovery	%	98.5	98.2	98.4

Source: HATCH – Wheeler River Project – Processing Pre-Feasibility, September 4, 2018

The outcomes of estimates in this report are heavily dependent on the grades of both feed sources into the McClean Lake mill. Publicly available data for Cigar Lake only covers years prior to Gryphon production. Data was requested but not received from Denison (as the information is not publicly available) on the expected future grades of Cigar Lake ore to support this study. Any change to the estimated feed grades from Cigar Lake (based on the last years of Cigar Lake Phase 1 production) during Gryphon production will materially impact the outcomes of this report.

A toll milling agreement will need to be established for the processing of Gryphon feed at McClean Lake. The McClean Lake mill currently has excess capacity and it is assumed that WRJV will be able to reach a toll milling agreement.

A co-milling approach, where both ore sources are received separately but then are processed together in the leaching and all downstream circuits, has been assumed in order to maximize the existing mill infrastructure and reduce capital costs. Although both ore sources will be processed together, each ore type will have its own slightly different recovery and reagent requirements. As a result, increased metallurgical accounting will be required for co-milling so that operating costs and recovery can be assigned back to the respective feed. The McClean Lake mill is currently licensed to 24 Mlbs of U_3O_8 per annum. This study considers the McClean Lake mill operating up to but not exceeding this licensed limit.

The current design basis for the mill is approximately 22.3 Mlbs per annum of U_3O_8, thus modifications have been considered to allow the mill to reach full licensed production.

As part of this study, Hatch has not been engaged to review the tailings management facility. It is assumed that sufficient tailings capacity will be made available.

Operating Time

The McClean Lake mill is expected to operate 24 hours per day, 327 days per year (89.5% utilization). This should be achievable based on industry best practice, and past mill performance prior to the 2009 shutdown. Since restart in 2014, due to excess production capacity and higher feed grades, the mill has not yet had to achieve this utilization target.

17.2.5 Process Modifications

Utilizing the process design criteria established in Section 13 and the processing requirements outlined above, a review of the required process capacity for the McClean Lake mill was completed. This review consisted of developing mass balances for each of the processing scenarios previously outlined and comparing the required operating rate and conditions to the currently installed equipment in the mill, on a circuit by circuit basis.

In completing this review, Hatch has relied upon current as-built engineering data, the current mill process design criteria, and benchmarking data from Orano on the maximum achieved capacity in each circuit.

The capacity review has been completed to a PFS level. In some areas of the mill, test work is not available, and Hatch has used historical requirements and industry best practice. Moreover, test work is not yet available on comingled ores, which may change the requirements for some circuits. Additionally, although the most recent mill upgrades should allow up to 22.3 Mlbs/a U_3O_8, this has not yet been demonstrated in all circuits on an ongoing, sustained basis.

The following sections outline the McClean Lake mill upgrades required to support co-milling of Gryphon and Cigar Lake ores.

Ore Storage and Receiving

Gryphon ores will be transported to site by highway truck and offloaded onto the ore pad. Sufficient space will be allocated on the ore pad for surge capacity at McClean Lake.

Gryphon ore will be reclaimed from the ore pad by front end loader and transferred to the existing grizzly. The existing grinding circuit will be utilized with no major modifications expected. It is anticipated that the grinding circuit will operate at ~30 t_{ore}/h, 24 hours per day.
Gryphon ore will be metallurgically sampled, and then transferred from the grinding circuit to two existing storage Pachucas in the leaching circuit, providing 4 hours of residence time.

Cost allowances have been included to refurbish the SAG and ball mills and optimize them for Gryphon operation. This includes allowances to install a variable frequency drive (VFD) and tramp material protection, and to make minor piping modifications. Additionally, the ore dump pocket requires retaining wall refurbishment, which has been accounted for in cost estimates.

Leaching

Gryphon and Cigar Lake ores will be blended into two existing leach feed Pachucas in the leaching circuit. Each ore source, as well as the comingled leach feed, will be metallurgically sampled.

The leaching circuit will need to be further expanded to maintain the required 8-hour residence time for high uranium recovery. Three additional primary leach tanks will be installed in place of the other existing primary leach tanks. The new primary leach tanks will be added in series with the existing secondary leach tanks. Moreover, two of the existing decommissioned tailings neutralization tanks will be replaced with new leaching tanks. These tanks will be added in series with the end of the leaching circuit.

The leach circuit will maintain all existing hydrogen evolution safety features, and the new primary leach tanks will be equipped with the required ancillary equipment per the secondary leach tank design. Additional cooling circuit capacity is assumed to be required, as well as additional sweep air ventilation for hydrogen evolution protection.

Solid/Liquid Separation

Leach circuit discharge will continue to report to the existing CCD circuit, however, due to the increased solids tonnage, a parallel filtration circuit will be added to manage the excess solids.

As most of the solids in the mill will be from Gryphon ore, initial settling tests show that Gryphon material should respond well in the CCD circuit, with potential increased throughput from the existing baseline. However, comingled test work (with Gryphon and Cigar Lake ores combined) has not been completed, which may impact settling performance.

The new filtration circuit (including a duty and standby filter) will be added adjacent to the CCD circuit. The filter press will utilize wash step to achieve a similar uranium recovery as the existing CCD circuit.

Tailings Neutralization

The flowrate will increase to the tailings neutralization circuit during Gryphon ore processing. No change to the tailings neutralization tanks is expected. However, the tailings thickener will not have sufficient capacity. As a result, a new parallel tailings thickener and associated pumping has been included as part of the mill upgrades for Gryphon.

Clarification

Changes are not expected to the existing clarification circuit.

Solvent Extraction

Modifications are not expected to be required for the solvent extraction circuits. Maximum sustained capacities to date indicate that continuous operation of the two existing solvent extraction circuits for a combined rate of 24 Mlbs/a U_3O_8 should be possible without any modifications.

Precipitation

The existing uranium precipitation circuit is not expected to require any modifications to allow for 24 Mlbs/a U_3O_8 production. This assumes the use of all three currently installed yellowcake precipitation tanks. Orano is currently in the process of converting one of the precipitation tanks to support the calciner off-gas circuit. If this conversion is still in place during the processing of Gryphon ores, a new tank will be required. It is suggested that during the feasibility study, testing be completed to see if acceptable yellowcake precipitation can be achieved with the reduced residence time of two precipitation tanks.

Calcining

The calciner circuit is assumed to be able to handle the increased production rate of 24 Mlbs/a U_3O_8. As part of the capacity review, Hatch has included the cost to upgrade the calciner feed centrifuge to support the larger flow of yellowcake solids to the calciner and the required washing. It is recommended that a detailed review of calciner capacity be completed during the feasibility study.

Reagent and Support Plants

Ammonium Sulphate Plant (CX)

It is expected that, with the higher uranium production, the bleed to the CX plant will increase. A detailed review of the CX plant was not completed. However, an allowance has been included to support the upgrade of the ammonium sulphate crystal material handling equipment. This includes the final product dryer and centrifuge. Moreover, the associated CX bleed ion exchange system is anticipated to require expansion, and 4 additional ion-exchange columns have been included as part of the mill modifications.

Acid Plant

The existing sulphuric acid plant will require upgrades for Gryphon processing. A nominal acid requirement of 180 t/d is required for Gryphon processing. As part of the review, a design acid requirement was established, based on the higher acid demands noted during the test work that can occur as a result of variability in the Gryphon mineralogy. In the case of the design acid consumption, a total acid requirement of 225 t/d could be required. As a result, the cost to expand the acid plant by 100 t/d, for a total onsite capacity of 225 t/d, has been established for Gryphon.

Ferric Sulphate Plant

The ferric sulphate plant should be sufficient for nominal Gryphon requirements. However, as part of the review, a design ferric requirement was established for Gryphon of 128 t/d, which will exceed the current plant capacity. As a result, it is expected that the ferric sulphate plant will need to be expanded to accommodate the new demand. One additional ferric sulphate batch reactor and ancillary equipment has been included as part of the mill upgrades for Gryphon.

Utilities

The processing of Gryphon ores will increase the overall site operating load, because of the operation of the grinding circuit and the additional equipment required in leaching and CCD. It is not anticipated that this increase in loads will require any significant modification to the existing site electrical infrastructure.

Significant changes to the site water balance or fresh water requirements are not currently expected to process Gryphon ores.

17.2.6 McClean Lake TMF

Tailings storage at the McClean Lake facility are provided by the existing tailings management facility. On April 19, 2017, the Canadian Nuclear Safety Commission (CNSC) approved a 1.7M m^3 expansion to the TMF. The expansion, along with the existing capacity in the TMF pit, provides for ~2.4M m^3 of tailings storage. Milling of Gryphon ore is expected to produce ~ 1.1M m^3 of tailings.

It is therefore assumed for the PFS that the existing capacity will be available for the Gryphon toll milling and would be part of the toll milling agreement. Cost estimates for toll milling fees include TMF storage fees.

18 Project Infrastructure

18.1 Access Roads and Site Preparation

Main land access to the sites is from Saskatchewan Highway 914, via a 5 km access road to be constructed approximately 35km north of Key Lake. During the PFS, an assessment was completed to evaluate access route options to both the Phoenix and Gryphon deposits. Several routes were analyzed for key factors, including length, cut and fill quantities, distance from cabins, distance from waterbodies, and distance from water crossings. The analysis was shared with local communities and, after consultation, the preferred routes were selected and incorporated into the design.

After exiting the provincial highway, the route heads 3 km NW, skirting to the north of an existing gravel pit before turning NNW another 2.5 km and arriving at the Phoenix site, the first of the two deposits to be developed. The road will continue another 0.5 km before turning NW for a further 1.3 km to intersect with an existing access road. About 4 km of the existing road to the NE of this intersection will be upgraded to access the new airstrip, which will roughly parallel the road, on its east side. Access to the future Gryphon site will be on a road extension from the NW quadrant of this intersection. It will extend about 2 km NNW to the Gryphon deposit, shaft, and service buildings.

Additional on-site Phoenix access roads will provide a service loop to the camp and on-site electrified parking. Also required is a short service road to the runoff pond and the designated treated effluent discharge point.

Site preparation earthworks will first be undertaken at the Phoenix deposit site, precipitation plant, and designated camp and services areas.

18.2 Project Site Layout

Figure 18.1 is a plan view of the Wheeler River project, showing the Gryphon and Phoenix deposits relative to the existing Wheeler River camp and Provincial Highway 914. The Gryphon deposit is roughly 3 km NW of the Phoenix deposit.

With the exception of the airstrip, all common facilities and services will be provided at the Phoenix site, as it will be developed first. Gryphon personnel will be housed at the camp facilities on the Phoenix site.

Production from the Gryphon deposit will be trucked to the existing McClean Lake mill to the NE, via existing Provincial Highway 914, including 45 km of new road between the McArthur River mine and the Cigar Lake mine.

A 1,600 m long airstrip is positioned in a natural, relatively flat valley to the NE of the Phoenix deposit. Its magnetic headings are 03/21, identical to the Key Lake airport, and similar to the Collins Bay airport (02/20). The runway length is the same as five other existing airports in northern Saskatchewan serving existing operations and will accommodate the usual aircraft presently used by these operations for transporting personnel into and out of the sites. The approach line to the airstrip from the SW clears the Phoenix surface facilities by 500 m.

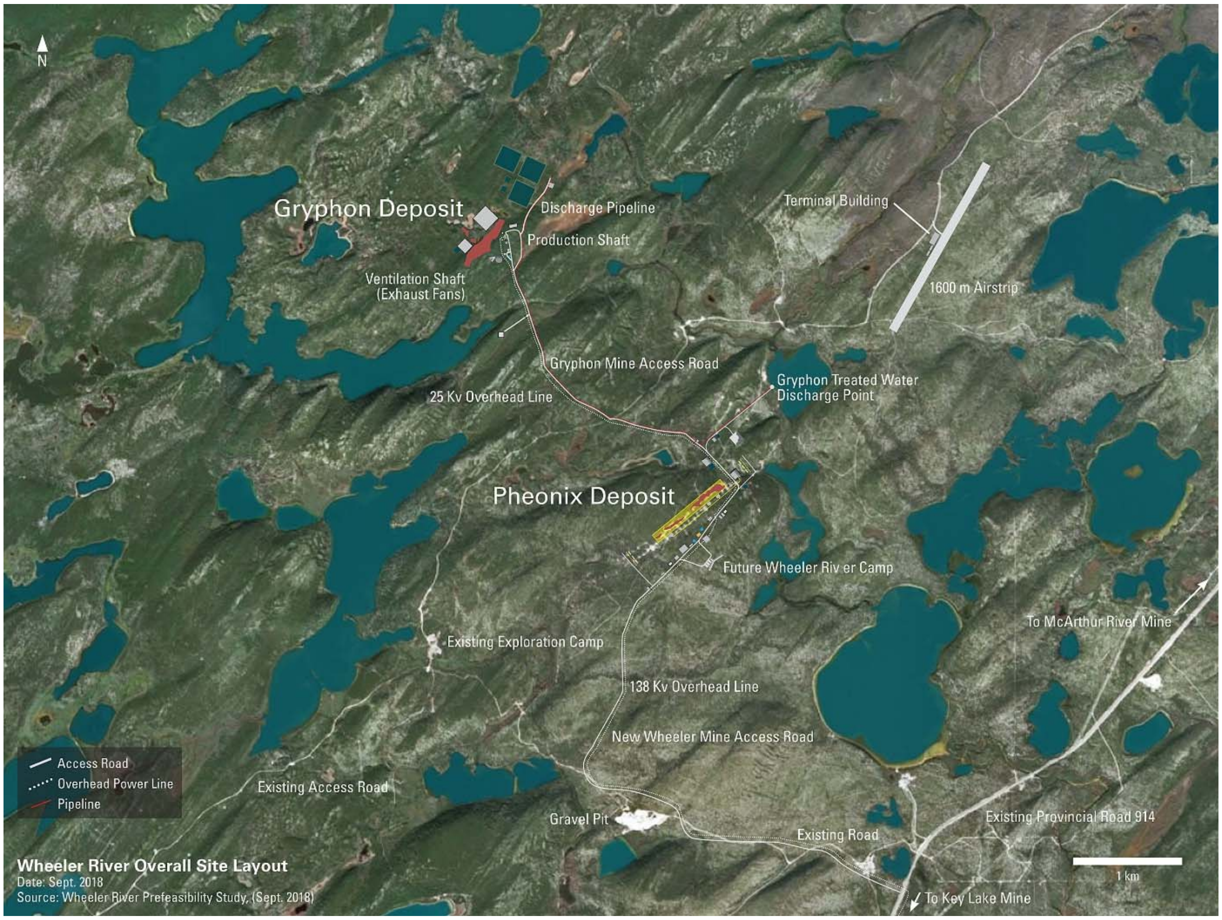


Figure 18-1. Wheeler River Project Site Showing Phoenix and Gryphon Deposits

18.3 Phoenix Site Layout

Figure 18.2 is a conceptual layout of the plan view of the Phoenix deposit surface facilities, showing the relative scale and nominal footprint size of major infrastructure items, as listed below. It should be noted that, since Phoenix site will be the first into production, it will host many of the sitewide infrastructure items for the entire operation including.

- Area allocation over the defined deposit for an in situ leaching wellfield (90 m x 800 m);
- ISR plant (90 m x 48 m);
- Operations centre (61 m x 41 m), including men's and women's dry facilities, 3-bay maintenance shop, welding bay, warehouse, emergency response vehicle storage, mine rescue and emergency response office, laboratory, nurse's station, training room, offices (administration, maintenance, and supply chain), meeting rooms, lunch room, and radiation monitoring room;
- 150-person camp with kitchen and laundry facilities;
- Personal vehicle parking;
- Main electrical substation (50 m x 50 m);
- North and south gatehouses;
- Outdoor fenced storage (15 m x 30 m);
- Covered storage (15 m x 30 m);

- Wash bay and scanning facility;
- 30 m long, 80 tonne weigh scale;
- Potable water treatment facility;
- Fuel storage and dispensing facility (gas and diesel);
- Fire water tank and pumphouse;
- One bullet propane tank;
- Sewage treatment facility;
- Incinerator;
- Backfill plant with storage facility;
- Outdoor fenced hazardous storage area (30 m x 30 m);
- Fenced landfill area (90 m x 90 m);
- Water discharge station;
- Special waste storage (46 m x 46 m, 3,200 cubic meter capacity); and
- Clean waste rock storage (60 m x 60 m, 7,100 cubic meter capacity).

The major tenets listed below were considered for the siting and relative positioning of the various facilities at the Phoenix site.

- The ground surface area directly over the defined underground deposit is reserved for wellfield drilling and piping equipment.
- As site access is from the south, the main access road runs parallel to and on the south side of the deposit, with a nominal 180 m separation from the area reserved for surface wellfield development activity.
- Most surface support facilities are positioned along the NW side of the access road, which also serves as the main product transport truck transfer route.
- The camp, with its 'gull-wing' layout affording views from the east-side rooms of an existing body of water, is located on the crest of the natural land rise to the SE of the deposit, well-separated (220 m) from the main site access road and operation centre.
- The main ISR precipitation plant is located to the NW of the east end of the reserved wellfield area, approximately 900 m direct-line separated from the camp and about 1 km of road driving distance from the operations centre.
- Sources of fumes or odours (incinerator, landfill, fuel storage) are located to the east of the camp and operation centre.
- The run-off/capture pond is located in a low-lying area, just SW of the designated discharge point.
- Storage areas are located at the west entry side of the site, just inside the south gatehouse, to reduce truck traffic on site.
- The weigh scale is positioned immediately east of the south gatehouse, as shipping is expected to be monitored by gate security.

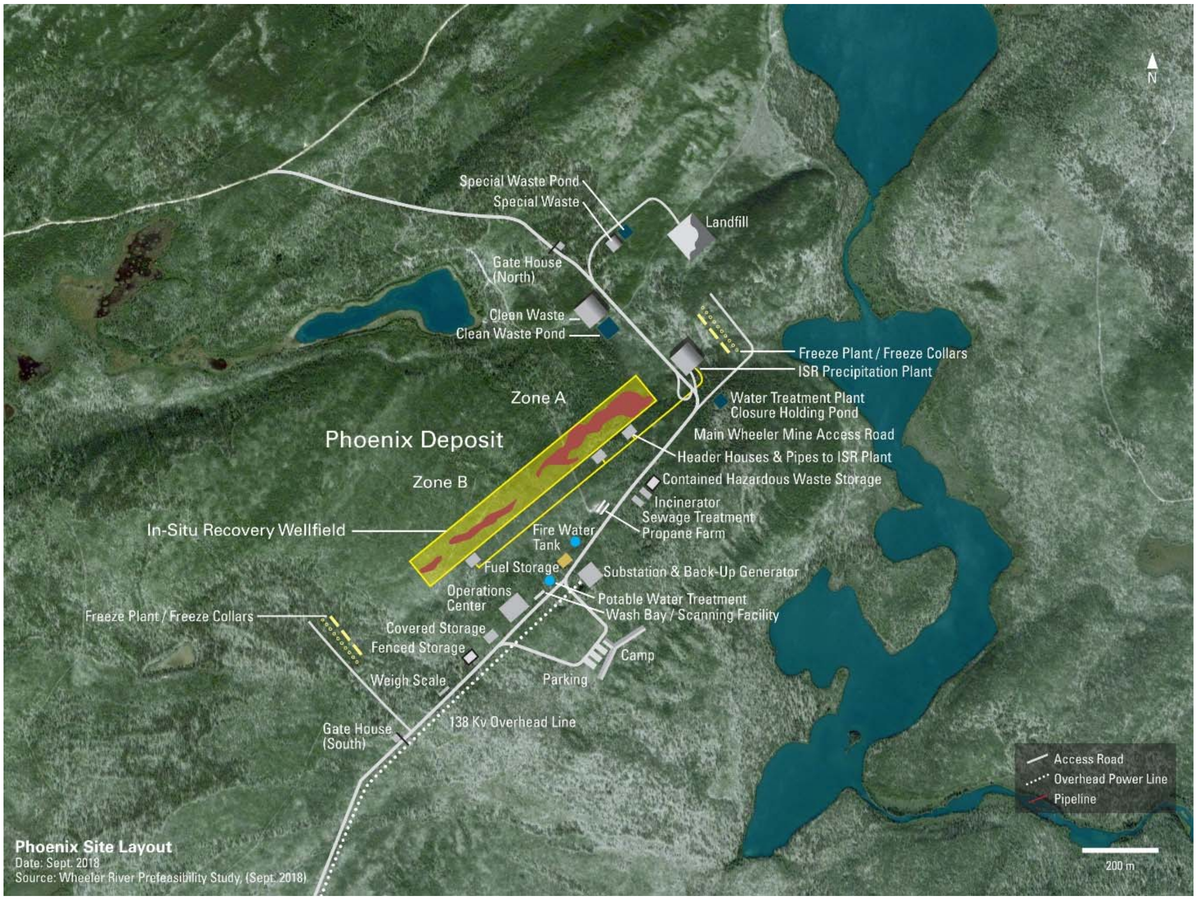


Figure 18-2. Phoenix Site Conceptual Layout

18.4 Gryphon Site Layout

Figure 18.3 is a conceptual layout of the plan view of the Gryphon surface facilities, showing the relative scale and nominal footprint size of major infrastructure items, including:

- Headframe and collar house for the 5.5 m diameter production shaft;
- Hoist house and production/service hoist for production shaft;
- Hoist house and hoist for auxiliary cage in production shaft;
- Headframe, hoist house, and hoist for auxiliary cage in ventilation shaft (secondary egress);
- Fresh air ventilation fans and propane fired air heaters with ventilation plenum at headframe;
- Surface ore storage (55 m x 55 m, 3,000 cubic meter capacity);
- Clean waste rock storage (104 m x 104 m, 45,000 cubic meter capacity);
- ARD/ML waste storage (180 m x 180 m, 210,000 cubic meter capacity);
- Main south gatehouse;
- Operations centre (20 m x 20 m), including space for mine rescue equipment and facilities, and a number of small offices;
- Backfill plant (20 m x 20 m) and 60 m diameter backfill aggregate pile;
- Electrical room module (20 m x 6 m);
- Explosives magazine (50 m x 50 m), with a designated security gate on its access road;

- Outdoor fenced storage (15 m x 30 m);
- Fuel storage and dispensing facility (gas and diesel);
- Fire water tank and pumphouse;
- Three bullet propane tank farm (close proximity to headframe);
- Water treatment plant (40 m x 40 m);
- Two water treatment plant holding ponds (each at 48 m x 48 m, 7,500 cubic meter capacity); and
- Three water management ponds (each at 200 m x 200 m, 270,000 cubic meter capacity).

The major tenets listed below were considered for the siting and relative positioning of the various facilities at the Gryphon site.

- All access into and out of the Gryphon site is from the south on a new, 2 km long road extending from the Phoenix site development and the road leading to the new airstrip.
- Shaft coordinates are dictated from previous studies and recommendations, which locate it SE of the defined deposit.
- The main road will run past the east side of the production shaft with an eastern 'loop' allowing for truck travel/access to all sides of the ore piles. This eastern arm will extend NNE to access the water treatment plant and ponds, which are positioned in natural low-lying area.
- Since personnel will arrive on buses from the Phoenix camp, a drop-off area is shown south of the new operation centre building, which is located south of the production headframe. A loop road will allow for easy exiting to the south from this loading area. This loop will also provide access to the backfill plant and aggregate pile.
- The propane storage facility and electrical room module are located in close proximity and north of the production headframe.
- The explosives magazine is separated from the other facilities at the Gryphon site, 700 m to the south of the ventilation shaft. Access is secured by a separate gate to the south of the main gatehouse.

Figure 18-3. Gryphon Site Conceptual Layout

18.5 Freeze Plant Surface Infrastructure

Freeze plants with a total capacity of 1,500 tons of refrigeration will be constructed on surface based on a modular design for easy installation and operation. The design includes:

- Six modular freeze plant skids;
- One electrical/control skid;
- Six evaporative condenser skids; and
- One insulated brine tank.

The freeze plant system being proposed for this project is "modular", which means that a shutdown in any one unit will not result in complete plant downtime. Having one unit offline during early freezing will mean the brine temperature supplied to the ground will warm slightly and the freeze duration extended, but breakdowns in new equipment near to their commissioning are not typical. If breakdown or maintenance takes a freeze module offline once the freeze is established, that is not such a concern since, over time, the ground heat load tends to decay and eventually a module will be intentionally taken offline to serve as back-up.

18.6 Gryphon Production Shaft

Primary access to the deposit will be by via a production shaft connecting surface to the main shaft station at 500 Level. The shaft will be used to transport personnel and materials underground, hoist blasted stoping and development material to surface, and provide the main fresh air route for the mine. The shaft will be excavated to a depth of 550 meters from surface at a diameter of 6.1 meters using a blind boring method.

In the blind boring method, a bottom assembly consisting of a cutting head and stabilizers, driven by a rotary table on a surface drill rig, will break the rock at the advancing shaft face. The excavated, unlined shaft will be kept flooded with water, drill mud, or a combination of the two, and the drill cuttings (broken pieces of rock) will be air lifted to surface up the drill pipe. Thus, blind boring will be a highly automated process and a non-entry excavation method, reducing the risk to personnel during construction.

By keeping the shaft flooded, a concentric outward pressure will be maintained on the excavated wall. This concentric pressure will reduce the potential for ground movement and increase overall excavated wall stability without the requirement of ground freezing or a grout cover.

Once the full depth of the shaft is reached, a water tight shaft liner will be installed leaving an inside diameter of 5.0 meters. The liner will be a pre-fabricated composite liner consisting of a 12 mm thick outer steel layer with a 540 mm thick inner concrete layer. The liner sections will be ‘floated’ into place. This will be accomplished by sealing the bottom liner section with a bulkhead that turns the liner assembly into a floating sealed unit. As each liner section is welded to the top of the assembly, water will be pumped into the liner to equalize the buoyancy and allow a controlled sinking of the liner sections. Once the liner reaches full shaft depth, the annulus between the liner and excavated wall is backfilled via tremie grouting and the water within the liner will be pumped out to permit access for installation of the shaft furnishings.

Production requirements are to hoist a maximum of 750 tonnes of ore and 400 tonnes of waste rock per day. To meet these requirements, the shaft will be serviced by a single hoist plant operating two cages over skip conveyances, only one of which will have a safety mechanism (dogs). The conveyances will operate from surface to the main access level at 500 m depth and the loading pocket at the 520 m level. The cage over skip assembly without the safety mechanism will serve as material transport only.

In October of 2017, Denison completed a shaft test hole drilling program to confirm the suitability of the proposed production shaft location and gather key information to support the design and cost estimation process of the shaft excavation and liner design for the PFS. The test hole was drilled through 16 metres of overburden, contacted the unconformity at a depth of 465 metres below surface, and was ultimately terminated at a depth of 600 metres below surface. Data collected via logging and on-site and off-site test work was summarized in the Denison report entitled “Wheeler River Project: Shaft Pilot Hole” and includes:

- Geological logging including lithology, alteration, and structural mapping;
- Abrasion and strength testing including Brazilian tensile strength, UCS, triaxial shear, drillability, bit ware index, and cutter life index and PLT;
- Freeze testing on intact fault gouge or clay weathered rocks for bulk moisture content and density/specific gravity;
- Geotechnical triple tube logging;

- Packer testing on 25 metre intervals and water sampling for dissolved metals; and
- Geophysical bore hole analysis including gamma, density, neutron, acoustic televiewer, full waveform sonic, mechanical caliper, and resistivity.

The shaft pilot hole data was shared with Frontier-Kemper (FK), a civil and building construction company offering diversified general contracting and design/build services including tunnelling and mining throughout the world, to provide shaft liner design and blind boring cost and performance guidance.

18.6.1 Hoists/Hoist House

The hoisting system will consist of one double-drum, single clutch, gear driven hoist and will be selected based on an assumed daily availability of 16 hours for ore, waste, personnel, and materials movement. The hoist plant is designed for fully automated or manual skipping, or for personnel and materials movement. The hoist will be designed as a base-frame mounted machine.

The hoist house is a basic pre-engineered building on a slab foundation, where outdoor type transformers will be used to minimize cooling. The building will be designed for rope penetrations through the walls and will contain partitions for the electrical room and a pre-fabricated type hoist operator booth. Critical hoist data is provided in Table 18-1.

Table 18-1. Critical Hoist Data

Item	Specifications
Drum Diameter	3.5 m
Drum Width	1.52 m
Rope Diameter	43 mm
Minimum Rope Breaking Strength	1,147 N
Rope Wire Grade	1,770 MPa
Skipping Speed	10 m/s
Personnel and Materials Speed	6 m/s

18.6.2 Headframe

The headframe will include both shaft compartments, a stairwell, and an open sheave deck. The dump will be a scroll type dump, with a dump chute to a ground based concrete dump bin. This dump bin will be mucked out with a wheel loader.

A collar house will be included within the headframe. Services will be transitioned into the shaft at the collar, or the sub collar if the design allows for it. Critical headframe data is provided in Table 18-2.

Table 18-2. Critical Headframe Data

Item	Specifications
Height to Sheave CL	37.5 m
Height to Dump Lip	18.85 m
Slinging Clearance	17 m
Ground Based Dump Pile Capacity	48 tonnes

18.6.3 Shaft Furnishings and Conveyances

The production shaft will be a circular cross-section, hydraulically sealed shaft with a composite liner. It will have two main conveyance compartments which will be defined by shaft brackets cantilevered from the shaft wall. The shaft will include a small auxiliary hoist described below. The conveyance guide system will be conventional timber shaft guides. The shaft will also include a ramp to provide access to shaft bottom and a ventilation duct that will be designed to operate under negative pressure. Critical conveyance data is provided in Table 18-3.

Table 18-3. Critical Conveyance Data

Item	Specifications
Skip Payload (wet)	8,000 kg
Assumed Broken Muck Density (ore and waste)	1,925 kg/m^3
Lump Size	0.4 m
Cage Payload	8,000 kg
Personnel Capacity	17
Cage Interior Width	1.67 m
Cage Interior Depth	1.67 m
Cage Interior Height	2.5 m

Services in the shaft will be wall mounted and will include two 25 kV power feed cables, fibre optic cable, two bare copper ground cables, four spare fingers, one 100 mm diameter process water line, and three 300 mm diameter dewatering lines.

The shaft will be serviced by one hoist and no ladderway is included, except below the 500 Level. To meet the Mines Regulation of Saskatchewan, the rescue plan for personnel stuck in a shaft conveyance will be to remove them via an auxiliary hoist backed up by a diesel generator. This conveyance will be guided by two HSS guides and will have a capacity for three people. Critical shaft data is provided in Table 18-4.

Table 18-4. Critical Production Shaft Data

Item	Specifications
Diameter	5.0 m
Depth	550 m
Main Access Level Depth	500 m
Loading Pocket Access Level	520 m
Loading Pocket Lip	535 m
Downcast Air Velocity	10 m/s
Conveyance Compartment Width (Guide F/F)	1.53 m
Largest Underslung Object Dimensions	Irregular (>1.5 m)

18.6.4 Loading Pocket and Shaft Bottom

The loading pocket will be a volumetric, double measuring flask arrangement and will be fed via a single pass and fully automated. The pass will include a truck dump with a grizzly and rock breaker arrangement and will terminate at the loading pocket, with hydraulically operated knife gates to control flow into the two flasks.

Beams will be installed close to shaft bottom to prevent a conveyance from being lowered into water. A ramp will be included in the shaft between the 500 Level and shaft bottom to access the shaft high water alarm sensor and shaft bottom water pump.

18.7 Gryphon Ventilation Shaft

The ventilation shaft will be excavated using the same blind boring method as the production shaft. In the case of the ventilation shaft, however, the shaft depth will be 500 metres from surface at a diameter of 5.8 meters. On completion of a water tight steel-concrete composite liner installation, the final inside shaft diameter will be 4.5 meters.

The ventilation shaft will serve as an exhaust air route and as a second egress from the mine.

18.7.1 Headframe

The headframe will be sealed from the exhaust fan below and will be put in place to support and protect the emergency egress cage to a height suitable for access, egress, and maintenance.

18.7.2 Shaft Furnishings and Conveyances

Shaft design is based on a blind boring excavation methodology, which produces a round cross-section, hydrostatically sealed steel lined shaft with an interior composite steel/concrete liner. The ventilation shaft will contain no main conveyances but will have an emergency egress cage as a use for secondary egress. Services within this shaft will include fibre optic cable, two bare copper ground cables, four spare fingers, three redundant spare 300 mm diameter dewatering lines, and backfill lines. There will be no fuel, slickline, or drain line within this shaft. Critical shaft data is provided in Table 18-5.

Table 18-5. Critical Ventilation Shaft Data

Item	Specifications
Diameter	4.5 m
Depth	500 m
Main Access Level Depth	500 m
Upcast Air Velocity	15 m/s

18.8 Ventilation Fans and Mine Air Heaters

The mine will be designed to have dedicated fresh and return ventilation raises/drifts, providing flow through ventilation in all production mining areas to mitigate recirculation of contaminated air. The ventilation system will be designed to operate mainly as a pull system, to reduce leakage throughout the mine, with a push fan at the intake directing the air through the heater and into the main fresh air raise (production shaft). The air will exhaust through a series of internal return air raises, return air transfer drifts, and the ventilation shaft to surface. Additional detail on the ventilation system is provided in Section 16.5.7.

18.9 Camp

The camp facility is located at the Phoenix site area, as this will be the original development at Wheeler River. It has been situated to the southeast of the industrial facilities on a natural rise to take advantage of the vista to the southeast that includes a large lake.

The camp facility has been sized to accommodate a peak load of up to 150 individuals on site at any one time. It has been designed with modularity in mind. If additional beds are required in the future due to unforeseen developments in operations, they can be easily added on as additional modules.

It is a turnkey modular design that will be manufactured in Saskatoon and assembled and commissioned on-site.

The design includes a central services complex, housing all the central services required for hospitality, recreation, and the smooth operation of the facility. Included in the central area are:

- Kitchen/preparation area and servery;
- Dining room;
- Camp office;
- Commissary;
- Recreation area; and
- Exercise facilities.

The rooms are single occupancy (Figure 18-4) and will not be occupied on back shifts. Washroom and shower facilities will be shared.

Figure 18-4. Typical Room Layout

18.10 Operations Centre

The operations centre has been sized and designed as a standalone, multi-functional building that will serve the administrative, technical, and maintenance needs of the site.

The building is a two-story pre-engineered structure with total usable space of 38,000 ft^2; 27,000 ft^2 on the first floor and 11,135 ft^2 on the second floor.

The first floor houses the two-story shops, dry space, and warehouses. The shops include three full-sized maintenance bays, with one being equipped as a welding bay. Men's and women's dry space is provided, with suitable wash spaces for each, including laundry facilities. The warehouse has two receiving doors and is adjacent to the shops. Office spaces are also provided in these areas for warehouse and procurement staff, and maintenance supervisors.

Additional facilities on the lower level include:

- Nurse station with waiting area;
- Parking space for emergency response vehicles;
- Space for storage of mine rescue/emergency response gear and supplies;
- Laboratory facilities, complete with radiation testing facilities;
- Training room; and
- Mechanical and electrical services rooms.

The top floor is comprised of an administrative area with offices and meeting room spaces, washroom facilities, and a lunch room. The general arrangement of the main and second floors of the operations centre is provided in Figures 18-5 and 18-6.

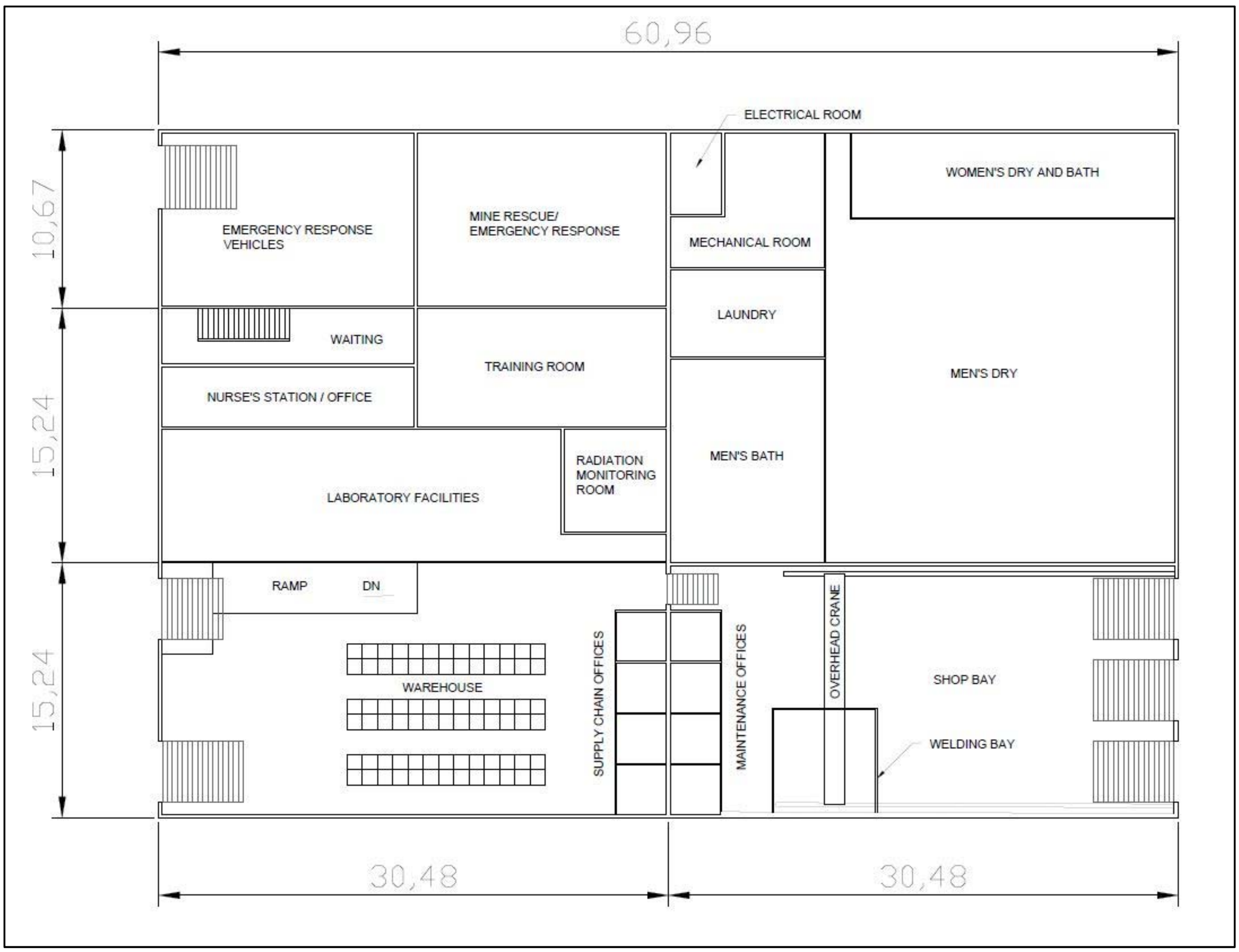


Figure 18-5. Proposed Layout for Main Floor of Operations Centre

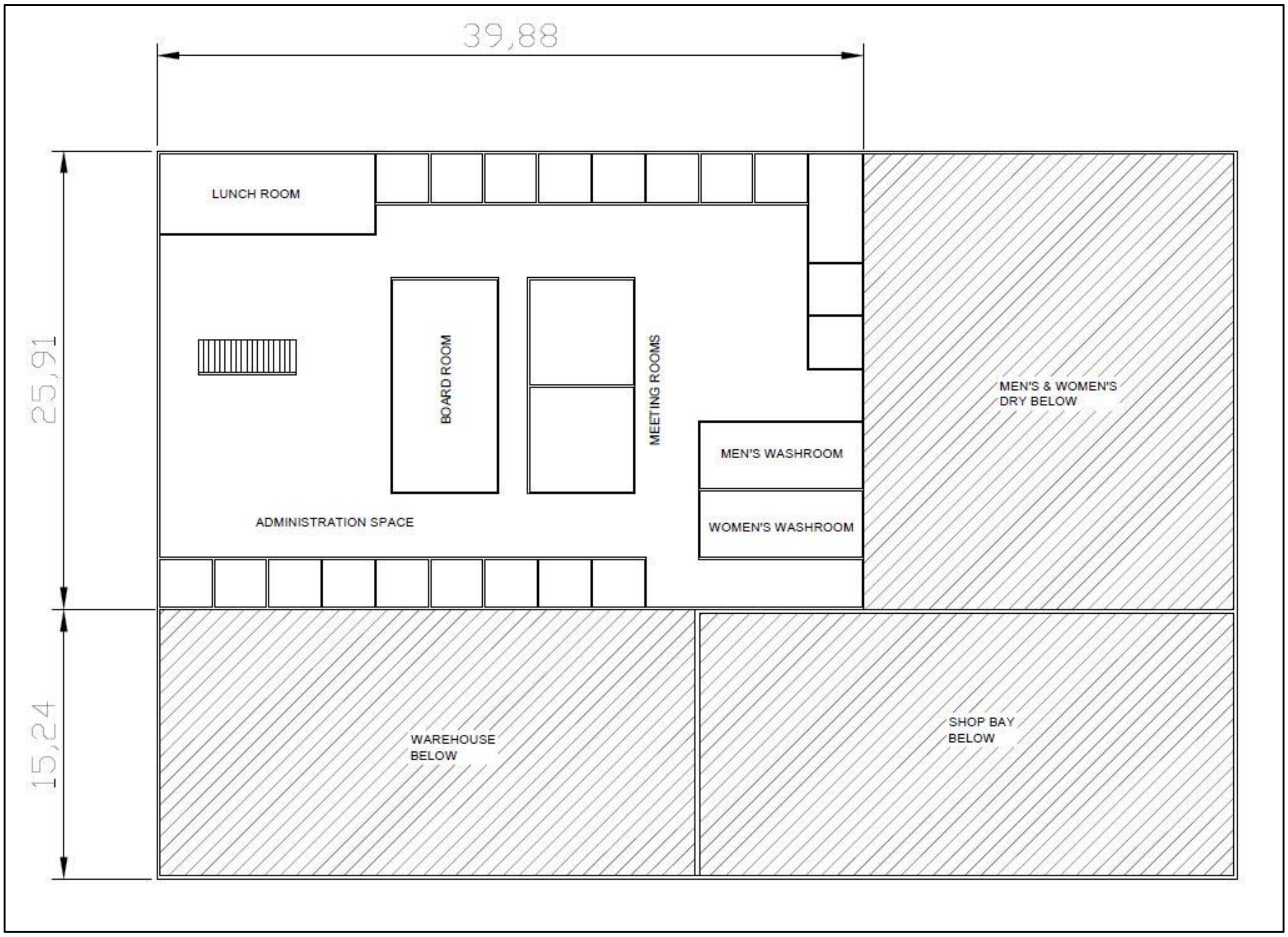


Figure 18-6. Proposed Layout for Second Floor of Operations Centre

18.11 Security Houses and Truck Scales

Access to the property will be controlled by a security gate that will be staffed 24 hours per day. The main security house near Phoenix will be equipped with an 80-tonne weigh scale that is hard-wired into the shack.

The buildings themselves are modular, pre-fabricated units that will be manufactured in Saskatoon and shipped to site for installation and commissioning. They will have a security gate, and appropriate power and communications capability.

18.12 Fuel Storage and Dispensing

A diesel and gasoline tank will be installed at the Phoenix site and a diesel tank will be placed at the Gryphon site. This fuel storage will be used to facilitate fueling both owner and contractor equipment. Each fuel tank will have a volume of 25,000 L and will feature double walled construction for fuel containment. The fuel tanks will be equipped with overfill prevention valves, bottom loading nozzles, and vents. Each tank will have a card lock fuel dispensing system and grounding reels. The fuel tank assemblies come equipped with full-length platform access and are mounted on I-beam skids for transport to site. Each tank will be placed on a concrete pad with an integrated sump for spills.

Fuel tanks can be filled by a conventional B-Train transport truck.

18.13 Propane Storage and Distribution

A propane storage and distribution system will be installed at both the Phoenix and Gryphon sites. The propane infrastructure will feature storage tanks, vaporizers, and a propane bottle fill station. The system capacity will be sufficient to supply 8 days of on-site storage and maximum consumption. Propane will be delivered to site on a weekly basis.

18.13.1 Phoenix Site Propane Infrastructure

The Phoenix site propane system will supply bottled propane to the camp kitchen and the incinerator. The system will feature a 30,000 uswg storage tank, a bottle fill pump, and a bottle weigh station, which will supply propane to remote buildings and mobile services. The propane bottle filling station will include a cylinder scale and a liquid propane filling pump.

18.13.2 Gryphon Site Propane Infrastructure

The Gryphon site propane system will supply propane to the mine air heaters, the headframe, and the mine office/mine rescue building. The estimated propane consumption rates for the Gryphon site are summarized in Table 18-6. The Gryphon site propane infrastructure will feature three 30,000 uswg storage tanks, two water bath propane vaporizers, and two distribution pumps. The propane storage area will be located a minimum of 100 ft from any building or entrance. The propane storage tanks will be protected by a perimeter of bollards.

Table 18-6. Gryphon Site Propane Consumption

Consumer	Rate (MBTU)
Mine Air Heaters	30.4
Headframe	1.0

18.14 Electrical Power

In accordance with preliminary studies by SaskPower, electrical service to the Wheeler River site will be provided via an approximate 5 km extension tap from the existing 138 kV overhead transmission line (defined as I3P) that runs along Saskatchewan Highway 914 linking the Key Lake and McArthur River sites. Optimization of the precise line route will be completed as the project proceeds.

The incoming 138 kV transmission line voltage will be stepped down via a single transformer to 25 kV for distribution throughout the site. The transformer will include an on-load tap changer to compensate for the estimated voltage regulation of the transmission line and will be protected with suitably rated primary and secondary protective devices.

A lineup of 25 kV class gas insulated type switchgear (GIS), housed in an e-house, will be used to distribute electrical power to the various site distribution transformers.

Redunadant 25 kV class utility style overhead pole lines will be used to interconnect the Phoenix and Gryphon sites, as well as other remote locations. 25 kV class power cables will be used to interconnect from the GIS equipment to the overhead lines and from the overhead lines to pad mounted distribution transformers.

600 V and 4.16 kV class distribution transformers will be installed in strategic locations to service the site loads. These transformers, interconnected with suitably rated switchgear and motor control equipment, will service the individual loads.

Smaller, suitably rated pole mounted transformers will be used to service small and/or remote loads along the 25 kV pole line routes.

The electrical systems (25 kV, 4.16 kV and 600 V) will each be resistance grounded to meet safety and mine electrical code requirements.

Based on the above basic design concepts, the electrical rate for the Wheeler River project will be in accordance with SaskPower's provincial "E24" service rate.

18.15 Back-up Electrical Power

Based on historical data provided by SaskPower, the outage rate of the existing I3P line is approximately 6 outages per year. To provide electrical service during times of utility outages, several standby diesel gensets will be installed in strategic locations to service the site emergency loads.

To avoid complexity, the standby units at the camp and operations centre will be operated as independent units (not interconnected nor synchronized with other gensets) to service the specific emergency loads local to each genset via a dedicated, suitably rated transfer switch.

At the Gryphon site, multiple synchronized and interconnected standby units will be installed to maintain the required mine ventilation, to evacuate personnel from underground using the Gryphon service hoist, and to maintain the underground water management systems.

18.16 Fresh Water Distribution

Fresh water will be used to feed the fire water system, the potable water treatment plant, and various operational requirements. There will be fresh water distribution systems at both the Gryphon mine and the Phoenix development. The fresh water system is sized to provide adequate water for both potable uses and process uses.

18.16.1 Phoenix Site Fresh Water Distribution

The Phoenix site fresh water distribution system is designed to provide fresh water to the fire water system, the potable water treatment system, the ISR plant, and the washbay. Fresh water is provided by means of a groundwater well and pump system, which pumps water directly to the 500m^3 fire water tank. A raw water distribution pump provides fresh water distribution to the potable water treatment plant, the wash bay, the ISR plant, and the temporary batch plant. The fresh water source pump will be a multistage well pump sized to fill the fire water tank within an 8-hour period. The fresh water distribution pump will be a conventional centrifugal pump and will have a back-up pump on standby. Estimated fresh water consumption rates are found in Table 18-7 below.

Table 18-7. Phoenix Site Estimated Fresh Water Consumption

Consumer	Flow Rate (L/day)
ISR Plant	2,000
Wash Bay	6,000
Potable WTP	30,000
Temporary Batch Plant	5,000

18.16.2 Phoenix Site Fire Water Distribution

The fire water system at the Phoenix site consists of a fresh water tank, two electric fire water pumps, and a back-up diesel fire water pump. The system is sized to provide a fire water supply capable of protecting 1,500 square feet of ordinary hazard classification at a flow rate of 0.3 gpm per square foot, plus an additional 6 hose stations at 50 gpm each. The resulting fire water system flowrate was estimated to be 750 gpm at 150 psi of total system head. NFPA requires that the fire water tank be designed to provide a minimum of 3 hours of water supply and be designed in such a way that it will be filled within an 8-hour period. The fire water tank is estimated to be approximately 500 m^3 in volume to meet these requirements. The fire water distribution piping will consist of a buried insulated piping loop that will supply pressurized fire water to the camp, the ISR plant, the operation centre, the potable water treatment plan, and the sewage water treatment plant.

The fire water tank will be constructed from carbon steel and will be fabricated at site due to its size. The tank will be insulated and potentially may require heating to ensure the water does not freeze in the winter. The tank will facilitate both fresh water distribution and fire water storage. The tank will be designed in such a way that the pump connection for the fresh water distribution will ensure that there is always a minimum volume of water in the tank.

Horizontal split case pumps were selected for fire water service. These pumps will be placed in a small pumphouse. A vertical multistage jockey pump, used to ensure that water is circulated, and a back-up diesel powered fire water pump, complete with a 250-gallon diesel fuel tank, an exhaust silencer, and a non-drip rim base, will also be installed.

18.16.3 Gryphon Site Fresh Water Distribution

The Gryphon site fresh water distribution system will be similar in design to the Phoenix fresh water system and will consist of a similar tank, distribution pump, and piping system. A groundwater well and pump will provide fresh groundwater to a fresh water storage tank, which will also serve as a fire water tank. The fresh water tank will provide fresh water to users, including the underground mine via the production shaft, using a conventional centrifugal fresh water distribution pump. A back-up fresh water pump will also be installed. The estimated consumption rate of fresh water by the Gryphon site is summarized in Table 18-8.

Table 18-8. Gryphon Site Estimated Water Consumption

Consumer	Flow Rate (L/day)
Headframe	2,000
Gryphon WTP	5,000
Batch Plant	5,000

18.16.4 Gryphon Site Fire Water Distribution

The Gryphon mine site will require a similar fire water tank and distribution system to that at Phoenix. It was assumed that the fire water requirements at the Gryphon site would be similar to those at the Phoenix site, and therefore the equipment was replicated. The Gryphon fire water system will provide fire water to the headframe, the Gryphon water treatment plant, and the office.

18.17 Potable Water Treatment and Distribution

Potable water will be generated on site by a modular potable water treatment plant. This plant will be located at the Phoenix site and will supply potable water to both the Phoenix and Gryphon sites. The potable water system consists of a treatment plant, a storage tank, and a bottle filling station. Water consumption was estimated to be 300 L per person per day. It was assumed that the maximum number of people working on site would be 100. Potable water will be piped to the camp, the operations centre, and the ISR plant to provide water for safety showers and eye wash stations. Other locations, such as the Gryphon site, the air terminal, and gate houses, will receive bottled potable water. The Gryphon water treatment plant will also have a large potable water tank located in an elevated location to provide potable water for safety showers and eye wash stations. This potable water tank will be filled using a tanker truck and monitored through instrumentation to ensure it is full at all times. Table 18-9 provides the potable water consumption by area.

Table 18-9. Potable Water Consumption by Area

Consumer	Flow Rate (L/day)
Camp	10,000
Operation Centre	10,000
Mill	5,000
Gryphon Site	2,000
Air Terminal	1,000
General Site	1,000

The potable water treatment plant will be a modular design and will come preinstalled in a 40 ft sea container. The filtration method will be either ultrafiltration or reverse osmosis with UV filtration. The plant will also feature an external potable water storage tank with a volume of approximately 2,000 L. The modular plant will come with all required HVAC and lighting and will be placed on a concrete pad. The potable water treatment plant will generate 1.4 m^3/hr of potable water and 33 m^3 of water per day. Raw water will be pumped to the potable water treatment plant from the fire water storage tank and fresh water distribution system.

18.18 Phoenix Waste Water Management

Waste water management at the Phoenix site will be handled as two separate waste water streams. Specifically, there will be domestic waste water production and contaminated waste water production. Domestic waste water will be treated at the Phoenix site sewage treatment plant, while the contaminated waste water will be treated at the Gryphon water treatment plant. Domestic waste was assumed to be generated at the rate of 300 L per person per day. Domestic waste sources located within a reasonable distance of the sewage treatment plant will be piped directly to the plant via force mains. In locations where it is not feasible to install force mains for sewage, septic tanks will be installed. Vacuum trucks will routinely empty septic tanks at the Gryphon mine and air terminal locations and deposit the sewage into the sewage treatment plant feed tank. Summary of domestic waste water generation is provided in Table 18-10.

Table 18-10. Domestic Waste Water Generation

Source	Flow Rate (L/day)
Camp	10,000
Gryphon Mine	2,000
Air Terminal	1,000
Mill Septic	5,000
Operation Centre Septic	2,000

18.18.1 Domestic Waste Water Treatment

Domestic waste water will be collected and treated in a central sewage treatment plant. The sewage treatment plant will be modular and will be installed in two sea containers, one being 40 ft and the other 20 ft. The containers will be insulated and heated. The sewage treatment plant will include a holding tank, associated filtration and treatment process equipment, and a sludge handling system. The sewage treatment plant will generate effluent suitable for discharge into the local water table. Treated effluent will first be discharged to surface testing ponds where the water quality will be checked to ensure it meets environmental quality standards. Reject solids from the treatment process will be collected, dewatered, and stored in the special waste facility on site.

18.18.2 Contaminated Waste Water

Contaminated waste water from the lab and the wash bay will be collected in a contaminated waste water sump tank. Contaminated water will then be hauled to either the Phoenix or Gryphon water treatment plants where it will be treated and discharged.

18.19 Gryphon Mine Site Water Treatment

The Gryphon water treatment plant will be designed to treat and discharge two primary inflows of water. Surface water will be collected from the ore and special waste piles through site grading to a surface water management pond, where it will be fed to the water treatment plant for processing. Water from the underground mine at Gryphon, or groundwater, will be collected in separate groundwater management ponds and fed to a second influent stream at the water treatment plant.

The Gryphon water treatment plant will have two sets of water treatment equipment, specific to each process stream. While surface water treatment will focus on metal precipitation, the underground water treatment will additionally focus on radium precipitation. Both water streams are considered contaminated with uranium.

Treated effluent will be discharged to holding ponds, which will be sized to hold effluent for a period of 24 hours for testing before discharge to the environment. The water will be treated to meet the regulatory requirements for effluent discharge in the region.

The design flow rates for the water treatment plant are summarized in Table 18-11.

Table 18-11. Water Production Rates for Gryphon Site

Water Source	Nominal Flow Rate	Design Flow Rate
Surface Water	5.4 m^3/h	12.1 m^3/h
Groundwater	249.0 m^3/h	298.9 m^3/h
Phoenix Lab and Wash Bay	4 m^3/d	4 m^3/d

Source: SRK – Wheeler River Project Water Management and Treatment Capacity Trade-Off Study, March 2018

18.19.1 Surface Water Treatment

The surface water that encounters the ore storage pile is considered contaminated, and therefore will be collected and treated before being released to the environment. The ore pile will be surrounded by liners and collection basins that will ensure any contaminated water generated at the ore pad will be redirected to the surface water management pond. Surface water from rainfall will be captured for treatment during the summer months and it is expected that surface water will require treatment from the months of May through October. In the spring, the surface water management pond will also collect the water from the melting snow that accumulated over the winter months. The surface water management pond is designed to accommodate the inflow of a 100 year 24-hour rain storm event. The size of the surface water management pond was calculated to be approximately 1,000 m^3 in volume to accommodate this event.

Contaminated water collected from the Phoenix operations centre laboratory and the wash bay sump will also be treated at the Gryphon water treatment plant. A vacuum truck will haul contaminated water from the Phoenix site and surrounding buildings to the Gryphon water treatment plant for processing.

The surface water treatment process will feature two treatment trains in series. Each treatment train will contain a reactor, a flocculator, and a lamella clarifier. The first train, metals coprecipitation, flows into the second train, metals pre-precipitation, which discharges into the surface water effluent well. Each train will precipitate metals from the effluent and clarify the process by adding reagents and flocculant. Once surface water effluent has been clarified, it is then fed to the influent stream of the groundwater treatment process (Section 18.19.2).

All solids and sludges precipitated from the surface water treatment process will be combined in a sludge tank, dewatered through a filter press system, and sent to a special waste storage facility.

18.19.2 Groundwater Treatment

Mine water is expected to have concentrations of metals, uranium, molybdenum, selenium, arsenic, and radium that will require treatment prior to discharge. Water inflows will be collected from the mine dewatering system, pumped to surface, and held in a groundwater management pond for treatment. It is anticipated that the Gryphon mine development will produce roughly 250 m^3/h of water inflow during normal operation and is designed to handle an emergency inflow of up to 1,650 m^3/h. The groundwater management pond is designed to accommodate the emergency inflow of water for a period of 28 days, with a volume of 810,000 m^3. Treated water from the surface water treatment process will be combined with the groundwater feed in the first reaction process.

The groundwater treatment process will be split into four process trains in parallel, thereby reducing the size of the equipment in each train to facilitate shipment of assembled components by road. Each process train will contain a reactor, a flocculator, a clarifier, and an effluent discharge well. Reagents will be added to the reaction circuit to precipitate radium and other metals. Solids are then separated by the clarifier. The clarifier underflow will report to a combined sludge tank, where it will then be processed with a filter press. Dewatered solids will then report to a special waste storage facility. Clarified water will be processed through a series of sand filters for further clarification.

Treated water will be stored in monitoring ponds for inspection and testing prior to discharge. It is anticipated that each monitoring pond will have a volume of approximately 7,500 m^3, which will provide 24 hours of storage for lab tests. Once the effluent has been tested and approved for discharge, it will be pumped through a heat traced and insulated pipeline along surface to the designated effluent discharge point between the Gryphon and Phoenix sites. If the effluent is not approved for discharge, it will be recirculated to the surface water management pond for additional treatment.

18.19.3 Water Treatment Plant Building

For the purpose of this PFS, the water treatment plant was proposed as a turn-key solution by Napier-Reid. The solution from Napier-Reid comes complete with all process equipment, the pre-engineered building, siding, roofing, man doors, overhead doors, lighting, and wall mounted HVAC with electric heat. The building will be insulated in a manor suitable for northern service and will be electrically heated. External piping, internal interconnecting piping, wiring, and the concrete foundations will be supplied and installed by the respective installation contractors on site. The water treatment equipment for both the surface water treatment and groundwater treatment processes will be housed inside the building, which is expected to measure approximately 32 m x 35 m and be 4.6 m in height. A control room, compressor room, and reagent storage will be included in the building design. The building will be located at the Gryphon mine site since, compared to the Phoenix site, the Gryphon site is expected to be net water positive.

18.19.4 Surface Discharge Location

In 2017, EcoMetrix Incorporated was retained to provide a preliminary evaluation (screening level assessment) of potential fresh water intake and treated mine water discharge locations. The objectives of the assessment were to utilize a staged approach to identify preferred intake and discharge locations from selected candidate sites. Based on proximity to the project site, 12 locations, located within two sub-watersheds that drain into the north end of Russell Lake, were identified as potential intake and/or discharge locations. The screening level assessment was divided into the following four stages:

Stage 1: A review of available information about traditional knowledge, and traditional, recreational, and industrial land use and evaluation of potential project effects.

Stage 2: An assessment of potential project effects on water quantity and quality. The effects of water withdrawal and discharge on water quantity and quality were predicted such that it was possible to eliminate locations from consideration that would be unsuitable for water withdrawal or discharge (i.e. ponds that would go dry, headwaters with low flows and little assimilative capacity).

Stage 3: Review of information about fish and fish habitat (i.e. fish spawning and number of fish species observed) from the aquatic baseline program for the Wheeler River project was reviewed to evaluate intake and discharge locations based on the potential for adverse effects on fish and fish habitat.

Stage 4: Environmental pathways modelling to further assess the interactions between surface water, sediment, and valued ecosystem components (VECs) when constituents of potential concern (COPCs) are released in treated mine water. The objectives of the modelling exercise were:

- To compare potential effects for an expected mine life of 15 years versus an extended mine life of 25 years; and
- To determine design criteria based on the assimilative capacity of each discharge location by deriving release limits protective of ecological health.

The screening level assessment reduced the number of environmentally safe treated effluent discharge sites down to five locations (LA-1, LA-5, LA-6, LA-7A, and Russel Lake), as identified in Figure 18-7. The five locations were presented during community meetings to gather additional information and community input, and to further refine candidate intake and discharge sites based on traditional land use.

In February of 2018, a working group of Denison Mines, SRK, and EcoMetrix professionals jointly evaluated the results of the staged screening level assessment and the community feedback. All options were comparatively assessed against a list of assessment criteria including safety, environment, community feedback, capital cost, operating cost, and risk. Weighting was applied to the criteria with safety and environment being the most critical, followed by community feedback. The least important parameter was operating cost.

The working group consensus ranked LA 5 and LA 6 as the most preferred options. LA 6 was selected as the discharge lake based on its proximity to the immediate project site, as well as being upstream of Russell Lake.

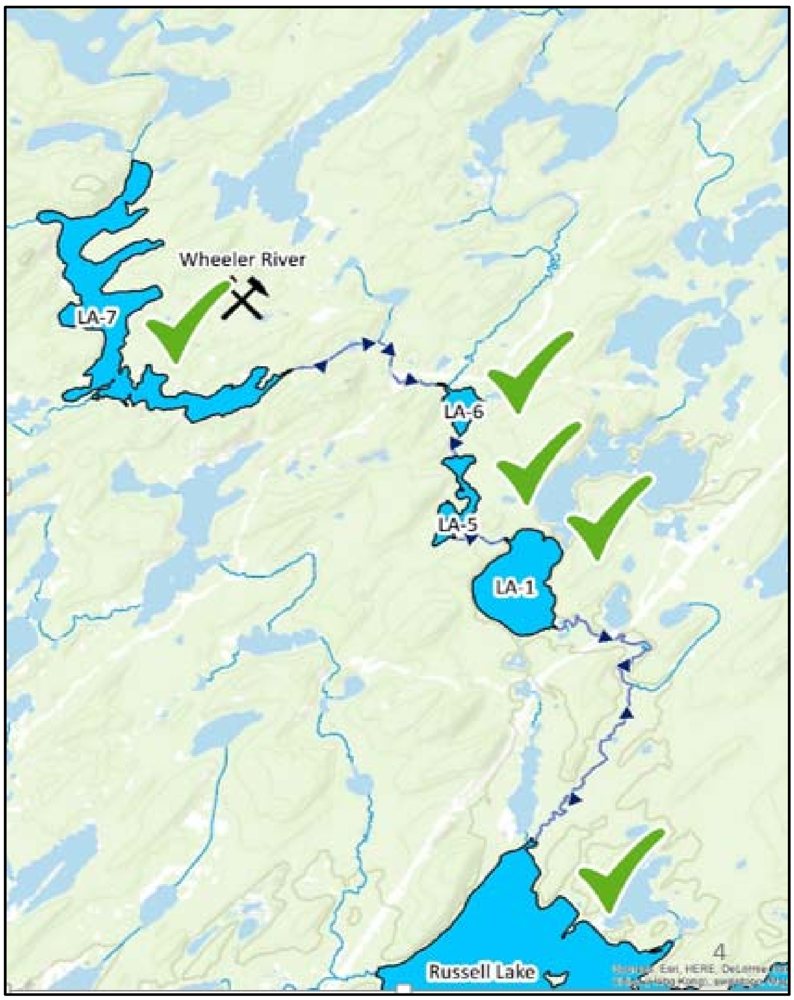


Figure 18-7. Refined Candidate Intake/Discharge Sites

18.20 Development Waste Rock Management

Waste rock generated from drilling at Phoenix will be stored on an unlined pad and will be available for use as necessary for road or concrete construction. Ore cuttings will be stored in IP2 containers until processed and residue from the ISR plant generated during processing will be stored in engineered, lined special waste storage facilities.

It is assumed that waste rock from Gryphon will be classified as ML/ARD. It is expected that any special waste, which is waste rock that contains a uranium content greater than 0.03% U_3O_8, will be stored underground. Life of mine waste will be stored in a waste rock pile near the headframe.

Clean waste rock from the shaft excavations is likely to be used for construction or road surfacing. However, a permanent stockpile has been assumed for the PFS.

Table 18-12 summarizes the material quantities.

Table 18-12. Phoenix and Gryphon Site Rock Storage Quantities

Deposit	Type	Volume
Gryphon	Clean waste rock	45,000 m^3
	ARD/ML waste	210,000 m^3
	Ore	3,000 m^3
Phoenix	Clean waste	7,100 m^3
	Special waste	3,200 m^3

18.21 Handling Infrastructure for Mined Materials

Final yellowcake products for Phoenix will be stored on site in the precipitation plant until transported off-site. Waste products are temporarily stored in lined storage areas until closure.

Gryphon ore and waste will be hoisted to surface, and the skips will discharge into a concrete storage chute adjacent to the headframe. A front-end loader will pick up the ore/waste, and stack waste on the waste pile and ore on the concrete ore pad adjacent to the headframe before transportation to the mill.

18.22 Concrete Batch Plant (Backfill)

A trade-off study was prepared to determine the optimum backfill for the Gryphon mine. Three types of backfills were reviewed (cemented rock fill (CRF), hydraulic fill, and pastefill), with various sources of material for each backfill type. There is insufficient waste rock in the Wheeler River area to utilize CRF only. Therefore, the local alluvial sand was sourced as the backfill material.

Sandfill can be discharged to underground in two forms - in paste form and in hydraulic slurry form. However, the paste form requires sand with significantly more fines (minus 20 microns) than the sand typically available in this area (which has been confirmed by testing to date). The sand will require additional classification to increase the fines content. Therefore, hydraulic sandfill is the only viable option, using 100% locally available sand. Preliminary testing has confirmed the sand is suitable for this backfill.

The sand is collected by loaders during the warmer months and transported by truck to a surface stockpile of approximately 50,000 tonnes.

The backfill plant will be equipped with the following:

- Classification circuit;
- Main sand load-in system;
- Mix tank;
- Binder system;
- Water addition;
- Discharge piping system from the mix tank to the underground backfill distribution system; and
- Dust collection system.

Backfill pour rate will be 50 dry tonnes/hour (dictated by distribution piping diameters). Therefore, the average backfill requirement is 8.8 hours per day. The water required at the backfill plant will be 31.4 m^3/hr (~138 USGPM).

Figure 18-8 illustrates the process flow for the proposed backfill facility.

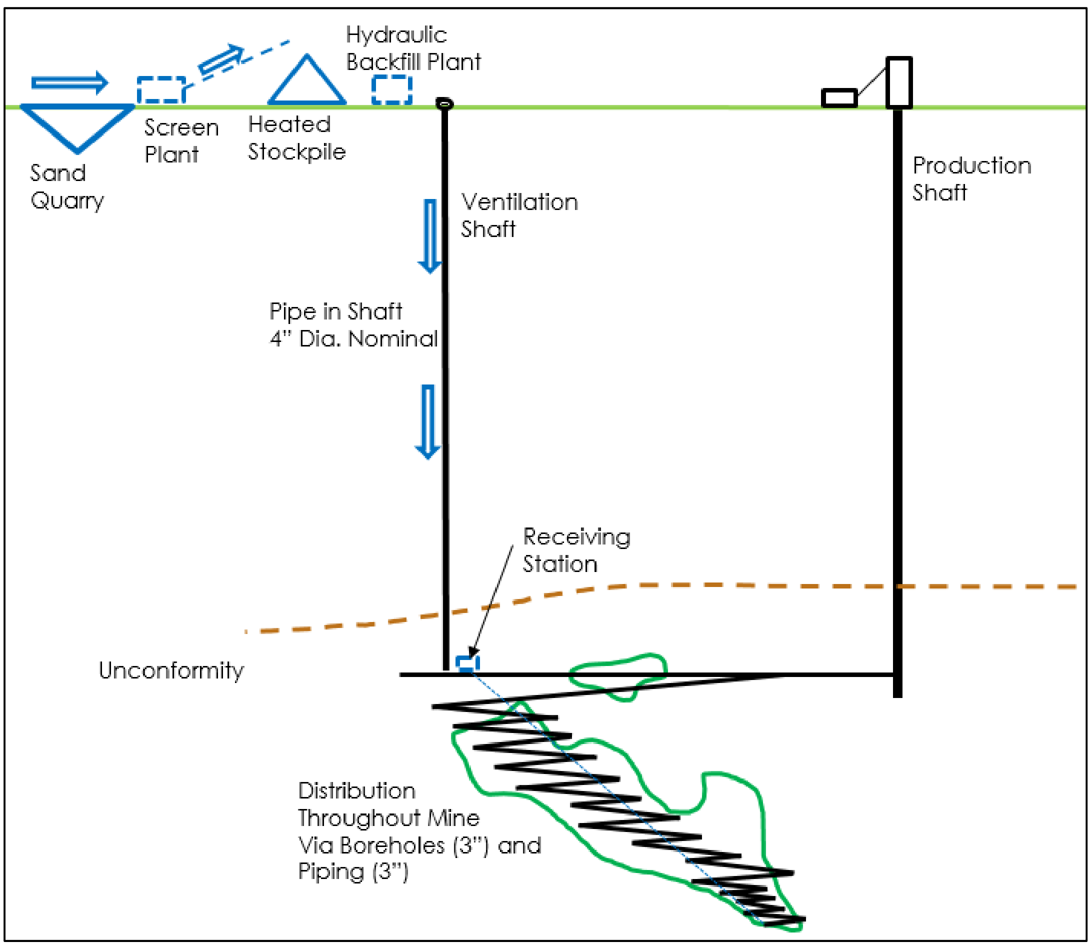


Figure 18-8. Backfill Plant Process Flow

18.23 Explosives Magazine

It is expected that weekly deliveries of explosives to the site will be immediately taken underground for storage and stored in the underground explosives magazine.

To account for weather or transportation disruptions, a surface explosives magazine has been accounted for. The magazine will be a heated and illuminated modular, pre-engineered building, and have security fencing and locks to prevent access.

19 Market Studies and Contracts

19.1 The Uranium Industry

In 2017, the uranium industry weathered yet another difficult and somewhat volatile year. An oversupplied spot market continued to put downward pressure on the spot price of U_3O_8, despite the announcement of various production curtailments from the world's largest uranium producers.

As noted in the UxC Q4 2017 Market Outlook, after reaching a 12-year low near USD$18.00 per pound U_3O_8 in December 2016, the spot price started 2017 at USD$20.25 per pound U_3O_8, traded north of USD$26.00 per pound U_3O_8 in the first quarter of the year, retreated back to the USD$20.00 per pound U_3O_8 level in the third quarter, then rallied in the fourth quarter to peak at USD$26.50 per pound U_3O_8 in early December 2017. After a volatile year, the spot price closed 2017 at USD$23.75 per pound U_3O_8, representing an increase of over 17% for the year.

Industry insiders have pointed to multiple reasons for the volatility in spot prices during 2017, including negative demand side stories from nuclear heavy-weight countries like the United States, France, and South Korea, continued disappointment with the rate of nuclear reactor restarts in Japan, the deferral of utility contracting activity, and an abundance of secondary supplies entering the market (including underfeeding from under-utilized enrichment plants). These negative stories were offset at various times during the year by high profile production curtailments announced by Cameco (see Cameco press release November 8, 2017) and National Atomic Company Kazatomprom (Kazatomprom press release January 10, 2017). The oversupplied spot market has also weighed on the long-term contract price of uranium, which has fallen 30% over the past two years, from a price of USD$44.00 per pound U_3O_8 at the beginning of 2016 to USD$31.00 per pound U_3O_8 at the end of 2017. With an estimated 75 million pounds U_3O_8 contracted during 2017 (approximately 30% of the annual contract volumes seen during the 2005-2012 contracting cycle), there have been few opportunities for the market to discover an appropriate long-term price for uranium.

Low prices and minimal contracting volumes seem illogical when juxtaposed to statistics from the U.S. Energy Information Administration and American Nuclear Society regarding the fact that, on a net basis, more new nuclear power capacity was added to the global electricity grid during 2015 and again in 2016 than in any other year over the last 25 years. With demand forecasts for uranium increasing steadily through 2030, meaningful new nuclear capacity is expected to come onto the grid while the uranium mining production pipeline has been stagnated by several years of low uranium prices. Uranium prices at current levels fail to incentivize the majority of undeveloped uranium projects towards construction, and, as a result, logic would suggest that prices should be on the rise. Underpinning that logic; however, is the assumption that growing demand in the future translates into increased buying today, and an oversupplied spot market, and historically low prices, will be fixed by opportunistic buying for long-term utility needs.

The UxC Q4 2017 Market Outlook noted volumes in the spot market during 2017 were sporadic, varying week to week with a total volume of approximately 44 million pounds U_3O_8 being traded during the year. With buyers staying on the sidelines, sellers have simply outnumbered buyers in the market and prices have battled downward pressure all year. This dynamic, combined with the reality of higher priced long-term contracts falling off in the not too distant future, led to the announcement of significant production curtailments in 2017. The most notable of these curtailments being Cameco's announcement regarding the shutdown of the McArthur River mine. The announced curtailment represents the removal of approximately 15 million pounds U_3O_8 from the market in 2018, and up to 18 million pounds U_3O_8 in future years.

Kazatomprom, the world's largest uranium producer, also declared that it would exercise restraint in 2017 and future years, having announced in early 2017 that it would cut production by 10% in 2017. Later in 2017, Kazatomprom also confirmed that it would constrain production levels for a further three years, through the end of 2020 (Kazatomprom press release December 4, 2017).

As a result of these and other production curtailments, the uranium market could swing to a deficit position in the near future, which would help to consume excess inventories that could otherwise leak into the market as secondary supplies. For a price recovery to be sustained, however, utility buying must resume, and contracting volumes must increase as utilities work towards securing approximately 1.2 billion pounds U_3O_8 in estimated uncovered uranium requirements for the period of 2018 to 2030. (UxC Q4 2017 Market Outlook)

With few economic sources of new supply able to advance through the project development pipeline in this market, and the potential for additional production curtailments as high-priced contracts at various high-cost operations are expected to drop off in the coming years, a significant utility contracting cycle is expected to lead to the realization that current uranium prices are below the level required to incentivize sufficient new sources of primary supply into the market.

19.1.1 Uranium Demand

The WNA reports that there are 454 nuclear reactors operable in 30 countries as of August 2018 (http://www.world-nuclear.org/information-library/facts-and-figures/world-nuclear-power-reactors-and-uranium-requireme.aspx). These reactors can generate 374 gigawatts of electricity and supply over 11% of the world's electrical requirements. As of August 2018, 55 nuclear reactors are under construction in 14 countries with the principal drivers of this expansion being China (15 reactors under construction), Russia (6), India (7), United Arab Emirates (4), and South Korea (4). Based on the most recent statistics from the WNA, there are a total of 207 reactors that are either under construction or planned around the world, and an additional 335 reactors that are proposed, with the potential to be operating by 2030.

19.1.2 Primary Uranium Supply

According to UxC's Q4 Outlook, uranium production for 2017 was estimated to decrease by nearly 7% year over year from 162.0 million pounds U_3O_8 in 2016 to 151.1 million pounds U_3O_8 in 2017. Production in 2018 is expected to decrease even further, with the Q1 2018 Outlook projecting 2018 production will drop a further 6.7% (from 2017 estimates) to only 141.1 million pounds U_3O_8. Production from Canada, Kazakhstan, Australia, Africa, and the United States all declined in 2017, while production from Russia remained essentially flat. Production in Canada decreased by nearly 6% or 2.1 million pounds U_3O_8. Cigar Lake production is expected to remain constant at 18 million pounds of U_3O_8 per year through 2025. McArthur River has been modeled by UxC to produce 1 million pounds of U_3O_8 in 2018 (owing to the announced 10-month shut-down). Canada remains the second largest producing nation with approximately 23% of the world's production from 2017 coming from within Canada. Kazakhstan continues to be the world's largest producer of uranium, representing approximately 40% of production in 2017.

In its Q1 2018 Outlook, UxC estimates that existing mine production, plus new planned and potential mine production, will increase primary uranium supply to 161.7 million pounds U_3O_8 by 2020, before declining to only 115.9 million pounds U_3O_8 by 2030. At its height in 2020, the projected production levels include the resumption of mining at McArthur River (estimated at 18.7 million pounds U_3O_8 per year) and represents a total increase of only 7% from estimated 2017 production levels. This is in contrast to the dramatic increases in uranium demand outlined above. In past years, UxC projected that Kazakhstan was expected to continue to be one of the principal drivers for the increases in primary mine production. In the Q1 2018 Outlook, the main drivers are now limited to the resumption of mining at McArthur River and the ramp up of production at the Husab mine in Namibia. For other projects to move forward to increase UxC's production forecasts, uranium prices will need to increase appreciably to support their higher cost production profiles and the significant capital expenditures that will be required.

19.1.3 Secondary Uranium Supply

In the Q1 2018 Outlook, primary mine production is estimated to supply approximately 73% of estimated 2018 base case demand (compared to approximately 80% in 2017). The balance of demand is expected to be supplied from secondary sources such as commercial inventories, reprocessing of spent fuel, sales by uranium enrichers, and inventories held by governments, in particular the U.S. Department of Energy. In years prior to 2017, primary supplies have normally made up 85% or more of annual demand.

Excess commercial inventories, which were once one of the major sources of secondary supplies during the period from the early 1970s to the early 2000s, have largely been consumed; however, as a result of the shutdown of the German nuclear program and the continued shutdown of the majority of the Japanese nuclear fleet, commercial inventories could become a more significant factor. A large source of secondary supplies continues to be government inventories, particularly in the U.S. and Russia. The disposition of these inventories may have a market impact over the next 10 to 20 years, although, the rate and timing of this material entering the market is uncertain. Secondary supplies remain a complexity of the uranium market. The Q1 2018 Outlook forecasts that 49.7 million pounds U_3O_8 will enter the market from secondary supplies in 2018, leaving a shortfall of 3.3 million pounds U_3O_8 for supplies to match the base case demand scenario for 2018.

Looking ahead, UxC expects that secondary sources of supply will fall from estimated 2018 levels to 23 million pounds U_3O_8 per year by 2030.

19.1.4 Uranium Prices

Nuclear utilities purchase uranium primarily through long-term contracts. These contracts usually provide for deliveries to begin two to four years after they are signed and provide for delivery from four to ten years thereafter. In awarding medium and long-term contracts, electric utilities consider the producer's uranium reserves, record of performance, and production cost profile, in addition to the commercial terms offered. Prices are established by a number of methods, including base prices adjusted by inflation indices, reference prices (generally spot price indicators, but also long-term reference prices), and annual price negotiations. Contracts may also contain annual volume flexibility, floor prices, ceiling prices, and other negotiated provisions. Under these contracts, the actual price mechanisms are usually confidential.

The long-term demand that actually enters the market is affected in a large part by utilities' uncovered requirements. UxC estimates, in the Q1 2018 Outlook, that uncovered demand is only 4.4 million pounds U_3O_8 in 2018. Uncovered demand, however, is projected by UxC to increase significantly over the period of 2018 to 2020, such that up to 27.9 million pounds remains uncovered for 2020. Annual uncovered demand rises rapidly for years after 2020, to 108.6 million pounds U_3O_8 for 2025 and over 138.2 million pounds U_3O_8 for 2030 (representing roughly 80% of total base case demand in those years). Taken together, nearly 1.1 billion pounds U_3O_8 remain uncovered between 2018 and 2030. At 138.2 million pounds, uncovered demand in 2030 is over 35 million pounds U_3O_8 more than total production expected from existing uranium mines for the same year. In order to address the rising portion of demand that is uncovered, utilities will have to return to the market and enter into long-term contracts. From 2006 to 2010, on average, 39 million pounds U_3O_8 equivalent were purchased on the spot market per year and roughly 200 million pounds U_3O_8 equivalent were contracted in the long-term market each year. By comparison, in 2017, 43.5 million pounds U_3O_8 equivalent were purchased on the spot market, and approximately 73 million pounds U_3O_8 equivalent were contracted in the long-term market. With low contract volumes in recent years and increasing uncovered requirements, we expect that long-term contracting activity will have to increase in the near future as utilities look to secure supply and move U_3O_8 through the nuclear fuel cycle in order to fuel the world's growing fleet of nuclear reactors.

The long-term price is published on a monthly basis and began the year at USD$30.00 per pound U_3O_8. On low volumes, as noted above, the long-term price increased to USD$31.00 per pound U_3O_8 by the end of the year.

19.1.5 Competition

The uranium industry is small compared to other commodity industries, and in particular other energy commodity industries. Uranium demand is international in scope, but supply is characterized by a relatively small number of companies operating in only a few countries. Production, in general, is concentrated amongst a small number of producers and is also geographically concentrated with approximately 70% of the world's production in 2017 coming from only three countries: Kazakhstan, Canada, and Australia.

Competition is somewhat different amongst exploration and development companies focused on the discovery or development of a uranium deposit. Exploration for uranium is being carried out on various continents, but expenditures by public companies have been generally concentrated in recent years in Canada, Africa, and Australia. In Canada, exploration has focused on the Athabasca Basin region in northern Saskatchewan. Explorers have been drawn to the Athabasca Basin region by the high-grade uranium deposits that have produced some of the most successful uranium mines operating in the world today. Within the Athabasca Basin region, exploration is generally divided between activity that is occurring in the eastern portion of the Basin and the western portion of the Basin. The eastern Basin is a district that is defined by rich infrastructure associated with the existence of several operating uranium mines and uranium processing facilities. Infrastructure includes access to the provincial power grid and a network of provincial all-weather highways. By comparison, in the western Basin, there are no operating uranium mines or processing facilities and access to the provincial power grid is not currently available. Several uranium discoveries have been made in the Athabasca Basin region in recent years, and competition for capital can be intense.

19.2 Contracts

Denison has historically sold its uranium under a combination of long-term contracts and spot market sales. The long-term contracts had a variety of pricing mechanisms, including fixed prices, base prices adjusted by inflation indices, and/or spot price or long-term contract reference prices. The company currently has no long-term contracts in place.

20 Environmental Studies, Permitting, and Social or Community Impact

20.1 General

Uranium has been mined in Saskatchewan since the mid-1900s. The development of new deposits in the late 1970s (Cluff Lake uranium mine) saw an increase in public interest/concern with uranium mining in the province. This public interest/concern has been present with the onset of each new uranium development in the province since the Cluff Lake mine. As a result, governments, both federal and provincial, and industry have continued to increase their attention to addressing social and environmental considerations associated with uranium mining in Saskatchewan. There are a number of uranium mine and/or milling operations in the Athabasca Basin in northern Saskatchewan. The economy and employment statistics of northern Saskatchewan are heavily influenced by uranium mining and milling.

20.2 Denison's Approach to Sustainable Mining at Wheeler River

Denison Mines recognizes the importance of a healthy environment for future generations and is committed to integrating its economic activities with environmental integrity and the principals of environmental sustainability.

In order to ensure the environmental sustainability of the Wheeler River project, Denison plans to adopt sustainable mining practices with the aim of reducing the impact of the Wheeler River project on the surrounding environment, reducing cradle to grave costs, and paving the road to the Company's ultimate goal of returning the site to pre-mining conditions. Through the use of available technology and best practices, the Phoenix operation is expected to be one of the most environmentally friendly mines in the world. The project will utilize innovative strategies to reduce water and energy consumption, minimize land disturbance and waste production, prevent soil, water, and air pollution; and plan for successful mine closure and reclamation.

20.2.1 Land Disturbance

Traditional mining practices of underground or open pit mines require relatively large areas of surface disturbance for construction of the mining facilities and storage of waste material. For the Phoenix deposit, the total new surface disturbance is ~14 hectares and is limited to roads on site, various surface facilities, and the construction of a 150 m x 100 m precipitation plant. At Gryphon total land disturbance is estimated at ~19 hectares. Other shared infrastructure like the main access road and airstrip total an additional 15 hectares.

20.2.2 Water

Extensive quantities of fresh water are required for traditional mining applications, from which complex water balances are frequently derived. Such mining applications often require aquifer dewatering, surface water diversions, as well as water storage, treatment, and final treated water discharge to the surrounding environment.

For the Phoenix operation, Denison intends to recycle process water to the greatest extent possible contributing to a reduced demand for fresh water supply. The proposed recycling process design incorporates a closed-loop system within which only limited make-up water is required to supplement the process. Make-up water will be preferentially drawn from contaminated surface or facilities run-off or otherwise from a deep water well eliminating impact on surface water bodies. The make-up water will be mixed in with recycled mining solution water and used in the mining process. It is expected that the processing circuits will require 3.4 m^3/h of make-up water. In the event that a 10% bleed is required to maintain contaminant levels, the ISR process operations will require a maximum of 6.2 m^3/h of make-up water.

Due to the recycling of solution within the ISR plant and the potential use of a crystallizer for handling the BLS bleed stream, treatment of process solution water is not expected. When combined, other water sources, such as wash bay/process sumps and contact area site run-off, will generate water treatment plant effluent for total discharge volume of 3,000 m^3/a. Denison has planned to re-inject the treated effluent at the same horizon as the make-up water is drawn from to eliminate any regional pressure variations. This will eliminate the need for any discharge to surface water bodies.

At Gryphon, no mill or tailings management facility is planned. As a result, significantly lower volumes of water will be produced, with the majority of these originating from the underground mine. In the mine, a water impermeable shaft liner and mine openings that are well below the unconformity are expected to minimize water inflow. As a result, moderate levels of treated effluent are expected. All water from underground, as well as contaminated surface run-off, will be collected and treated at the conventional water treatment plant. Treated effluent will be pumped from the Gryphon site approximately 3 km to the preferred discharge point west of the Phoenix site. After extensive site characterization and consultation with communities, this discharge point was selected as the preferred discharge point due to technical, environmental, and social reasons.

20.2.3 Energy

Through innovation and planning, Wheeler River operations will draw its primary power supply from the provincial electrical grid to the greatest extent possible. Preliminary studies by SaskPower have indicated that electrical service to the Wheeler River site can be provided via an approximate 5 km extension tap from the existing 138 kV overhead transmission line that runs along Saskatchewan Highway 914. Diesel generators will be installed on site and only used to provide emergency site power for health, safety, and environmental demands in the event of power grid outages.

In an effort to further improve its energy efficiency, Denison Mines is assessing the state of technology for battery-powered light vehicles and mobile equipment for its site requirements. Similarly, Denison has determined it is viable to utilize an AC powered duel rotary drill for ISR wellfield development rather than a traditional diesel-powered unit. Additional site infrastructure has been designed to draw power from the provincial power grid, including camp buildings, operations buildings, the ISR precipitation plant, and the freeze plant.

20.2.4 Fuel

Since the site's primary power supply will be from the provincial electrical grid, Wheeler River fuel consumption will be limited to emergency power supply, auxiliary vehicles (i.e. ATVs and snowmobiles), miscellaneous equipment (i.e. portable pumps), and freight and personnel transportation to site. This will reduce project fuel consumption and the project's overall reliance on fossil fuels.

20.2.5 Tailings

One of the major benefits for ISR operations is the elimination of tailings. Every other conventional mining method produces tailings that require special handling and storage to minimize environmental damage. In the long term, these tailings storage facilities often require annual maintenance programs that can be required indefinitely. For the Phoenix deposit, tailings are not produced during the process, as only the uranium (and minor amounts of impurities) are removed from the ground. The uranium is sold as a finished product, and the minor amounts of impurities are temporarily stored on surface until permanent disposal in the underground stopes at Gryphon is possible.

20.2.6 Waste Rock

Traditional mining methods generate acid rock drainage (ARD), which is produced when sulphide minerals in waste rock and ore at mine sites are exposed to air and water. ARD can pollute surface and groundwater with acidity and dissolved metals, which can adversely affect aquatic organisms and water users downstream if it is not treated.

During ISR operations, the host uranium-bearing ore is not removed from its geological deposit, thus remaining in situ. A leaching agent is injected via injection wells into the ore deposit initiating the leaching process, and the pregnant solution is subsequently returned to the surface via recovery wells. By utilizing ISR to mine the Phoenix uranium deposit, with the exception of the small volume of waste rock generated from the development of the ISR recovery and injection wells, Denison proposes to eliminate the generation of waste rock, which would otherwise require handling and storage, and consequently generate the potential for ARD.

By eliminating the generation of waste rock (and the potential for ARD), ISR also eliminates the potential for metal leaching, which would likely take place where waste rock composition demonstrates concentrations of leachable metals.

In the Gryphon mine, there will be waste rock generated that will be classified as special waste and/or potentially acid generating. During production, this rock will be used as underground backfill when available. Prior to production and the need for backfill, some rock with this characterization will be stored on surface on a lined pad. At closure, the portion of this rock that has not been backfilled in the underground workings will be properly graded and covered in a manner to eliminate infiltration and subsequent contact with water produced through precipitation and runoff.

20.2.7 Fugitive Dust Emissions

Air emissions from the Wheeler River project will be comprised primarily of fugitive sources, such as vehicular transport to, from, and around the project site, as well as from transport truck travel on gravel roads. Fugitive emissions traditionally generated from stockpiling and loading activities, as well as wind entrainment from the stockpiles themselves, will be eliminated from the project scope, as these activities are not required for ISR operations.

20.2.8 Site Closure and Rehabilitation

Mining is a relatively temporary activity, and mine sites have finite operating lives, which are determined by the size and quality of the ore deposit being mined.

Based on head grade, estimated resource, flow rates, and closure requirements for the Phoenix deposit, it is estimated that production will occur across the Phoenix deposit over a period of approximately 11 years, after which, site reclamation and closure activities (which are estimated to last approximately five years) will commence, with the aim to restore the groundwater and ISR footprint to its pre-mining condition. Due to the nature of the proposed Phoenix ISR activities (i.e. minimal impact to the surrounding environment and small footprint), when an assessment was completed to identify potential long-term environmental impacts or shortfalls associated with reclamation requirements, scenarios that could not be mitigated and/or prevented were not identified. This provided Denison with confidence that a successful site closure and reclamation plan that meets federal and provincial standards can be achieved.

At the Gryphon site, conventional closure activities will commence following completion of mining. With most of the disturbance occurring deep underground, returning the site to its pre-mining state is expected to be achievable. Removal of surface infrastructure, capping of shafts, and re-vegetation of the roads and surface disturbance will be completed. The only noticeable change on surface will be the surface waste rock piles, which will be contoured to conform to the natural topography and re-vegetated. It is not expected that any long-term or institutional monitoring of the facilities will be required and a full return to pre-mining conditions is expected.

20.3 Regulatory Setting

In Saskatchewan, the environmental assessment and permitting framework for the development of a mining project consists of a two-tiered system. The first tier consists of an environmental assessment (EA) phase involving departments from both the federal and provincial governments. Following a successful EA, the project would proceed to the second tier of regulation, which consists of a construction and operating licensing/permitting phase, again involving both federal and provincial government departments and agencies. The project is then regulated through all phases (construction, operation, closure, and post closure) by the same federal and provincial departments and agencies.

Unique to uranium, which is classified as a strategic mineral under federal legislation, the Canadian Nuclear Safety Commission (CNSC), a commission federally established in 2000 reporting to the federal cabinet through the Minister of Natural Resources Canada, regulates the use of nuclear energy and materials to protect the health, safety, and security of Canadians and the environment, and implements Canada's international commitments on the peaceful use of nuclear energy.

20.4 Environmental Assessment

The assessment of a proposed uranium project in Saskatchewan involves both a provincial and federal assessment. In Saskatchewan, the assessment of a project with joint federal and provincial jurisdiction is coordinated through established protocols in order to align with the "one project-one assessment" model for the proponent and the public without compromising any statutory requirements of the legislation of either jurisdiction.

20.4.1 Provincial Requirements

In the province of Saskatchewan, the Environmental Assessment Act is administered by the Ministry of Environment (MOE). The level of assessment for mining projects is dependent on the specific characteristics of each individual project.

In Saskatchewan, the proponent of a project, that is considered to be a "development" pursuant to Section 2(d) of the Environmental Assessment Act, is required to conduct an environmental impact assessment (EIA) of the proposed project and prepare and submit an environmental impact statement (EIS) to the Minister of Environment.

Section 2(d) of the Environmental Assessment Act reads:

> *…"development" means any project, operation or activity or any alteration or expansion of any project, operation or activity which is likely to:*
>
> - *Have an effect on any unique, rare or endangered feature of the environment*
> - *Substantially utilize any provincial resource and in so doing pre-empt the use, or potential use, of that resource for any other purpose*

- *Cause the emission of any pollutants or create by-products, residual or waste products which require handling and disposal in a manner that is not regulated by any other Act or regulation*
- *Cause widespread public concern because of potential environmental changes*
- *Involve a new technology that is concerned with resource utilization and that may induce significant environmental change*
- *Have a significant impact on the environment or necessitate a further development which is likely to have a significant impact on the environment (Sask. Env. Act, 2002)*

The Wheeler River project, as it is currently defined, meets the province's definition of a "development" and will therefore be required to conduct a provincial EIA.

20.4.2 Federal Requirements

CEAA 2012

The Canadian Environmental Assessment Act (CEAA) was amended in the spring of 2012 and the Regulations Designating Physical Activities (2012) were established to clarify when a federal EA is required and define what federal agency is required to be the "responsible authority" for the conduct of the EA.

Under CEAA 2012, an EA focuses on potential adverse environmental effects that are within federal jurisdiction, including:

- Fish and fish habitat;
- Other aquatic species;
- Migratory birds;
- Federal lands;
- Effects that cross provincial or international boundaries;
- Effects that impact on aboriginal peoples, such as their use of lands and resources for traditional purposes; and
- Changes to the environment that are directly linked to or necessarily incidental to any federal decisions about a project.

There are two main methods of "triggering" a federal EA under CEAA (2012):

1. A project will require an EA if the project is described in the Regulations Designating Physical Activities.
2. Section 14(2) of CEAA (2012) allows the Minister of Environment to designate (by order) a physical activity that is not prescribed by regulation if, in the Minister's opinion, either the carrying out of that physical activity may cause adverse environmental effects or public concerns related to those effects may warrant the designation.

Because the Wheeler River project is a uranium project, the CNSC is designated as the "responsible authority" under Section 15 of CEAA (2012) and carries full authority under CEAA (2012) to complete the screening of the proposed project and subsequent environmental assessment should it be determined during the screening process that one is required, on the basis of potential adverse environmental effects to one or more of the federal jurisdictions discussed above.

To initiate the EA process under CEAA (2012), the proponent is responsible to submit a project description to the responsible authority for screening. If it is determined an EA is required, there are two types of EAs that can be conducted under CEAA (2012):

1. An EA by a responsible authority (similar to a comprehensive study EA under CEAA, 1992).
2. An EA by a review panel.

The Wheeler River project is defined as a "designated project" under CEAA (2012) and will need to be screened under this legislation. A self-screening of the proposed project suggests it will require a federal EA to proceed. The CNSC will be the responsible authority for conducting this assessment.

In addition to the legislated federal requirements defining the need for an environmental assessment, the federal government introduced the Major Projects Management Office (MPMO) in 2007. The MPMO role is to provide a management and coordinating role for major resource development projects in Canada. The authority and mandate of the office is provided through a committee comprised of deputy ministers from federal departments typically identified as "responsible authorities" in the conduct of a federal environmental assessment. The MPMO has no legislative authority. The MPMO would self-determine their level of involvement in the assessment as part of the original screening process. Given the promulgation of CEAA (2012) and the expected manageability of the environmental risks associated with the Wheeler River project as it is currently defined, SRK believes the MPMO will determine the assessment of this proposed project can be completed without significant involvement from their office. Other federal legislation that will need to be considered throughout the EA and licensing phase of this project includes:

- Fisheries Act.
- Species at Risk Act.
- Migratory Birds Convention Act.
- Navigable Waters Protection Act.
- Canada Water Act.
- Canada Labour Code.
- Transportation of Dangerous Goods Act.

Impact Assessment Agency

In 2016, the Canadian government announced it was undertaking a review of environmental and regulatory processes. An expert panel was established and spent several months obtaining feedback from the public, indigenous groups, and other governments and interest groups. The panel's report was submitted to the Minister of Environment in April 2017.

A fundamental reform suggested in the report was a change in terminology from an environmental assessment to an impact assessment. In addition, the report suggested changes to how the federal regulatory regime addresses cumulative effects, early engagement and planning (with a particular focus on Indigenous peoples), greater transparency, and public participation, as well as a shift from the assessment focusing on "significant environmental effects" to one focusing on whether the project is in "the public interest".

In February 2018, under Bill C-69, the Impact Assessment Act (IAA) capturing all of the above recommendations was tabled in the House of Commons. The current government schedule is on track for the IAA to come into force by June 2019, at which time this act will replace CEAA (2012).

The new act represents a number of changes to the federal assessment process and scope of the assessment. In general, changes to the act will require the assessment consider gender analysis, a need for projects to meet Canada's climate change commitments and contribution to sustainability, an increased emphasis on cooperation with other jurisdictions, and increased recognition of Indigenous rights throughout the process.

From a scheduling and cost perspective, the IAA proposes more of the process being covered by set timelines, with respect to the government's role. Uranium projects will automatically be assessed by an independent panel. The government timeline for the panel process, as it is currently understood, consists of an initial 180-day early planning phase, 600 days to complete their review, followed by a 90-day period to render their final decision. Similar to CEAA, the IAA still allows the government to "stop the clock" during any phase should they require additional information from the proponent. It is estimated the new process will require a minimum of 4 years to complete an assessment.

The IAA process will also be structured as a cost recovery process, meaning completing an assessment under the new act will represent increased costs to the overall assessment process to that expected under CEAA (2012).

Current understanding of the transition phase between CEAA (2012) and the IAA suggests if a uranium project is initiated under CEAA (2012) through the acceptance of a "complete" Project Description prior to the IAA coming into force, the project will be assessed under the CEAA (2012) regulatory regime.

20.5 Licensing and Permitting

In the event that environmental assessment approvals by both the provincial and federal governments are granted, the project will be allowed to proceed to the second tier of environmental approvals. This requires the proponent to obtain a variety of approvals/permits/authorizations again from both levels of government.

The federal (CNSC) licensing process requires the submission of detailed engineering design packages as well as detailed management plans for all facets of the operation as part of their licensing process. The first licence to be applied for from the CNSC would be a licence to prepare a site and to construct. The CNSC licence application can be developed by the proponent and submitted for review during the EA process. The licensing decision would not be made until after the EA decision is provided. Other licenses that will be required from the CNSC in the life of the mine and mill would be a license to operate, decommission, and abandon.

The proponent would need provincial approval through the submission of various applications to Construct a Pollutant Control Facility, followed by an Approval to Operate a Pollutant Control Facility, which would also outline the proponent's various monitoring and reporting requirements throughout the life span of the approval. Provincial approvals are renewed approximately every five years and also dictate the schedule the proponent must follow with respect to updating the project's decommissioning and reclamation plan and associated financial assurance obligations for the project.

20.6 Assessment Schedule and Estimated Costs

Based on a review of the CEAA (2012) and using previous assessments of similar projects for comparison, it is estimated that the environmental assessment of the Wheeler River project will require approximately 24 to 36 months from the submission of the project description to the receipt of the environmental assessment approvals to proceed with the project. Amendments to CEAA (2012) and the Nuclear Safety and Control Act (NSCA) have been made to define timelines within an

EA that must be followed by the responsible authority. The CNSC, as the responsible authority, is obligated to contain those portions of the EA process that are controlled solely by them to a 24-month timeline. However, this timeline starts and stops while the CNSC waits for the proponent's input and/or response to deficiencies.

It is estimated that gathering the necessary data, drafting the environmental impact assessment, and completing the EA process will cost approximately $3 million. In addition, the CNSC is a cost recovery regulator agency, which means once a federal EA is initiated, all involvement of the CNSC and its staff is billed back to the proponent in accordance with their Cost Recovery Fees Regulations. Communications with representatives of the CNSC indicate that the cost recovery of the assessment process can be expected to range between $2 million and $4 million. The range of these costs will be a function of the type of environmental assessment required and its complexity. For inclusion in the economic analysis of this study, it is suggested that $3 million be carried as a reasonable expected cost to complete the environmental assessment if it is completed under CEAA (2012).

20.7 Environmental Considerations

The main environmental considerations associated with this project are centred on the management of its various waste streams and the preferred mining method for the Phoenix deposit. The dominant and/or potentially more problematic of these waste streams are water and waste rock. Run of mine ore from the Gryphon deposit will be transported via truck on provincial highways to the McClean Lake mill (Denison ownership - 22.5%) for processing under a toll milling arrangement. The management of the tailings and associated effluent generated through the process will be managed in accordance with the terms and conditions of the McClean Lake mill's existing federal and provincial licenses and approvals. It is anticipated the characteristics of the tailings or their potential contaminant source terms will need to be evaluated in order to demonstrate the processing of Gryphon ore and management of the associated waste streams can be done in an environmentally acceptable manner.

The preferred mining method proposed for the Phoenix deposit is the use of in situ recovery (ISR). If implemented, processing of the Phoenix deposit will be completed in situ, therefore, the solid waste (waste rock) streams associated with this mining method will be limited to drill cuttings produced through the establishment of the injection and recovery wells located in the wellfield. Processing of the uranium enriched solution brought to surface would be done by direct precipitation at or near the wellfield. This process will require a water treatment plant in order to remove iron and radium from the uranium enriched water prior to precipitating the uranium. In addition, a water treatment process will be required to complete wellfield restoration following the in situ recovery process.

Mining uranium through the use of in situ recovery currently accounts for greater than 50% of the world's uranium production. The application of this well-known mining method to extract the Phoenix deposit would be the first time this mining method is used to mine uranium in a Canadian jurisdiction. Discussions with representatives from both the federal and provincial regulatory regimes as well as the project's northern, aboriginal, and First Nations stakeholder groups did not identify any fatal flaws with respect to pursuing this form of mining. In fact, the discussions generally centred around the environmental benefits of in situ recovery versus conventional mining methods.

Based on the existing understanding of the proposed project, there are no environmental fatal flaws identified and there is no reason to assume the project could not complete an environmental assessment which could be acceptable to both regulatory regimes and the project's stakeholders.

20.8 Environmental Baseline Studies

During the spring of 2016, following the publication of the 2016 Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada, Denison initiated a comprehensive environmental baseline collection program. The previous environmental work completed in 2012 (hydrologic and aquatic investigations) and 2014 and 2015 (geotechnical and hydrogeological investigations) will be combined with the results of the 2016 and 2017 studies in order to support the completion of a federal and provincial environmental assessment for the project.

The data collected from the baseline studies will be used to analyze and quantify the relevant environmental parameters for the local and regional study areas to provide a record of the environmental conditions prior to commencing project activities. This data will also be fundamental to the project's environmental assessment and to subsequent regulatory permitting and licencing processes.

The following sections summarize the environmental baseline work completed to date, much of which remains ongoing.

20.8.1 Air Quality/Radon Monitoring

Atmospheric radon monitoring commenced in September 2016 to establish baseline radon levels in the project area. Passive radon detectors were deployed at 10 select locations in duplicate. On a quarterly basis, each deployed detector is exchanged with a new replacement detector, and each collected detector is sent to an accredited laboratory for analysis.

While there is currently no Canadian regulation that prescribes a radon threshold value in outdoor environments, Health Canada has developed a guideline for radon in indoor air for dwellings of 200 Bq/m^3. This guideline provides Canadians with guidance pertaining to when remedial action should be taken to reduce radon levels. Results to date demonstrate that baseline atmospheric radon levels within the project area are low, with the average radon concentration not exceeding 10 +/- 3 Bq/m^3 at any location. Radon monitoring will continue throughout 2018 and into the permitting and licencing processes, as required.

20.8.2 Aquatic Studies

Aquatic environment baseline field surveys were completed in all seasons throughout 2012 to 2018 to characterize seasonal variability in lakes and streams and identify preferred intake and discharge locations from candidate sites within proximity to proposed project infrastructure. Aquatic environment baseline field surveys completed in 2016 and 2017 by EcoMetrix Inc. augmented previous hydrological and aquatic baseline work completed in 2011 to 2014 by Golder Associates Ltd., included hydrology, water quality, sediment quality, aquatic habitat and bathymetry, plankton community, benthic invertebrate community and tissue chemistry, and fish community, spawning, and tissue chemistry.

20.8.3 Surface Water Hydrology

The project area is located within two distinct drainage areas that drain into Russel Lake, the Wheeler River, and ultimately into Wollaston Lake (via the Geikie River). Extending north and east of the project area, the Islander River drainage area drains approximately 371 km^2, while the Williams Lake drainage area is located south of the project area and drains approximately 78 km^2.

Hydrological baseline studies included manual streamflow measurements, staff gauge and elevation surveys, detailed bathymetric surveys, and continuous water level recording using dataloggers to develop rating curves at select stream locations within the project area. In September 2016, water level elevations were surveyed at 13 lakes and two ponds within the project area. In addition, manual streamflow measurements were taken at 16 watercourses and continuous streamflow monitoring equipment was installed at eight locations.

Project area lake and pond surface water elevations ranged from 520.86 MASL at an unnamed headwater lake, to 488.26 MASL at Russell Lake. In the Islander River drainage area, water level elevations at the stream stations ranged from 520.73 MASL at the most upstream station, to 492.71 MASL at the most downstream station. Stream flow measurements were recorded at 2.34 cm/s at the most downstream location of the Islander River drainage area.

In the Williams Lake drainage area, water levels at stream stations ranged from 518.33 MASL at the most upstream station, to 488.55 MASL at the most downstream station. Stream flow measurements recorded during this time were recorded at 0.64 cm/s at the most downstream location of the Williams Lake drainage area.

20.8.4 Surface Water Quality and Limnology

Baseline surface water quality was assessed at 17 lentic locations and 11 lotic stations within the project area. Water quality data was compiled for the years 2012, 2014, and 2016 and characterized by measuring physical and chemical constituents obtained in situ, as well as by accredited laboratory analyses. Surface waters within the project area were found to be comparable to other lakes in the region, which are classified as being soft with typically low levels of alkalinity, nutrients (nitrate and phosphorus), total dissolved solids, and total suspended solids. The pH of surface waters within the study area are slightly acidic to neutral.

In general, the concentrations of metals and metalloids were similar throughout the study area. Radionuclide concentrations were low, with the majority of measurements lower than their respective laboratory detection limits. For parameters with Saskatchewan Surface Water Quality Objectives (SSWQO) or Canadian Water Quality Guidelines (CWQG), most were below their respective guideline limits. Aluminum, cadmium, and lead concentrations exceeded guideline values at some locations, however, this appears to be a natural occurrence as demonstrated in surface water throughout the project area. Elevated concentrations of iron and mercury were measured near the lake bottom in lakes that exhibited thermal stratification at the time they were sampled.

Radionuclide concentrations measured in surface water are low within the study area, and generally below the laboratory detection limits of 0.02 Bq/L for lead-210, 0.005 Bq/L for polonium-210, 0.005 Bq/L for radium-226, and 0.01 Bq/L for thorium-228, thorium-230, and thorium-232.

Limnology profiles were recorded at the deepest location in each lake, measuring conductivity, pH, temperature, and dissolved oxygen. Thermal stratification of the water column was not observed in most project area lakes.

20.8.5 Aquatic Sediment Quality

Sediment samples were collected from the depositional areas of selected lakes for analysis of metals, radionuclides, total organic carbon, and particle size during the 2016 field study. Lake sediments within the project area were found to be generally silty-clay or sandy-silt with total organic carbon present at approximately 16%. For parameters with prescribed sediment quality guidelines, all constituent concentrations were found to be at, or below, their respective threshold values.

20.8.6 Benthic Invertebrate Community and Tissue Chemistry

Benthic invertebrate tissue samples were collected for analysis of metals and radionuclides at select project area locations in September 2016. The baseline field study identified 38 major taxonomic groups (families) present in project area waterbodies.

Benthic invertebrate larvae were collected from selected project area lakes, including Russel Lake and Kratchkowsky Lake, and analyzed for metals and radionuclides. The results of the analyses identified that radionuclide levels were generally below the laboratory method detection limit, with the exception of Po-210 and Ra-226. While metal concentrations observed in larvae tissues collected from project area lakes were generally consistent across all locations, cobalt and nickel concentrations were observed to be more variable. Larvae tissues collected from Russell Lake had higher concentrations of some metals, including aluminum, cobalt, and uranium, than other lakes in the project area.

Fish Community, Spawning, and Fish Tissue Chemistry

Aquatic environmental baseline field surveys commenced in 2012 to assess aquatic habitats throughout seasonal fluctuations in fish movements and spawning activities. Fish community surveys were undertaken in various habitat types in selected project area waterbodies to characterize fish species presence and community diversity. A total of 13 species of fish were collected within the project area during baseline surveys in September 2016 and May 2017. All waterbodies sampled, except one headwater pond, supported fish.

Eleven fish species were collected within study area lakes and stream stations, including lake chub, spottail shiner, longnose sucker, white sucker, lake whitefish, lake trout, northern pike, burbot, ninespine stickleback, yellow perch, and walleye.

Large-bodied fish spawning surveys were undertaken during the fall of 2016 and spring of 2017 at selected lake and stream locations to determine the utilization of these areas for spawning. Fall spawning species present within the study area include lake whitefish and lake trout, and potential spawning habitats for these species were identified in several project area lakes, including Kratchkowsky Lake. Spring spawning species present within the study area include walleye, northern pike, arctic grayling, white sucker, longnose sucker, and yellow perch. Spawning habitats for walleye and suckers were observed at most stream stations. Northern pike spawning habitats were present in nearly all study area lakes, as well as most stream stations.

Tissue samples (muscle and bone) collected in 2016 and 2017 from northern pike and white sucker were submitted for chemical and radiological analyses. Guideline values for constituents in fish tissue are available for mercury and selenium. Tissue samples collected were below these guideline values.

20.8.7 Plankton Community

Phytoplankton and zooplankton samples were collected in September 2016 at select project area lakes. The total biovolume (μm^3) of phytoplankton was greatest at LA-1 (13,992,383,000 μm^3) located east of the project site and was an order of magnitude less at all other sampling areas. In total, 55 phytoplankton taxa were identified from seven classes and at least six classes were identified in each of the waterbodies sampled. Diatoms were dominant at all locations, representing more than one-quarter and up to 90% of the total biovolume at each location.

The total biovolume for zooplankton ranged from approximately 5.0 x 109 μm^3 to 1.1 x 1,012 μm^3 at select project area lakes. Within each of the waterbodies sampled, 15 to 22 taxa were present, and seven to nine classes were identified. At all locations, chlorophyll-a concentrations were below the laboratory method detection limit (<0.60 µg/L), identifying that there is low primary productivity in the lakes, characteristic of the oligotrophic state of lakes within the project study area.

20.8.8 Geochemistry Baseline

On-site Kinetic Testing

As part of the geochemical characterization of waste rock, on-site kinetic testing (barrel testing) was initiated in March 2018 to assess the synergistic effects of site conditions (temperature, precipitation, and relative humidity) and scale on mine waste leaching behavior. On-site kinetic test results will be used to assist in the calibration of laboratory kinetic tests to field conditions for use in future project water quality predictions. While it is recommended that kinetic testing data be collected for a period of not less than three years to gain a comprehensive prediction of forecasted leaching behavior of waste rock, data compiled after one year can be used for laboratory calibration.

Fourteen core samples (six from the Gryphon deposit and eight from the Phoenix deposit) were collected and considered representative of waste rock based on the following considerations:

- Proximity to potential mine workings.
- Proximity to faults or pyritic zones identified in the geological model.
- Sulphur and static testing data, where available.
- Nickel, cobalt, and molybdenum content.

Static Testing

Static geochemical testing and acid base accounting (ABA) was completed in 2017 on Gryphon and Phoenix deposit core samples collected during the exploration drilling phase to characterize preliminary acid rock drainage (ARD) potential. Elemental data were used to examine preliminary metal leaching (ML) and acid potential (AP) for further assessment of management requirements for potentially acid generating (PAG) waste rock generated from mining activities at the Gryphon deposit.

Neutralization potential (NP) and carbonate (as total inorganic carbon (TIC)) were considered in estimating ARD potential of the Gryphon and Phoenix core samples. Using TIC/AP as the primary indicator of ARD potential, 50% of samples tested were PAG, 10% were non-PAG, and 40% were low sulphur (less than or equal to 0.02% S).

Sandstone and basement quartzite were predominantly classified as low sulphur and may have low enough sulphide content that they will not be characterized as acid generating. The samples tested demonstrated low potential for metal leaching under neutral pH conditions; however, further confirmatory analyses will be completed to ascertain this predicted lack of acid generation.

20.8.9 Groundwater Quality

Phoenix Deposit

In April 2017, two bulk groundwater samples were collected from the area immediately above the Phoenix ore zone. Based on exploration drilling/grouting practices, these samples were considered to be representative of the inflow from surrounding sandstone formations into the area.

The results from the groundwater sampling indicate a neutral pH (6.9 to 7.5), as anion chemistry was dominated by bicarbonate alkalinity and sulphate, whereas chloride was comparatively low. Cation chemistry was shown to be dominated by sodium, calcium, iron, and aluminum.

Dissolved iron concentrations were higher than expected given the pH of the samples, as iron hydroxides have low solubility at neutral pH, and under oxidizing conditions, iron is expected to precipitate. The iron results indicate it is likely that iron is out of equilibrium with surface conditions due to the change in redox conditions (to more oxidizing) produced by removal of the water from depth.

When compared to Canadian Water Quality Guidelines for the Protection of Aquatic Life, 2017 (CCME guidelines) (freshwater), results exceeded the prescribed criteria for aluminum, dissolved iron, dissolved arsenic, dissolved copper, dissolved lead, and dissolved uranium.

One Phoenix bulk groundwater sample was also analyzed for the radionuclides lead-210 and radium-226. The results of the analyses demonstrate that radium-226 was 7.2 Bq/L, exceeding the Saskatchewan Environmental Quality Guideline (SEQG) of 0.11 Bq/L for surface water (Government of Saskatchewan, 2017), while the concentration of lead-210 was found to be 2.1 Bq/L.

Gryphon Deposit

The most recent groundwater samples were collected from isolated zones spanning a total depth range of 75 to 465 meters below ground surface (MBGS) in September to October 2017 to characterize the groundwater quality at the Gryphon deposit. All samples were collected from sandstone units, as groundwater could not be collected from the underlying basement rock due to the low permeability characteristic of the formation in this area. Samples were assessed for dissolved metals, acidity, fluoride, pH, conductivity, total alkalinity, chloride, sulphate, and radionuclides.

Field pH was shown to be neutral in all four samples (6.5 to 7.0), and field conductivity measurements ranged from 105 to 203 µS/cm. Average water temperature was 5.5°C.

Bicarbonate was the dominant form of alkalinity, which is as expected for the measured pH values. Bicarbonate alkalinity ranged from 59 to 118 mg/L as $CaCO_3$ and were comparable to samples collected from drilling in the Phoenix deposit earlier in 2017. Chloride concentrations ranged from 4.3 to 7.0 mg/L and were below the CCME guidelines (freshwater), however, fluoride concentrations ranged from 0.11 to 0.16 mg/L and were above the CCME guideline of 0.12 mg/L for long-term freshwater concentrations. Sulphate concentrations ranged from 1.0 to 6.4 mg/L.

Other dissolved metals with concentrations exceeding or approaching the CCME guidelines included aluminum, iron, arsenic, copper, lead, silver, and uranium.

The most recent groundwater characterization at the Gryphon deposit also included analyses of the radionuclides lead-210, polonium-210, radium-226, thorium-228, thorium-230, and thorium-232. Results of the analyses demonstrate that radium-226 values ranged from 0.90 to 0.49 Bq/L, exceeding the Saskatchewan Environmental Quality Guideline (SEQG) of 0.11 Bq/L for surface water (Government of Saskatchewan, 2017). Lead-210 concentrations ranged from 0.40 to 0.29 Bq/L. Polonium-210 values were highest at the 438-462 m depth (0.41 Bq/L), and lowest at the 75-201 m depth (0.04 Bq/L). Thorium-230 also had highest and lowest values at 438-462 and 75-201 MBGS, with values of 0.32 and 0.06 Bq/L, respectively. Thorium-228 and thorium-232 values were greatest at 414-438 m depths, with values of 0.41 and 0.27 Bq/L, respectively.

20.8.10 Terrestrial Baseline

Commencing in September 2016, terrestrial (wildlife and vegetation resources) baseline studies were undertaken to characterize the existing terrestrial environment in the project area.

Predictive Ecosite, Anthropogenic, and Fire Mapping

In order to develop baseline disturbance and vegetation cover/fire mapping, as well as provide an accurate characterization of the vegetation cover for future monitoring and/or impact assessment purposes, predictive ecosite mapping was obtained from the Saskatchewan Technical Branch and enhanced to increase accuracy for both the project and regional study areas.

The predictive ecosite mapping identified that there are 22 different ecosite classifications located throughout the project area, with the most abundant being jack pine/blueberry/lichen (70%), waterbodies (13%), and jack pine/black spruce/feathermoss (5%). The results also identified that the broader regional study area was comprised of the same ecosite classifications, however differing slightly in their proportions (jack pine/blueberry/lichen (52%), waterbodies (21%), and jack pine/black spruce/feathermoss (13%)).

The results of the baseline anthropogenic map of the project study area identified that the total amount of anthropogenic disturbance in the project area is 2.9% (1.4 km^2), and 1.5% (5.8 km^2) identified in the broader regional study area.

Historical fire data was obtained from the Saskatchewan Ministry of Environment, Wildfire Management Branch to characterize the proportion of the project and regional study areas which have been disturbed by past fires. The results of the fire mapping survey identified that 43% percent of the project area landscape has burned within the last 30-50 years, and the remaining 57% of the landscape has forests aged 70 years and older.

Ecosite Characterization, Plant Structural Diversity, and Species Richness Assessment

Detailed vegetation and wildlife habitat characterization field surveys were undertaken between July 7 and 16, 2017 to describe and quantify the ecological and botanical conditions within recurring mapped ecosite types and regeneration forests. Sample site locations were widely distributed throughout the project area. One hundred and ninety-four species and/or genus of spp. were recorded during the vegetation field survey.

Rare Vascular Plant Surveys

Rare vascular plant surveys were completed to identify rare vascular plant occurrence(s) within the project and regional study areas, as well as to provide a scientifically defensible baseline for potential follow-up/monitoring requirements. In total, 66 vascular plant species and eight identifiable non-vascular species were identified, of which none were found to be invasive non-native plant species, and two of which are considered rare plant species (Alaskan clubmoss and three-seeded sedge).

Vegetation and Soil Collection and Chemistry Analysis

The vegetation and soil sampling program was undertaken between August 2 and 7, 2017. Blueberry stems, leaves, fruit (currents years' growth), terrestrial lichen, and soil samples were collected to determine baseline conditions of physical properties, inorganic ions, metals, and radionuclides in vegetation (blueberry and lichen) and soil samples, as well as to support future monitoring, mitigation, and impact assessments.

Lichen and blueberry radionuclide levels were relatively consistent across the project study area. Metal parameters were variable but relatively consistent, aside from elevated levels of aluminum, chromium, iron, lead, titanium, and vanadium observed at one location.

Radionuclide levels in soil were also variable but relatively consistent, with the exception of one sample site located northeast of Russel Lake where higher levels of lead-210 and polonium-210 were observed compared to other sample sites. Elevated levels of calcium, copper, lead, and manganese were also observed at this location compared to other sample sites.

Winter Track Count Survey

Winter track count surveys were completed between January 25 and February 3, 2017 and in February and March 2018 to determine the presence and relative abundance of winter-active animals, to enhance project understanding of species-ecosite affiliation, and to support future monitoring, mitigation, and impact assessments. Tracks from the following species were observed in the project area during the winter track count surveys:

- Snowshoe hare;
- Red squirrel;
- Grouse;
- Fisher;
- Moose;
- Microtine;
- Marten;
- Canada lynx;
- Otter;
- Ermine;
- Woodland caribou;
- Mink; and
- Red fox.

Ungulate Pellet Group/Browse Availability Survey

Pellet group/browse availability transects were completed between June 9 and 20, 2017, and June 6 and 12, 2018 to collect baseline data on the presence and relative abundance of ungulates (moose and woodland caribou), carnivores, and game birds (grouse/ptarmigan species). The transects were also used to determine the frequency of occurrence and abundance of terrestrial and arboreal lichen, as this species is vital to the woodland caribou population. Pellets or scats of the following seven species were detected during the pellet group/browse availability surveys:

- Grouse/ptarmigan;
- Moose;
- Woodland caribou;
- Black bear;
- Red fox;
- Mink; and
- Martin.

The pellet group/browse availability surveys will provide for scientifically defensible baseline data to support future impact assessments and to allow for potential future follow-up/monitoring requirements.

Terrestrial lichen was observed in all ecosite/vegetation cover types sampled, except in areas where black spruce/balsam poplar/river alder swamp and willow shrubby rich fen covers were most prominent. Frequency of occurrence was the highest (greater than 99%) in areas covered by jack pine/blueberry/lichen.

Arboreal lichen occurred in 79% of ecosites/vegetation cover types surveyed throughout the project area and were observed to be most abundant in areas covered by jack pine/blueberry/lichen.

Small Mammal Trapping Survey and Tissue Analysis

A small mammal trapping program was completed between September 24 and October 2, 2016 to determine the species composition and relative abundance of voles, mice, and shrews, as well as to collect specimens for baseline metal and radionuclide tissue analyses.

With an overall capture rate of 7.7 captures per 100 trap nights, a total of 197 individual small mammals from the following three species were captured during the program:

- Red-backed voles;
- Meadow voles; and
- Dusky shrews.

The small mammal trap lines were stratified by three general areas: Gryphon deposit, Phoenix deposit, and a reference location. A total of 124 red-backed vole specimens were submitted for metals and radionuclide analysis (29 from the Phoenix deposit and 18 from the Gryphon deposit). Samples collected at the Phoenix deposit indicated elevated levels of aluminum, titanium, uranium, and radium-226 in comparison to other sites surveyed.

Amphibian Nocturnal Call Survey

An amphibian nocturnal call survey was completed between June 16 and 20, 2017 to establish the presence and relative abundance of amphibian species within the project and regional study areas. While wood frogs were detected during the survey in the project study area, Boreal chorus frogs were observed incidentally during other field surveys.

Breeding Songbird Point Count Call Survey

Breeding songbird point count call surveys were undertaken between June 7 and 17, 2017 to document the diversity and relative abundance of breeding songbirds within the project study area, as well as to determine the presence of known or potential avian species at risk. Three hundred and nineteen indicated pairs were observed in the project study area. The highest number of breeding songbird pairs were detected in jack pine/white birch/feathermoss cover. The following list provides the 10 most common species detected:

- Ruby-crowned kinglet;
- Dark-eyed junco;
- Gray jay;
- Yellow-rumped warbler;
- Swainson’s thrush;

- Hermit thrush;
- Lincoln sparrow;
- Chipping sparrow;
- Fox sparrow; and
- American robin.

Semi-aquatic Furbearer Shoreline Survey

Semi-aquatic furbearer shoreline surveys were conducted along shorelines of select creeks, lakes, and ponds between September 29 and October 3, 2016 to provide quantitative data on the occurrence and relative abundance of semi-aquatic furbearing mammals (muskrat, mink, beaver, and otter) and to collect spatial data on the distribution within the project and regional study areas. Signs of three target species, namely muskrat, beaver, and river otter, were observed during the survey.

Aerial Waterfowl and Raptor Stick Nest Survey

The aerial waterfowl and raptor stick nest survey was completed across 33 survey sections containing 353 water bodies on June 15 and 16, 2017 to document the presence, diversity, and abundance of breeding waterfowl (including species at risk), as well as to identify the occurrence of active, inactive, and old raptor nests (i.e. bald eagle, osprey, and red-tailed hawk). The survey recorded 20 confirmed unique species and six species groups, for a total of 681 individual waterfowl/raptor(s). The ten most commonly observed species were:

- Ring-necked duck;
- Common merganser;
- Common loon;
- Mallard;
- White-headed gull;
- Bald eagle;
- Canada goose;
- Lesser scaup;
- Yellow legs spp; and
- Bufflehead.

A total of 24 active (currently occupied), inactive (not currently occupied), and old (dilapidated) nests were observed in the project area during the survey. Eleven nests were active including four bald eagle nests, four osprey nests, one raven nest, one herring gull nest, and one common loon nest, as well as one mew gull colony of 12-15 nests.

20.8.11 Species at Risk and Sensitive Species

Sensitive species are defined as a species having a ranking of S3 or lower by the Saskatchewan Conservation Data Centre (SKCDC), or a species with a disturbance setback outlined in the Saskatchewan Ministry of Environment Activity Restriction Guidelines for Sensitive Species (2017).

Thirteen sensitive or federally/provincially listed species at risk were observed within the project study area (olive-sided flycatcher and common nighthawk), and broader regional study area (woodland caribou), nine of which have seasonal setback distances based on the activity restrictions guidelines. Prescribed setbacks have been established and will be considered in future project planning, monitoring, and impact assessments.

20.8.12 Heritage Resource

The project was submitted to the Heritage Conservation Branch (Ministry of Parks, Culture and Sport) for heritage screening. It was identified that portions of the proposed infrastructure and access roads will impact hilly terrain and prominent uplands located within heritage sensitive areas. Accordingly, a Heritage Resource Impact Assessment (HRIA) requirement was attached to the project, pursuant to Section 63 of The Heritage Property Act (1979-80).

A heritage resources baseline study was initiated on July 5, 2017 (Golder Associates Ltd.) under Archaeological Resource Investigation, Permit 17-091. Heritage sensitive areas were assessed through a combination of pedestrian reconnaissance and visual inspection field programs, complimented by the excavation of 258 shovel probes and 5 shovel tests. The assessment identified an Artifact Find site (HiNi-6) of an unknown precontact cultural affiliation located on the western terrace of a lake adjacent to the Phoenix 2 access road option.

Upon completion of the Heritage Resources Impact Assessment, it was determined that the regulatory requirements were satisfactorily completed, thereby which an approval was issued to the project by the Heritage Conservation Branch on December 14, 2017.

Where additional project infrastructure areas and/or access roads are required to provide for site layout modifications, subsequent heritage resource baseline studies will be undertaken, and approval will be received prior to executing future land disturbances.

20.9 Social Considerations and Stakeholder Engagement

During the spring of 2016, Denison developed a Stakeholder Engagement Management Plan. This plan was developed in accordance with provincial, federal, and international guidance (International Finance Corporation Performance Standards on Environmental and Social Sustainability) for stakeholder engagement.

The initiation of the Stakeholder Engagement Management Plan was timed to coincide with the scoping of the environmental baseline data collection programs to allow for the integration of traditional knowledge obtained from the stakeholder groups into the design of the baseline collection programs and the ongoing engineering designs of the proposed project.

Four communities were identified as key stakeholders to be engaged as part of the advancement of the Wheeler River project; Patuanak and English River First Nation, Pinehouse and the Kineepik Metis Local, Beauval and the Sipisishik Metis Local 37, and Ile a la Crosse and the A La Baie Metis Local 21.

Since the initiation of the Stakeholder Engagement Management Plan in the summer of 2016, over 20 face-to-face meetings have taken place between the communities, First Nations and Metis leadership, community economic development representatives, and community residents. Input from these engagement sessions has resulted in the offering and subsequent integration of traditional knowledge into the environmental baseline collection programs, the engineering design of the project's infrastructure, and the mining method options evaluation.

As the project advances with respect to a toll milling agreement, Denison is prepared to expand its stakeholder engagement to include the Athabasca Basin based stakeholders. To that end, Denison presented an overview of the project to the Northern Saskatchewan Environmental Quality Committee, which has members representing each of the Athabasca Basin communities.

20.10 Decommissioning and Reclamation

The project will involve mining of two deposits, Gryphon and Phoenix. The Gryphon deposit will be mined by conventional underground longhole stoping accessed via a headframe. This mine will require the typical surface infrastructure, such as a special waste and clean waste rock pads, ore pads, overburden stockpile area, lay down areas, water treatment plant, and associated clean and dirty water storage ponds.

The Phoenix deposit will be mined using in situ recovery, which consists of a series of injection, recovery, and monitoring wells, along with a number of distribution points (headerhouses) for the transfer of uranium-bearing mining solutions to and from the wellfield. The uranium bearing solution is transferred to a processing plant that will recover the dissolved uranium via direct precipitation. The mining of this orebody will also require ground freezing, a water treatment plant, and clean and dirty holding pond(s). The water treatment plant will be needed to remove iron and radium from the uranium bearing solutions as part of the uranium recovery process. Water treatment will also be required as part of the wellfield restoration process, such that clean waters will be re-injected into the subsurface geology where uranium has been removed to ensure acceptable pre-mining groundwater conditions remain post extraction and over the long term. A relatively small lined pad will also be required. This pad will be dedicated to storing precipitates from uranium recovery, as well as mineralized drill cuttings produced from drilling the injection, recovery, and monitoring wells for the production phase of this deposit. Non-mineralized drill cuttings from these wells will be stored on an unlined clean rock pad.

Closure of the entire project will be completed in accordance with all provincial and federal regulations and guidance documents, with the fundamental consideration being to ensure physical and chemical stability of the site in order to protect human and ecological health and the environment. The five main closure activities include:

- Decontamination;
- Asset removal;
- Demolition and disposal;
- Rehabilitation; and
- Monitoring and reporting.

Progressive rehabilitation will be completed throughout the life of the project whenever feasible. Progressive rehabilitation activities will focus on the decontamination, demolition, and disposal of unused buildings and infrastructure, as well as the removal of unused equipment and machinery. Progressive rehabilitation of waste rock piles and other inactive areas will take place when these areas become available. Progressive rehabilitation will be reported to the regulatory agencies as part of the annual reporting requirements throughout operations.

20.10.1 Decontamination

Surface facilities and the underground workings at Gryphon and injection, recovery, and monitoring wells at Phoenix will be decontaminated as necessary. Surplus chemicals and other hazardous materials will be removed and stored in designated temporary storage facilities. Sumps will be cleaned. All hazardous materials will be disposed of at approved off-site facilities. All radiologically contaminated material will be disposed of on-site in accordance with license conditions.

Empty tanks will be removed from the site and sold as scrap or reused. Otherwise, they will be transported to an approved waste management facility. Fuel tanks will be managed by a contractor licensed to handle these types of tanks. Any remaining fuel will be removed by the contractor and then the contractor will remove the tanks from site. As much waste as possible will be hauled off-site and disposed of at appropriate licensed facilities.

20.10.2 Asset Removal

Salvageable machinery, equipment, and other materials will be dismantled, decontaminated, and taken off-site for resale or recycling. Remaining items will either be managed at a facility licensed to manage radioactive wastes or disposed of in an approved facility on-site.

20.10.3 Demolition and Disposal

All permanent structures that cannot be removed from the property as an asset will require demolition. Most process equipment and non-supporting structures will be removed from buildings prior to demolition and the buildings will be demolished.

During demolition, dust control will be required. An initial wash may be necessary, in addition to the wetting of demolition debris as structures are disturbed during demolition. The requirement and duration of misting will be determined on a case-by-case basis.

A review prior to the start of demolition will identify areas requiring additional procedures. Where possible, dust generating materials will be removed prior to demolition. Appropriate personal protective equipment and personnel decontamination procedures will be employed.

Valuable recyclable materials will be separated and processed for transport and sale concurrent with demolition. Excavators equipped with grapples will sort the recyclable products from the non-recyclables. Shears will be used to size recyclables for shipping and sale. Cleaning procedures of recyclables will be integrated into demolition, as necessary.

Concrete foundations will be left in place. Any portions of concrete foundations remaining above grade will be levelled and rebar will be cut-off at grade. Large slabs will be perforated on a 2 m grid to permit drainage. Concrete slabs will be covered with 0.5 m of development rock or locally stockpiled till.

The demolition process will produce:

- Saleable recyclable materials (steel, stainless steel, copper, steel sections, and sheet metal);
- Hazardous materials;
- Roofing materials and insulation;
- Wood;
- Concrete; and
- Contaminated soils.

Saleable recyclable materials will also be transported off-site as scrap or recycled.

Hazardous materials will be handled and disposed of in accordance with the appropriate regulations and good practice. Where possible, chemicals will be mixed to produce a neutral solution and disposed of in an approved manner at site. Hazardous materials, such as spent chemicals (that cannot be managed onsite), waste oil, and sludges, will be disposed of off-site at licensed facilities.

Non-hazardous waste materials, such as roofing materials, insulation, wood, co-mingled concrete, and light steel (i.e. hand railings), will be disposed of off-site in a licenced landfill. Soil testing will be conducted in any areas of known contamination and/or potential spills, including areas around chemical, fuel, and explosive storage areas. Testing will be conducted according to industry standard procedures and compared to provincial and federal soil standards.

20.10.4 Rehabilitation

An overview of the rehabilitation activities that will be completed for main project components is provided below. The main project components that will require rehabilitation at closure include:

- Underground workings and openings to surface;
- ISR wellfield and infrastructure;
- Transportation corridors and laydown areas;
- Ancillary infrastructure;
- Waste rock and overburden piles; and
- Water impoundments.

Detailed descriptions of the rehabilitation requirements for the above project components are provided below.

Underground Workings and Openings to Surface

The closure of the underground mine will require the following activities:

- Removing pumps, rolling equipment, oils, fuels, solvents, and all hazardous materials;
- Allowing the underground workings to naturally flood;
- Demolishing aboveground infrastructure (i.e. head frame, fans, heaters, collars, etc.);
- Capping the shaft with a reinforced concrete cap to prevent inadvertent access, in accordance with the provincial code; and
- Capping or backfilling raises and other openings to the surface to prevent inadvertent access, in accordance with the provincial code.

ISR Wellfield and Infrastructure

Closure of the ISR wellfield operations will require the following activities:

- Decommissioning of all injection and recovery wells, following acceptable wellfield restoration;
- Removal, decontamination, and disposal of all surface piping;
- Decontamination and removal of the transfer station;
- Decontamination, removal, and/or disposal of the direct precipitation plant;
- Allowing the freeze curtain to thaw and decommissioning of all freeze pipes and freeze plant;
- Re-slurry the iron/radium sludge from the storage pond and placing solid/slurried material into the underground workings of the Gryphon deposit or in an approved long-term licensed facility; and
- Placement of all special waste drill cuttings into the mined-out portions of the Gryphon deposit as backfill.

Transportation Corridors and Laydown Areas

Transportation corridors will be graded and scarified to promote natural revegetation. Access roads required for post-closure monitoring will be left as is and maintained to permit access. Access to the site will be restricted by gates and/or berms.

Laydown areas will be scarified, covered with 0.5 to 1.0 m of stockpiled overburden, and vegetated with native self-sustaining species.

Ancillary Infrastructure

Rehabilitation of ancillary infrastructure components involves the following:

- Decommissioning and removal of power transmission lines and electrical infrastructure once they are no longer required to support passive closure activities (i.e. post-closure water treatment);
- Decommissioning and removal of aboveground water pipelines;
- Scarifying corridors and allowing them to naturally revegetate (portions of the corridor located near sensitive environments, or that are easily erodible, will be seeded to enhance the physical stability); and
- Decommissioning and removal of the water treatment plants and appurtenances once water quality meets discharge requirements without treatment.

Waste Rock and Overburden Piles

Any piles remaining at closure will be left in a stable condition. This may involve leaving the pile as constructed or re-contouring as necessary. If the waste rock is clean and the quality of the environment is protected, then no cover will be applied.

Remaining portions of the overburden stockpile that are not used for cover material elsewhere on the site will be re-contoured and vegetated with native self-sustaining species. The footprint of the overburden stockpile will be scarified to reduce compaction and vegetated with native self-sustaining species.

Water Impoundments

Water ponds and lined settling pond pads will be decommissioned once they are no longer required for water management. Berms and/or dams will be breached and re-contoured to restore natural drainage. Any liners will be removed and placed within the underground workings or hauled to an off-site landfill. The footprints of impoundment areas will be vegetated with native self-sustaining species.

20.10.5 Monitoring and Reporting

Following closure, physical, chemical, and biological monitoring of the site will be conducted to ensure that the site is chemically and physically stable. The monitoring programs will be designed and conducted in accordance with the provincial and federal regulations and license conditions.

The following is a summary of the anticipated monitoring programs:

- Mine water quality;
- Groundwater quality;
- Physical stability;
- Biological; and
- Surface water quality.

The monitoring programs will be conducted until the site-specific decommissioning and reclamation objectives for the project are met. Monitoring reports will be developed and submitted to both the provincial and federal regulators, in accordance with licensing conditions.

21 Capital and Operating Costs

21.1 Basis of Cost Estimates

Capital and operating cost estimates were developed to support the pre-feasibility study of the Gryphon and Phoenix deposits. The estimates address the initial capital, sustaining capital and operating costs required to engineer, procure, construct, commissioning, start-up and operate the mines and ISR precipitation plant and related infrastructure at the Wheeler River site and processing of Gryphon feed at the McClean Lake mill.

Several consultants, contractors and Denison's engineering team contributed to the estimates. Estimates by contributor along with the scope for which they were responsible include:

- Denison – Phoenix in-situ mining wellfield drilling;
- Newmans Geotechnique – Phoenix ground freezing;
- Engcomp – Phoenix and Gryphon surface infrastructure, and in-situ recovery;
- Stantec – Gryphon production and ventilation shaft equipping and underground mining;
- Frontier Kemper – Gryphon ventilation and production shafts excavation and lining;
- SRK – Phoenix and Gryphon Decommissioning, water treatment plant costing;
- Hatch – McClean mill modifications for Gryphon feed; and
- Denison – Toll milling and tailings storage fees for Gryphon mill feed.

The general parameters of the cost estimates include:

- Cost estimates are completed to AACE class 4 level with an accuracy of -15% to -30% on the low side and +20% to +50% on the high side;
- All costs are in Q2 2018 Canadian dollars;
- Quotations received in US dollars were converted to Canadian dollars at $1US = $1.25CA. Quotations received in Euros were converted to Canadian at $1EU = $1.54CA;
- Calculations are based on SI (metric) units;
- Productivity and labour parameters of the cost estimates for activities at the Wheeler site include:
    - The construction schedule consists of one 12 hours shift per day, seven days per week on 14 days in and 14 days out rotation;
    - Operating schedule is typically two 10.5 hour shifts per day, seven days per week on a 14 day in and 14 days out rotation; and
    - Annual schedule of 362 days per year.
- Slightly different parameters were applied for McClean Lake activities to conform to practices applied at McClean including application of a 66-hour work week with 14 days in and 14 days out rotation;
- Contractor workforce is assumed for all construction activities on surface, shaft and underground infrastructure construction activities; and
- Company employees are assumed for operations and underground development and production activities.

The estimate is broken down into three main components 1) Initial Capital Costs, 2) Sustaining Capital Costs and 3) Operating Costs.

21.2 Capital Cost Summary

The Wheeler River project total capital cost is estimated at approximately $1.13 billion, comprised of $322 million of initial capital for Phoenix and $623 million of initial capital for Gryphon as outlined in Table 21-1. All initial capital costs include a 25% contingency while the majority of sustaining capital costs carries zero contingency.

Table 21-1. Capital Cost Summary

Wheeler River Capital Cost (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Phoenix	$ 322,539	$ 103,411	$ 425,950
Gryphon	$ 623,120	$ 82,743	$ 705,862
Sub Total	**$ 945,659**	**$ 186,154**	**$ 1,131,813**

Capital costs are broken down into direct, indirect, owners and contingency categories.

Direct Costs

Direct costs are the costs of all equipment and bulk materials, together with construction and installation costs for all permanent facilities. The estimating methodology is generally a combination of factored equipment and unit costs for direct costs. Major equipment costs are based on recent budget prices from vendors based on preliminary requirements. Facility costs are estimated by approximate quantity take-offs from drawings and applying unit costs. Earthworks and infrastructure are derived from preliminary contours for overland piping, overhead power lines, etc. Equipment installation is estimated by a combination of equipment factoring, equipment lists and unit costs based on available scope definition.

Direct cost estimates for mine development and production costs are developed from first principles. This includes combining the following information:

- Quantity take-offs developed by the mining, estimating and design teams for select items with allowances applied; and
- Build-up of composite costs for labour, material and operation of equipment.

Mobile equipment costs are based upon procurement of new equipment. For major overhauls and rebuilds, a 15% allowance per year is applied to each piece of major mobile equipment, beginning 3 years after initial purchase.

Examples of direct costs include, but are not limited to the following:

- Supply, assembly and installation of permanent equipment;
- Supply, fabrication and installation of bulk materials;
- Supplemental resources for equipment and bulk material installation, such as labour and construction equipment;
- Site preparations (bulk earthworks) and the construction of roads and storm water ditching Supply, fabrication and erection of permanent buildings and associated services including a permanent camp and airstrip;
- Supply, fabrication, erection of utilities and distribution systems;
- Process control systems including software programming and DCS/HMI configuration costs;
- Labour, equipment and materials to complete underground excavation activities; and
- Decommissioning costs for demolition, earthworks, etc.

Indirect Costs

Indirect costs are defined as support or temporary activities not directly related to the project but still required for project execution. Most of the costs have been estimated as a percentage of direct costs with some costs being determined from first principles where appropriate. Examples of indirect costs include, but are not limited to the following:

- Purchase and installation of a temporary construction camp;
- Temporary facilities for construction, offices, laydown areas, warehousing, etc.;
- Temporary services including IT, catering, camp and office cleaning, etc.;
- Provision of power, heat and water and other consumables during construction;
- Construction rental equipment;
- Contractor mobilization / demobilization;
- First fills of equipment;
- Freight & Duty;
- Detailed engineering for both Phoenix and Gryphon;
- Construction Third Party Inspection and Testing;
- Vendor Representatives;
- Commissioning & Start-up; and
- Engineering, procurement and construction management services for McClean mill (including travel expenses).

Owner's Costs

Owner's Costs includes: owner's management team for procurement, construction management, human resources, safety, environment, miscellaneous field labour, costs for operation of camp during construction activities, flights and travel costs during construction from Saskatoon to the Wheeler site; sales taxes and regulatory fees, etc. Owner's costs have been estimated on a first principles basis.

Contingency

For initial capital, three (3) contingency percentages were applied to all direct, indirect and owner's costs depending on the level of project definition and quality of the cost applicable to various scopes of work.

- A 25% contingency was applied to the cost estimates except for:
  - Highway 914 Extension – 20%; and
  - Main power line connecting the provincial power grid to Wheeler River property 20%.

Exclusions

Excluded from the cost estimates are:

- Escalation beyond Q2 2018;
- Contingency on most sustaining capital costs and operating costs;
- Company risk reserve;
- Sunk costs;
- Legal costs; and

- Costs resulting from unanticipated scheduling delays caused by scope changes or labour disputes, inadequate construction management or changes in execution strategy.

21.3 Phoenix Capital Cost Summary

The capital costs for the ISR mining of the Phoenix deposit are categorized in Table 21-2.

Table 21-2. Phoenix Capital Cost Summary

Phoenix Capital Cost Details (1,000's)			
Direct Capital Costs	**Initial**	**Sustaining**	**Total**
Wellfield	$ 63,674	$ 35,402	$ 99,076
ISR Precipitation Plant	$ 50,935	$ 4,606	$ 55,541
Water Treatment Plant	$ 1,268	$ 18,676	$ 19,944
Surface Facilities	$ 22,325	$ 49	$ 22,374
Utilities	$ 6,538	$ 803	$ 7,341
Electrical	$ 18,834	$ -	$ 18,834
Civil & Earthworks	$ 44,309	$ 1,331	$ 45,640
Offsite Infrastructure	$ 7,950	$ -	$ 7,950
Decommissioning	$ -	$ 27,454	$ 27,454
Total Direct Costs	**$ 215,834**	**$ 88,321**	**$ 304,155**
Indirect Costs	$ 28,288	$ 5,669	$ 33,957
Owner's Costs	$ 14,227		$ 14,227
Contingency Costs	$ 64,190	$ 9,421	$ 73,611
Total Costs	**$ 322,539**	**$ 103,411**	**$ 425,950**

Overall, direct costs for Phoenix are the largest component of the project at 67% with indirects and owner's costs totalling 13% as indicated in Figure 21-1. In remote sites indirect costs are normally higher (20-40%) but due to several factors project costs are focused on the direct drivers.

These factors include:

- Procurement strategy of modularized facilities to minimize installation requirements on site;
- Large capital components (wellfield) are primarily drilling of wellfield and freeze holes. Inherently drilling operations are self-sufficient and require minimal personnel and support cost;
- Large quantities of civil and earthworks for site preparation which similarly require minimal personnel; and
- Overall simplified processing plant layout and with smaller modularized tanks to minimize on site construction.

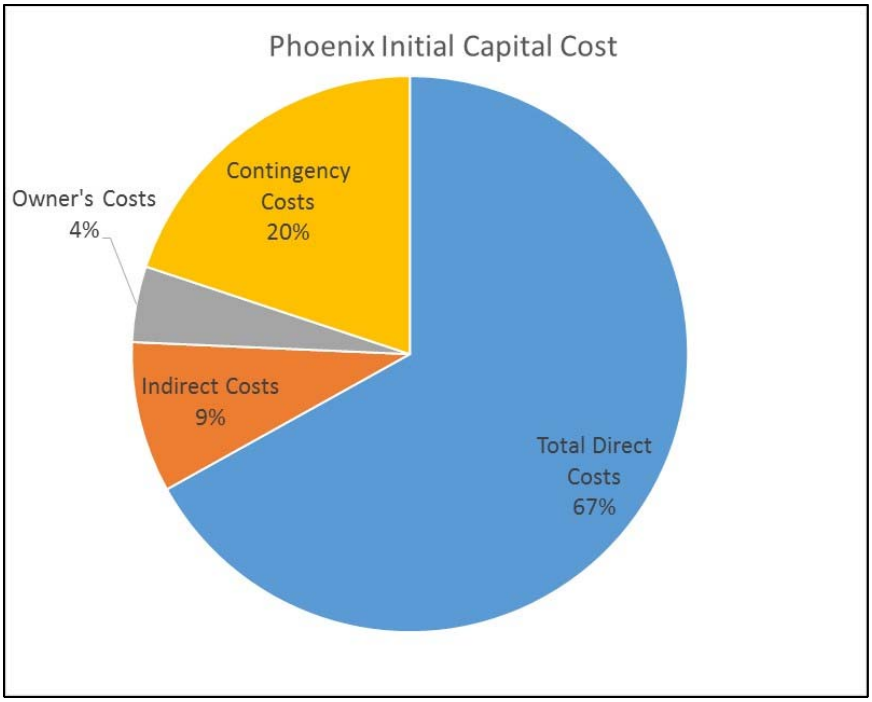


Figure 21-1. Phoenix Initial Capital Cost Distribution

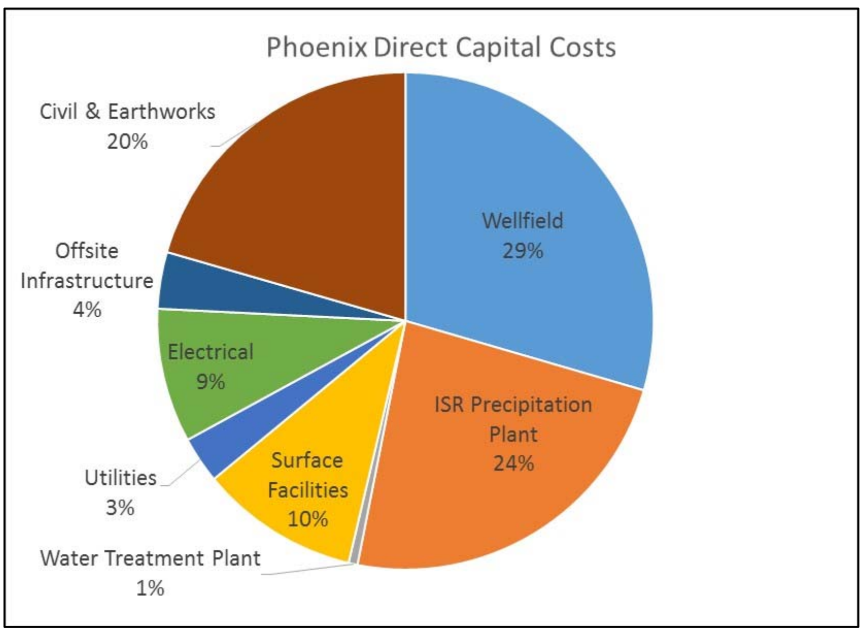


Figure 21-2. Phoenix Direct Capital Cost Distribution

In evaluating the major drivers of cost the top three items of the wellfield, civil and earthworks and the ISR precipitation plant construction amount to over 70% of the capital costs as identified in Figure 21-2. Table 21-3 provides the design and cost basis for these items.

Table 21-3. Phoenix Key Capital Cost Estimate Details

AREA	Design Basis	Cost Basis
Wellfield	Offtake from drawings	Contractor Quotation
Civil and Earthworks	Lidar based quantity estimate	Engineer's Cost Database
ISR Precipitation Plant	Derived from metallurgical Mass balance and flow sheet sizing	Equipment quotes from vendors, building costed based on $/sq.m

Another important aspect of the project is the level of control that Denison would have in execution. Based on the Phoenix project design, the only piece of off-site infrastructure required is a new five km 138 kV powerline installed from the provincial power transmission line to the Phoenix site.

21.3.1 Wellfield Capital Cost Details

Table 21-4 provides a breakdown of the wellfield capital costs.

Table 21-4. Wellfield Capital Cost Details

Wellfield Capital Cost Details (1,000's)			
Area	Initial	Sustaining	Total
Wellfield	$ 6,485	$ 33,905	$ 40,390
Ground Freezing	$ -	$ -	$ -
Freeze Hole Drilling	$ 29,336	$ -	$ 29,336
Ground Freezing Infrastructure and initial freeze costs	$ 25,320	$ -	$ 25,320
Pumping and Piping	$ 660	$ 1,475	$ 2,135
Special and Clean Waste Storage	$ 1,873	$ 22	$ 1,895
Sub Total	**$ 63,674**	**$ 35,402**	**$ 99,076**

Wellfield capital costs include and are based on:

- The wellfield drilling estimate is based on a budget quote from a drilling contractor. The cost estimate is inclusive of mobilization, demobilization, engineering support, supervision, labour, subsistence materials and equipment rentals and all contractor indirects;
- Wellfield downhole HDPE liner pipes, grout and screen material and installation labour estimates were provided by Denison based on calculated totals;
- Freeze hole drilling is based on a quotation provided by an experienced drill contractor in the Athabasca basin. The contractor's quote is all inclusive of mobilization, demobilization, engineering support, supervision, labour, materials, subsistence, equipment rentals and fuel;
- Freeze plant pricing is based on a detailed 2012 Calgary, Alberta located vendor supplied budget quotation with an assumed 30% increase to account for the five years elapsed since the quotation was provided. NGI has had personal communication with freeze plant vendors in the past year to suggest the total values presented here are reasonable within the level of accuracy of the study. The freeze plants are assumed to be modular in design and shipped on a structural frame to be placed on a concrete pad;
- Brine system pumping piping, valves, insulation, fittings, hoses, clamps, flanges etc. pricing is based on a percentage of the freeze plant costs except for specific items where pricing is based on recent NGI experience;

- Freeze plant operational staff are assumed contracted out on a yearly basis at $350,000 per year. This would be two operators cross shifting each other on a year-round basis;
- Electrical and instrumentation to monitor the freeze process is based on an assumption of 4 extra vertical holes to monitor ground temperature, though existing exploration holes could be re-purposed. Other monitoring is listed as a nominal lump sum but it is a small part of the overall freeze budget;
- Labour for freeze plant construction is estimated to include a 20-person crew over two months. This would include supervision. Excluded from the estimate are travel/accommodation for all construction personnel on site, detailed engineering design, freeze operations oversight, and other Denison related indirect costs; and
- Storage pads for clean and special waste generated during wellfield and freeze hole drilling have been estimated from first principles based on design quantity takeoffs.

21.3.2 ISR Precipitation Plant Capital Cost Details

The ISR precipitation plant capital costs are provided in Table 21-5.

Table 21-5. ISR Precipitation Plant Capital Cost Details

ISR Precipitation Plant Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Process Plant Equipment	$ 17,912	$ 4,538	$ 22,451
Process Plant Distributable Materials	$ 15,215	$ 68	$ 15,283
Process Plant Building	$ 17,808	$ -	$ 17,808
Sub Total	**$ 50,935**	**$ 4,606**	**$ 55,541**

The engineering deliverables produced for the ISR plant include mass balance, process flow diagrams, a mechanical equipment list and general arrangement drawings. The costs include and are based on:

- Process plant equipment includes vessels, tanks, pumps, etc. that have been sized based on the plant flow sheet design and mass balance. The costs were obtained from vendor quotations.
- Process plant distributable materials includes all equipment foundations and support steel within the process plant, interconnecting process piping, process ventilation, process electrical, instrumentation, insulation and special coatings. Costs for these items are based on a percentage of the plant equipment costs.
- The process plant building includes all foundations, concrete, pre-engineered building, control room, electrical and mechanical rooms, and building services. The building cost is based on quotation.

21.3.3 ISR Surface Infrastructure Capital Cost Details

Table 21-6 lists the surface infrastructure cost detail.

Table 21-6. Surface Infrastructure Capital Cost Details

Surface Infrastructure Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Operations Complex	$ 12,397	$ -	$ 12,397
Camp	$ 5,388	$ -	$ 5,388
Wash Bay / Scanning Facility	$ 1,313	$ -	$ 1,313
Fenced Storage	$ 32	$ -	$ 32
Outdoor Covered Storage	$ 265	$ -	$ 265
Surface Mobile Equipment	$ 2,640	$ -	$ 2,640
Security Gatehouse	$ 47	$ -	$ 47
Weigh Scale	$ 152	$ -	$ 152
Incinerator	$ 92	$ 49	$ 141
Sub Total	**$ 22,325**	**$ 49**	**$ 22,374**

The engineering deliverables produced for the surface infrastructure include a site plan with facilities from Table 21-6, approximate size of building and other facilities and general requirements. Costs for the surface infrastructure are based on a combination of budget quotes for and Engcomp's cost database from recently completed, similar projects.

21.3.4 Phoenix Civil and Earthworks Capital Cost Details

Table 21-7 lists the Phoenix associated civil and earthworks cost details.

Table 21-7. ISR Precipitation Plant Capital Cost Details

Surface Infrastructure Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
General Site Prep, ponds and landfill	$ 25,914	$ 1,331	$ 27,245
Roads	$ 11,262	$ -	$ 11,262
Airstrip and Terminal Building	$ 7,133	$ -	$ 7,133
Sub Total	**$ 44,309**	**$ 1,331**	**$ 46,123**

The design basis for the Phoenix civil & earthworks scope includes existing contours and grading plans for surface works a typical road cross section. The scope of the civil works includes, clearing and grubbing, removal of topsoil, earth excavation to embankment, import of fill, pond and landfill geosynthetic liners, fencing, road base course, traffic gravel, CSP culverts, rip-rap and seal coat.

Costs for the Phoenix civil & earthworks scope are based on unit pricing from recent material supply and construction costs and adjusted for the location of the project. Costs were determined by applying the unit prices to the quantity take-offs based on the conceptual design. Allowances were made for haul and borrow material and disposal or surplus rock.

21.3.5 Phoenix Electrical Capital Cost Details

Table 21-8 lists the Phoenix associated electrical cost details.

Table 21-8. Phoenix Electrical Capital Cost Details

Surface Infrastructure Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Main 138kV to 25kV substation	$ 4,656	$ -	$ 4,656
Site Electrical Distribution	$ 7,455	$ -	$ 7,455
Phoenix Process Plant Electrical	$ 1,465	$ -	$ 1,465
Operations Centre Electrical	$ 1,046	$ -	$ 1,046
Camp Electrical	$ 516	$ -	$ 516
Freeze Plant Electrical	$ 3,705	$ -	$ 3,705
Sub Total	**$ 18,834**	$ -	**$ 18,834**

The design basis for the Phoenix electrical scope is based on pre-feasibility level engineering design including, overall surface electrical single line diagram, preliminary calculations using ETAP software, electrical input the mechanical equipment list, and the SaskPower Conceptual Load Interconnection Assessment Report.

Costs for the Phoenix electrical scope is based on a combination of budgetary quotes from vendors for large conductors and most of the major electrical equipment shown on the single line diagram such as transformers, switchgear, MCC's and RVSS, etc. and historical pricing for the overhead powerline. Medium and low voltage electrical distribution not shown on the single line diagram was estimated as a percentage of the cost of major electrical equipment for which budgetary quotes were received.

21.3.6 Phoenix Decommissioning Sustaining Capital Cost Details

Uranium mining companies in Saskatchewan are required by the Saskatchewan Ministry of Environment (SMOE) and the Canadian Nuclear Safety Commission (CNSC) to develop decommissioning and reclamation plans, including financial surety. These requirements are stated in Section 12 of The Mineral Industry Environmental Protection Regulations, 1996 and Section 3 of the General Nuclear Safety and Control Regulations (Section 3(1)(l) requires a description of any proposed financial guarantee).

The CNSC and SMOE have advised uranium mining companies that the requirements of both sets of regulations will avoid any duplication of financial assurances and therefore call for a common report on the technical description and the cost evaluation of the future decommissioning activities. The cost estimates provide the basis for a financial assurance, which would be used by the land owner in the event the mining company was unable to carry out its commitment to decommission and reclaim the facility. Financial assurance is typically provided via financial guarantee letter from approved financial institutions on behalf of the proponent.

The development of the Wheeler River project will be carried out using modern technology, minimizing surface disturbance and incorporate eventual reclamation into the initial design phases. The decommissioning design and estimate were prepared by SRK and costs are based on a mix of first principles and SRK's cost database. Table 21-9 provides a summary of the decommissioning costs for Phoenix.

Table 21-9. Phoenix Decommissioning Capital Cost Details

Decommissioning Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Stockpiles and Pads		$ 3.8	$ 3.84
Ponds		$ 5.6	$ 5.63
Infrastructure		$ 384	$ 384.48
Roadways		$ 40	$ 40.13
Other		$ 118	$ 118.00
Wells		$ 839	$ 838.90
Restoration Water Treatment Plant	$ -	$ 26,063	$ 26,062.7
Sub Total	**$ -**	**$ 27,454**	**$ 27,454**

21.4 Gryphon Capital Cost Summary

The capital costs for the underground mining of the Gryphon deposit are shown in Table 21-10.

Table 21-10. Gryphon Capital Cost Summary

Gryphon Capital Cost Details (1,000's)			
Direct Capital Costs	**Initial**	**Sustaining**	**Total**
Shafts	$ 131,522	$ -	$ 131,522
Surface Facilities	$ 46,932	$ 6,074	$ 53,006
Underground	$ 49,518	$ 68,842	$ 118,360
Utilities	$ 3,946	$ 263	$ 4,209
Electrical	$ 3,613	$ -	$ 3,613
Civil & Earthworks	$ 11,791	$ 483	$ 12,274
McClean Mill Upgrades	$ 49,920	$ -	$ 49,920
Offsite Infrastructure	$ 32,392	$ -	$ 32,392
Decommissioning	$ -	$ 1,575	$ 1,575
Total Direct Costs	**$ 329,634**	**$ 77,236**	**$ 406,871**
Indirect Costs	$ 142,015	$ 5,112	$ 147,127
Other (Owner's) Costs	$ 28,143		$ 28,143
Contingency Costs	$ 123,328	$ 394	$ 123,722
Total Costs	**$ 623,120**	**$ 82,743**	**$ 705,863**

Further breakdown of the direct capital costs is provided in Figure 21-3. More consistent with remote projects, indirect and owner’s costs make up a larger portion of the total initial capital costs at 27%. Due to the overlap in Gryphon construction and Phoenix operations, many of the indirect costs (camp, flights, etc.) are shared between the sites allowing for some economies of scale efficiencies.

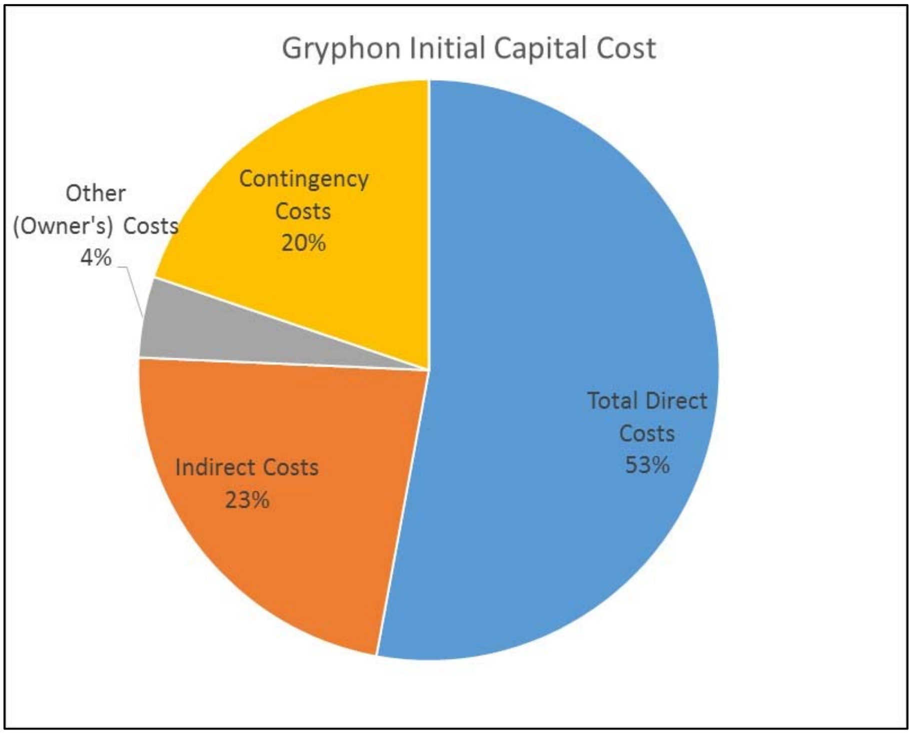


Figure 21-3. Gryphon Initial Capital Cost Distribution

As shown in Figure 21-4 shaft excavation and furnishing along with costs of the surface facilities associated with the shaft (i.e. headframe) account for over 40% of the direct costs for Gryphon.

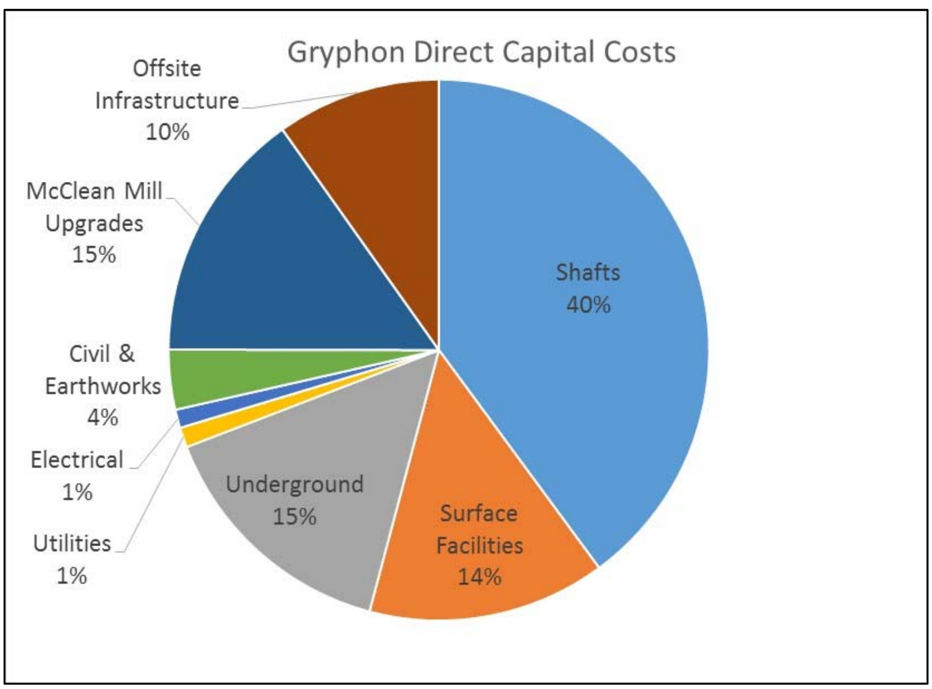


Figure 21-4. Gryphon Direct Capital Cost Distribution

The design and cost basis for the shaft, underground development, infrastructure and McClean Mill upgrades are provided in Table 21-11.

Table 21-11. Gryphon Key Capital Cost Estimate Details

AREA	Design Basis	Cost Basis
Shaft Excavation and lining	Engineering design criteria, General Arrangement drawings	Contractor Quotation
Production and ventilation shaft Equipping	General Arrangement Drawings	First principles productivity, quotations, factored previous similar project cost buildups, Stantec database costs
Fresh Air and Return air ventilation plants	Engineering Design Criteria	Quotation, previous similar project cost quotation, Cost build-ups factored as required
Backfill Plant	Engineering Design Criteria	Vendor provided quote and allowances
Underground Development	Engineered 3D mine layouts, quantity take offs from drawings	First principles, quotations for equipment and materials, benchmark against existing operations
Underground Infrastructure	General Arrangement Drawings, Engineering design criteria	Factored previous project buildups and Stantec database costs
McClean Mill Upgrades	Derived from metallurgical mass balance, comparison to existing circuit capacities	A factored estimated was prepared by mill area based on a detailed mechanical equipment list. Major equipment was priced based on a combination of budget quotations and in-house data. Installation hours and labour rates were applied to calculate a total mechanical installed cost. Discipline costs (Piping, Electrical, Instrumentation etc.) were then applied by area based on past experience and industry best practice. Standard allowances were then applied for indirect costs to arrive at a total installed cost. Labour productivities consistent with previous mill upgrades were applied.

The shaft works make up a large proportion of the capital costs. One of the advantages of the Blind Boring technique is that it comes from the civil industry and is more typically priced on lump sum contracts that cities and municipalities prefer. In adopting this technology into the Wheeler project, the company aims to reduce cost escalation risk (along with other shaft sinking risks) for the Gryphon project.

One of the key items required for Gryphon will be the construction of the approximately 50 km extension of Highway 914 to connect the McArthur River and the Cigar Lake mines. This new section of highway will reduce the haul distance from Wheeler River to the McClean Lake mill from 972 km to approximately 160 km. In 2012 the province of Saskatchewan announced its support for infrastructure expansion in the north in its “Saskatchewan Plan for Growth – Vision 2020 and Beyond” and will be required to continue to support this aspect.

21.4.1 Gryphon Production and Ventilation Shaft Capital Cost Details

The capital cost detail for the shafts at Gryphon is provided in Table 21-12.

Table 21-12. Gryphon Production and Ventilation Shaft Capital Cost Details

Shaft Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Production Shaft Headworks	$ 20,112	$ -	$ 20,112
Production Shaft Construction	$ 9,989	$ -	$ 9,989
Production Shaft Excavation	$ 7,780	$ -	$ 7,780
Production Shaft Loading Pocket	$ 2,122	$ -	$ 2,122
Production Shaft Grizzly	$ 1,969	$ -	$ 1,969
Ventilation Shaft Headworks	$ 6,104	$ -	$ 6,104
Ventilation Shaft Construction	$ 6,923	$ -	$ 6,923
Blind Boring	$ 76,524	$ -	$ 76,524
Total Costs	**$ 131,522**	**$ -**	**$ 131,522**

The Blind Boring estimate was prepared by Frontier Kemper (FK). FK is a civil and building construction company offering diversified general contracting and design/build services including tunnelling and mining throughout the world. FK's estimate is based on the following general considerations:

- Mobilization and demobilization;
- Supply of all construction equipment;
- FK personnel transportation to Saskatoon;
- All required freight for equipment and materials to project site;
- Detailed engineering and design services and procurement services;
- Slurry plant site preparation, installation and operation;
- Collar pads for both shafts;
- Collar pad supports and coping sections;
- Blind drill mobilization, erection, operation dismantling and demobilization;
- Blind drilling of production shaft to 6.1m diameter x 570m deep;
- Blind drilling of ventilation shaft to 5.8m diameter x 508m deep;
- Fabrication, transportation and installation of composite liner segments at the site; and
- Placing of annular grout.

The costs of equipping and commissioning the two shafts was completed by Stantec. Following blind boring on each shaft, a suitable headframe / collar house will be constructed on top of the existing foundations utilized for blind boring. The hoist / winch houses will be constructed prior to completion of the shaft blind boring. Shafts will be equipped in a manner similar to a conventional shaft sinking arrangement utilizing fit for purpose suspended work decks.

The production shaft equipping is completed with the permanent 10-foot double drum hoist and the contractors temporary winch setup. Once the production shaft equipping was completed to the 500 Level station, a temporary skip loading system is set up for initial off shaft development. The off-shaft development is completed with handheld drills, slushers and a 2yd LHD. Excavations included roughly 66 meters of development to allow the first set of development gear (jumbo, 6yd LHD and scissor deck) to be mobilized.

The loading pocket / ore and waste pass is developed in a similar manner to the 500 Level station using handheld drills, slushers and temporary skip loading system. The ore and waste pass will be driven with an Alimak up to the grizzly location. Once the loading pocket and grizzly construction is completed development and hoisting using fully mechanized equipment can be initiated.

The ventilation shaft is equipped with an egress auxiliary cage running off a single drum hoist. Only a minor amount excavation is required here as the development from the production shat will break into the vent shaft.

Following equipping, the two shafts are commissioned for normal operations. The cost estimate completed by Stantec, was complete from a first principles work-up starting with crew size and expected productivities. Costs for headworks, hoist houses, shaft equipping, and off shaft development and construction are based on quotations and previous project buildups factored as required to suit this scope

21.4.2 Gryphon Surface Facilities Capital Cost Details

A breakdown of the costs for the surface facilities located at Gryphon are provided in Table 21-13.

Table 21-13. Gryphon Surface Facilities Capital Cost Details

Surface Facilities Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Ventilation	$ 4,842	$ -	$ 4,842
Capital Purchases - FA & RA Fans	$ 5,859	$ 763	$ 6,622
Water Treatment	$ 17,025	$ 115	$ 17,139
Mine Rescue / Office Building	$ 178	$ -	$ 178
Ore and Waste Storage	$ 18,891	$ -	$ 18,891
Fenced Storage	$ 58	$ -	$ 58
Explosives Storage	$ 79	$ -	$ 79
Backfill Plant	$ -	$ 5,180	$ 5,180
Gryphon Allocation	$ -	$ 16	$ 16
Total Costs	**$ 46,932**	**$ 6,074**	**$ 53,006**

Cost estimates for these elements were completed by obtaining supplier quotes and from Stantec's historical cost database from past projects.

21.4.3 Gryphon Underground Capital Cost Details

Gryphon underground capital cost details are provided in Table 21-14.

Table 21-14. Gryphon Underground Capital Cost Details

Underground Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Ramp Development	$ 7,459	$ 9,416	$ 16,875
Lateral Waste Development	$ 11,292	$ 16,115	$ 27,407
Vertical Development	$ 5	$ 9,211	$ 9,215
Underground Infrastructure	$ 14,433	$ 17,262	$ 31,695
Capital Purchases – Equipment	$ 2,874	$ 1,152	$ 4,026
Capital Purchases - Mobile Equipment	$ 12,941	$ 15,572	$ 28,513
Diamond Drilling / Boreholes	$ 514	$ 114	$ 628
Sub Total	**$ 49,518**	**$ 68,842**	**$ 118,359**

Labour rates were provided by Denison for company personnel. Contractor labour rates are first principle buildups from previous projects that are benchmarked against current contractor rates. Underground development costs were estimated by completing a first principles work-up of crew size, equipment requirements, quantities of consumables and expected productivities. Unit cost for materials and supplies are from Stantec's database. Dimensions, expected geotechnical conditions and ground support requirements were all considered for each different type and geometry of excavation.

All vertical development will be completed using raiseboring methods and the costs are based on first principle build ups benchmarked against recent bids from an underground raise boring contractor. Infrastructure costs are based on Stantec's cost database of recent similar projects and factored as required.

The majority of the capital purchases are quoted specifically for this project. Mobile equipment capital costs are sources from an existing executed project as well as previous project quotations. Diamond drilling costs are provided by Denison based on recent quotations which also include assaying costs.

A summary of the ramp and lateral development rates and performances is provided in Table 21-15.

Table 21-15. Gryphon Lateral Development Rates

Description	Ramp Development Off Shaft 4.5m W x 5m H	Ramp at Orebody Development 4.5m W x 5m H	Capital Access Development 4.5m W x 5m H	Operating Access Development 4m W x 4m H	Ore Sill Development 5.9m W x 4.0m H
Single Heading 4-man crew	$4,845	$3,961	$4,096	$3,567	$5,536
Performance (m/day)	3.00	3.70	3.70	4.00	2.60
Double Heading 4-man crew	$4,295	$3,518	$3,653	$3,154	$4,888
Performance (m/day)	3.90	4.80	4.80	5.20	3.40
Multiple Heading 5-man crew	$4,244	$3,490	$3,625	$3,130	$4,859
Performance (m/day)	5.00	6.10	6.10	6.60	4.30

21.4.4 Gryphon Underground Mobile Equipment

Table 21-16 lists the underground mobile equipment purchase and sustaining costs.

Table 21-16. Gryphon Underground Mobile Equipment Capital Cost Details

Gryphon Underground Mobile Equipment Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
2 Boom Jumbo	$ 3,234	$ 1,393	$ 4,627
2.5 yd with Forks	$ -	$ 893	$ 893
30t Trucks	$ 1,310	$ 2,619	$ 3,929
Bolters	$ 2,787	$ -	$ 2,787
Boom Trucks	$ -	$ 554	$ 554
Compressor	$ -	$ -	$ -
Fan Handler	$ 179	$ -	$ 179
Fork Lifts	$ -	$ 179	$ 179
Graders	$ 1,618	$ 759	$ 2,377
ITH with Compressor	$ 2,110	$ 1,618	$ 3,728
LHD	$ -	$ 2,110	$ 2,110
Mobile equipment sustaining capital	$ -	$ 3,918	$ 3,918
Portable Refuge Station	$ 1,016	$ -	$ 1,016
Scissor Lifts	$ 279	$ 508	$ 787
Shotcrete Equipment	$ 407	$ 477	$ 884
Toyota Jeeps	$ -	$ 272	$ 272
Roadbed Crusher	$ -	$ 371	$ 371
Tractors	$ -	$ -	$ -
Sub Total	**$ 12,941**	**$ 15,671**	**$ 28,612**

Underground mobile equipment capital costs are sourced from an existing executed project as well as previous project quotations and Stantec database costs.

21.4.5 McClean Lake Mill Upgrades Capital Cost Details

The processing of Wheeler River ore at the McClean Lake Mill is advantageous to the project as many of the circuits required for processing are already in place with available capacity. Moreover, all of the ancillary facilities and infrastructure to support processing are available, in addition to the facility having the required licensed production capacity.

A factored estimated was prepared by mill area based on a detailed mechanical equipment list. Major equipment was priced based on a combination of budget quotations and in-house data. Installation hours and labour rates were applied to calculate a total mechanical installed cost. Discipline costs (piping, electrical, instrumentation etc.) were then applied by area based on past experience and industry best practice. Standard allowances were then applied for indirect costs to arrive at a total installed cost.

Section 17 (Recovery Methods) outlines the major equipment modifications required to the McClean Lake Mill to process Gryphon ore. Table 21-17 outlines the capital costs by area to implement the required modifications, as discussed in Section 17.

Table 21-17. McClean Lake Mill Upgrades Capital Cost Details

McClean Lake Mill Upgrades Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Grinding	$ 1,110	$ -	$ 1,110
Leaching	$ 24,250	$ -	$ 24,250
CCD	$ 7,947	$ -	$ 7,947
Precipitation	$ 1,243	$ -	$ 1,243
CX	$ 6,378	$ -	$ 6,378
Acid Plant	$ 5,768	$ -	$ 5,768
Reagents	$ 683	$ -	$ 683
Tailings	$ 2,540	$ -	$ 2,540
Sub Total	**$ 49,920**	**$ -**	**$ 49,920**

21.4.6 Gryphon Decommissioning

As with Phoenix, the decommissioning design and estimate were prepared by SRK and the costs are based on a mix of first principle estimation and SRK's cost database. Table 21-18 provides a summary of the decommissioning costs for Gryphon.

Table 21-18. Gryphon Decommissioning Capital Cost Details

Decommissioning Capital Cost Details (1,000's)			
Area	**Initial**	**Sustaining**	**Total**
Stockpiles and Pads	$ -	$ 401	$ 401
Ponds	$ -	$ 268	$ 268
Infrastructure	$ -	$ 386	$ 386
Roadways	$ -	$ 21	$ 21
Environmental Monitoring*	$ -	$ 315	$ 315
Other	$ -	$ 62	$ 62
Opening Closures	$ -	$ 124	$ 124
Sub Total	**$ -**	**$ 1,575**	**$ 1,576**

21.4.7 Other Capital Costs

Two feasibility studies will be completed. The Phoenix feasibility study is to be initiated in 2019 to complete associated field work and to support the completion of Environmental Assessment work. The second study will focus on the Gryphon project and will commence approximately one year before the commencement of production at Phoenix. The project's future work plan is provided in Table 21-19

Table 21-19. Wheeler River Future Work Plan

Future Work Plan	
Environmental Assessment and other studies	$ 5,000,000
Phoenix Feasibility Study and other studies	$ 5,275,000
Gryphon Feasibility Study and other studies	$ 11,275,000
Wheeler River Grand Total	**$ 21,550,000**

These capital costs are not included in the NPV calculations for the project. Table 21-20 provides the project's capital schedule.

Table 21-20. Wheeler River Capital Schedule

Capital Area	Total	Initial	Sustaining	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036-2043
Phoenix																			
Wellfield	$ 99,076	$ 63,674	$ 35,402		$ 49,295	$ 11,897	$ 4,177	$3,446	$ 3,456	$ 3,660	$ 3,470	$ 3,460	$ 3,667	$ 3,457	$ 3,657	$ 3,495	$ 1,939		
ISR Precipitation Plant	$ 55,541	$ 50,935	$ 4,606		$ 8,904	$ 33,750	$ 8,282		$ 14	$ 1,513	$ 14		$ 1,526		$ 14	$ 1,513	$ 14		
Water Treatment	$ 19,944	$ 1,268	$ 18,676				$ 1,267										$18,677		
Surface Facilities	$ 22,374	$ 22,325	$ 49		$ 1,320	$ 21,005				$ 22			$ 21			$ 7			
Utilities	$ 7,341	$ 6,538	$ 803		$ 2,179	$ 4,359			$ 5	$ 348	$ 5		$ 328		$ 2	$ 115	$ 1		
Electrical	$ 18,834	$ 18,834	$ -	$ 1,552	$ 8,688	$ 8,594													
Civil & Earthworks	$ 45,640	$ 44,309	$ 1,331		$ 18,395	$ 25,914			$ 363		$ 363		$ 336		$ 185		$ 83		
Offsite Infrastructure	$ 7,950	$ 7,950	$ -	$ 7,950															
Decommissioning	$ 27,454	$ -	$ 27,454															$ 6,560	$20,894
Subtotal	**$ 304,154**	**$ 215,833**	**$ 88,321**	**$ 9,502**	**$ 88,781**	**$105,519**	**$13,727**	**$3,446**	**$ 3,837**	**$ 5,542**	**$ 3,851**	**$ 3,460**	**$ 5,878**	**$ 3,457**	**$ 3,858**	**$ 5,129**	**$20,714**	**$ 6,560**	**$20,894**
Indirect Costs	$ 33,957	$ 28,288	$ 5,669	$10,794	$ 7,684	$ 6,768	$ 3,042	$ 25	$ 99	$ 620	$ 99	$ 25	$ 728	$ 22	$ 168	$ 393	$ 2,983	$ 505	
Owner's Costs	$ 14,227	$ 14,277	$ -	$ -	$ 4,571	$ 6,109	$ 3,547												
Contingency	$ 73,611	$ 64,190	$ 9,421	$ 4,677	$ 25,259	$ 29,599	$ 4,655										$ 2,432	$ 1,766	$ 5,223
Phoenix Total	**$ 425,949**	**$ 322,588**	**$ 103,411**	**$24,973**	**$126,295**	**$147,995**	**$24,970**	**$3,471**	**$ 3,936**	**$ 6,163**	**$ 3,951**	**$ 3,485**	**$ 6,606**	**$ 3,479**	**$ 4,026**	**$ 5,522**	**$26,129**	**$ 8,831**	**$26,117**
Gryphon																			
Shafts	$ 131,522	$ 131,522	$ -						$13,905	$ 42,634	$ 46,018	$ 26,979	$ 1,987						
Surface Facilities	$ 53,006	$ 46,932	$ 6,074						$ 254	$ 1,484	$ 40,659	$ 4,535	$ 2	$ 5,943	$ 57	$ 15		$ 57	
Underground	$ 118,360	$ 49,518	$ 68,842									$ 13,559	$ 36,144	$29,058	$16,245	$ 8,765	$ 6,799	$ 5,264	$ 2,525
Utilities	$ 4,209	$ 3,946	$ 263								$ 3,946		$ 25		$ 2	$ 232	$ 4		
Electrical	$ 3,613	$ 3,613	$ -						$ 903	$ 2,710									
Civil & Earthworks	$ 12,274	$ 11,791	$ 483						$11,791				$ 25		$ 178		$ 279		
McClean Lake Mill	$ 49,920	$ 49,920	$ -								$ 1,940	$ 12,093	$ 35,887						
Offsite Infrastructure	$ 32,392	$ 32,392	$ -									$ 8,098	$ 24,294						
Decommissioning	$ 1,575	$ -	$ 1,575																$ 1,575
	$ 406,871	**$ 329,634**	**$ 77,237**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$26,853**	**$ 46,827**	**$ 92,563**	**$ 65,264**	**$ 98,364**	**$35,001**	**$16,483**	**$ 9,012**	**$ 7,082**	**$ 5,321**	**$ 4,101**
Indirect Costs	$ 147,127	$ 142,015	$ 5,112					$4,367	$34,339	$ 30,432	$ 27,656	$ 18,705	$ 26,532	$ 1,734	$ 1,262	$ 699	$ 547	$ 479	$ 376
Owner's Costs	$ 28,143	$ 28,143	$ -			$ -	$ -	$ -	$ 398	$ 3,353	$ 4,537	$ 7,803	$ 12,052						
Contingency	$ 123,722	$ 123,328	$ 394			$ -	$ -	$1,092	$15,397	$ 20,153	$ 31,189	$ 22,538	$ 32,959	$ -	$ -	$ -	$ -	$ -	$ 394
Gryphon Total	**$ 705,863**	**$ 623,120**	**$ 82,743**	**$ -**	**$ -**	**$ -**	**$ -**	**$5,459**	**$76,987**	**$100,765**	**$155,944**	**$114,310**	**$169,908**	**$36,735**	**$17,745**	**$ 9,710**	**$ 7,630**	**$ 5,800**	**$ 4,870**
Wheeler Grand Total	**$ 1,131,812**	**$ 945,708**	**$ 186,154**	**$24,973**	**$126,295**	**$147,995**	**$24,970**	**$8,929**	**$80,923**	**$106,928**	**$159,895**	**$117,795**	**$176,514**	**$40,214**	**$21,771**	**$15,232**	**$33,759**	**$14,631**	**$30,988**

21.5 Operating Costs

Operating costs are estimated for the 14-year mine production period from July 1, 2024 through to March 31, 2037. Phoenix mine production is scheduled from July 1, 2024 to June 30, 2034 at and Gryphon mine production is scheduled from September 1, 2030 to March 31, 2037. Table 21-21 presents a summary of the Wheeler River prefeasibility level operating cost estimates and total estimated sales. For Gryphon operations, it is assumed that the McClean Lake mill is processing a maximum of 9M lbs per year from Gryphon and 15M lbs per year from Cigar Lake.

Table 21-21. Wheeler River Operating Cost Summary

Cost Area	Phoenix		Gryphon		Total Cost
	$000's	$/lb U_3O_8	$000's	$/lb U_3O_8	$000's
Mining	$ 44,020	$ 0.75	$ 266,202	$ 5.45	$ 310,222
Milling	$ 115,577	$ 1.97	$ 412,621	$ 8.45	$ 528,198
Transport to Convertor	$ 12,341	$ 0.21	$ 10,252	$ 0.21	$ 22,593
Site Support / Administration	$ 82,264	$ 1.40	$ 53,346	$ 1.09	$ 135,610
Total	**$ 254,202**	**$ 4.33**	**$ 742,421**	**$ 15.21**	**$ 996,623**
Total USD		**$ 3.33**		**$ 11.70**	
U_30_8 Sales - lbs in 000's	**58,767**		**48,817**		

The operating cost estimates for mining and milling are quite different for the two deposits due to the different mining methods involved and the different processing facilities required as a result – each of these items will be discussed in more detail below.

Transport to Convertor costs represent the haulage of processed U_30_8 product from the mill processing facility to the point of sale at the conversion site. These costs have been estimated using a cost factor of $0.21 per lb U_30_8.

Site support and administration costs represent the cost of maintaining the shared surface infrastructure at Wheeler River as well as putting in place the appropriate management and functional personnel to run the operation – these items are discussed in more detail below.

21.5.1 Phoenix Wellfield Operating Cost Details

The Phoenix mining operating cost estimate consists of: a) labour and materials to maintain the ground freezing equipment, injection and recovery pumps and piping systems; b) the electrical power consumption to run the freeze plants, injection and recovery well pumps; and c) annual costs to carry-out surface water treatment operations on-site. Table 21-22 identified the costs. The costs are based mostly on first principle estimation with percentage allowances for maintenance costs

Table 21-22. Phoenix Wellfield Operating Cost Details

AREA	$000's	$/lb U_3O_8
Freeze Plant, Wellfield Operations		
Maintenance	$ 1,611	$ 0.03
Electricity	$ 36,616	$ 0.62
Labour	$ 5,335	$ 0.09
Water Treatment Operations	$ 458	$ 0.01
Total	**$ 44,020**	**$ 0.75**

21.5.2 Phoenix Plant Operating Cost Details

The Phoenix Process Plant Operating cost estimate includes: a) the labour, materials, electrical power consumption and reagents required to operate the ISR processing plant for the duration of the Phoenix mining operation; and b) the labour, materials and equipment required to perform ongoing maintenance of the plant. Table 21-23 presents a summary of the Phoenix prefeasibility level process plant operating cost estimate (also referred to as the milling estimate).

Table 21-23. Phoenix Precipitation Plant Operating Cost Details

AREA	$000's	$/lb U_3O_8
Processing Plant		
Maintenance	$ 6,894	$ 0.12
Electricity	$ 6,803	$ 0.12
Reagents	$ 56,183	$ 0.96
Labour	$ 45,697	$ 0.78
Total	**$ 115,577**	**$ 1.97**

21.5.3 Gryphon Mine Operating Cost Details

The Gryphon Mining Operating cost estimate consists of three main elements: a) underground mining operation costs; b) Surface facility operating costs for dedicated Gryphon mining facilities; and c) haulage costs to transport ore from Wheeler to McClean Lake for processing.

Underground mining operation costs consist of direct labour, materials and equipment costs for lateral waste development, ore silling, production and diamond drilling/borehole activities. Costs also include indirect labour, power, heat, freight, taxes and indirect plant / mobile equipment operating and maintenance supplies.

Ore haulage transportation costs cover the cost of shipping Gryphon ore from Wheeler River to the JEB Mill at McClean Lake for processing. A budget quote was obtained from Northern Resource Trucking to haul the ore. Table 21-24 presents a summary of the Gryphon prefeasibility level mining operating cost estimate.

Table 21-24. Gryphon Mine Operating Cost Details

AREA	$000's	$/lb U_3O_8	$ / Tonne
Underground Mining Operations			
Lateral Waste Development	$ 2,138	$ 0.04	$ 1.70
Ore Silling	$ 53,933	$ 1.10	$ 42.89
Production	$ 54,716	$ 1.12	$ 43.51
Diamond Drilling / Boreholes	$ 3,599	$ 0.07	$ 2.86
Freight	$ 2,274	$ 0.05	$ 1.81
Owner's Costs			
Indirect Labour	$ 63,000	$ 1.29	$ 50.10
Power	$ 21,656	$ 0.44	$ 17.22
Plant / Equip. Maintenance and Supplies	$ 14,093	$ 0.29	$ 11.21
Propane for Mine Air Heating	$ 9,989	$ 0.20	$ 7.94
Subtotal	**$ 225,398**	**$ 4.62**	**$ 179.25**
Surface Facilities			
Water Treatment	$ 2,114	$ 0.04	$ 1.68
Mine Rescue	$ 155	$ 0.00	$ 0.12
Other			
Ore Haulage to McClean Lake	$ 38,535	$ 0.79	$ 30.65
Total	**$ 266,202**	**$ 5.45**	**$ 211.70**

21.5.4 McClean Mill Operating Cost Details

The Gryphon Milling Operating Cost Estimate is based on two items. The first is McClean Lake Mill Processing Costs which are the estimated operating costs for processing Gryphon ore at McClean Lake. These costs estimated by Hatch utilize an assumed basis for allocation of common processing costs in the mill between Wheeler River and the McClean Lake Joint Venture. Table 21-25 provides allocation details. The second component of the mill operating costs are toll milling fees which include any tailings storage fees. These items were derived by Denison and are based on similar fees for toll milling operations in northern Saskatchewan and estimates of costs to store mill tailings associated with the Gryphon ore feed. Table 21-26 presents a summary of the McClean Mill operating cost details.

Table 21-25. McClean Lake Cost Allocation

Operating Cost Component	Basis
Power	• Current site electrical costs were used as a baseline. • New electrical equipment for each processing option were summarized into an electrical load list. The load list was used to generate the additional operating costs by each circuit for each option. • 100% of grinding costs to Wheeler River ores, 100% of slurry receiving costs to CLJV • Other costs were split between CLJV and Wheeler River by circuit based on tonnage (Leaching, CCD, Tailings) or uranium throughput (all others).
Labour	• Current labour costs used as a baseline. • New labour costs were estimated by circuit for each processing option. • 100% of grinding costs to Wheeler River, 100% of slurry receiving costs to CLJV. • Other costs were split between CLJV and Wheeler River by circuit based on tonnage (Leaching, CCD, Tailings) or uranium throughput (all others).
Maintenance	• Current maintenance costs used as a baseline. • New maintenance costs estimated based on a factor of the capital cost of new equipment. New total maintenance cost calculated. • 100% of grinding costs to Wheeler River, 100% of slurry receiving costs to CLJV. • Other costs were split between CLJV and Wheeler River by circuit based on tonnage (Leaching, CCD, Tailings) or uranium throughput (all others).
Consumables	• Consumables costs for major reagents were developed from plant model consumption rates based on the process design requirements. • Consumables costs estimated independently for each ore source. • Current site consumables costs were used. • For common use consumables, costs were split between CLJV and Wheeler River based on tonnage or uranium production as applicable.
General & Administrative (G&A)	• Current JEB G&A costs used as a baseline. • Increase of 5% to account for additional metallurgical accounting and accounting of two ore sources. • New total cost then split between CLJV and Wheeler River based on uranium throughput.

Table 21-26. McClean Mill Operating Cost Details

AREA	$000's	$/lb U_3O_8	$ / Tonne
McClean Lake Mill Processing Cost			
Power	$ 11,434	$ 0.23	$ 9.09
Labour	$ 66,117	$ 1.33	$ 52.58
Maintenance	$ 13,422	$ 0.27	$ 10.67
Consumables	$ 100,916	$ 2.03	$ 80.25
Indirects / Site Support / Admin	$ 91,470	$ 1.84	$ 72.74
Subtotal	**$ 283,359**	**$ 5.71**	**$ 225.34**
Other Processing Costs			
Toll Milling / TMF Fees	$ 128,764	$ 2.59	$ 102.40
Total	**$ 412,123**	**$ 8.45**	**$ 327.74**

21.6 Site Support and Administration Cost Details

Table 21-27 presents a summary of the Wheeler River PFS site support and administration operating cost for all operations. These costs have been estimated for the 14-year production period and have been allocated between the two deposits as follows:

- F'2024 to F'2029 – annual costs allocated 100% to Phoenix;
- F'2030 to F'2034 – annual costs allocated pro-rata to Phoenix and Gryphon based on mine production as measured by lbs U_3O_8; and
- F'2035 to F'2037 – annual costs allocated 100% to Gryphon

Table 21-27. Wheeler River Site Support and Administration Operating Cost Details

Cost Area	Phoenix		Gryphon		Total Cost
	$000's	$/lb U_3O_8	$000's	$/lb U_3O_8	$000's
Site Support					
Maintenance	$ 3,212	$ 0.05	$ 1,898	$ 0.04	$ 5,110
Electricity	$ 4,460	$ 0.08	$ 2,634	$ 0.05	$ 7,094
Heat	$ 643	$ 0.01	$ 380	$ 0.01	$ 1,023
Administration					
Labour	$ 31,814	$ 0.54	$ 20,626	$ 0.42	$ 52,440
Camp and Flights	$ 26,435	$ 0.45	$ 17,625	$ 0.36	$ 44,060
Miscellaneous	$ 15,700	$ 0.27	$ 10,183	$ 0.21	$ 25,883
Total	**$ 82,264**	**$ 1.40**	**$ 53,346**	**$ 1.09**	**$ 135,610**

Phoenix and Gryphon Site Support cost estimates include the materials, fuel, electrical power consumption and propane associated with operating and maintaining the shared facilities and utilities on site including:

- Operations centre;
- Permanent camp;
- Wash bay / scanning building;
- Fenced and outdoor covered storage;
- Security and gatehouse;
- Incinerator;
- Fuel systems;
- Sewage collection and treatment;
- Potable water treatment and distribution;
- Communication systems;
- Fire water systems;
- Airstrip and terminal building; and
- Site roads.

The administration cost estimates include management and administration labour, camp and flight costs, and miscellaneous cost items.

Management and functional group labour is estimated from first principles and includes:

- Management Administration – General Manager, Administrative Assistants;
- Functional Administration – Human Resources Support (Co-Ordinator, Payroll & Benefits Administration) / Information Technology Support / Finance Support (Controller, Accounts Payable);
- Logistics / Services Team – Logistics, Warehousing, Site Services, Security;
- Health, Safety, Environment and Community Team ("HSEC") - HSEC Superintendent, HS Coordinators, Environmental Engineer, Environmental Techs, Community Relations, Water Treatment Operator, Trainers;
- Site technical support (assay lab, engineering and technical staff); and
- Procurement and Contracts Group - Procurement Superintendent, Buyers

Camp and Flights costs are estimated as follows:

- Camp operating costs were based on a budget quote from Athabasca Catering. Daily camp costs vary according to the number of people in camp, including Denison staff listed above, plus operations, maintenance, technical, and mining personnel on site, camp staff and a 5% allowance for occasional visitors; and
- Flight costs were based on personnel's turn around schedule and a flight cost of $391 per person round trip flight.

Miscellaneous costs are based on a combination of allowances, historical cost databases and first principles determination and include:

- Mining and Surface Leases;
- CNSC Fees;
- Insurance;
- Operations Complex Office Supplies;
- Operation Complex Laundry & Dry Supplies; and
- Emergency Response Supplies.

22 Economic Analysis

The Wheeler River Joint Venture ("WRJV") is a joint venture between Denison Mines Inc. ("DMI") (90% and operator) and JCU (Canada) Exploration Company Limited ("JCU") (10%). The WRJV, which owns the Wheeler River project, is a joint venture and is not itself a taxable entity. Instead each joint venture partner reports its share of the joint venture operations in its own tax return. As each JV partner has a unique tax profile, the Wheeler River project has been evaluated using two different cash flow model approaches:

- A pre-tax discounted cash flow model (Table 22-1) shows the economics of the project on a 100% basis and excludes tax specific items related to Canadian Federal and Provincial income taxes and Saskatchewan profit based royalties, each of which will vary depending on each joint venture participants unique facts and circumstances; and
- A post-tax discounted cash flow model, specific to Denison (Section 22.6), which shows the economics of the project on a 63.30% basis (Denison's interest in the Wheeler project as of September 24, 2018) and a on 90% pro-forma basis (Denison's interest in the Wheeler River project as of October 29, 2018) which includes tax specific items related to Canadian Federal and Provincial income taxes and Saskatchewan profit-based royalties and other non-tax related items which are unique and applicable to Denison's economic interest in the Wheeler River project.

22.1 Input and Assumptions

Inputs to both the pre-tax and post-tax cash flow models include:

- An estimated 3.5-year pre-production period from January 2021 to June 2024;
- Life of mine production of 1.399 million tonnes at an average grade of 3.55% U_3O_8 containing 109.374 million lbs of U_3O_8;
- A project mine production period of approximately 14 years from July 2024 to March 2037 comprised of: a) Phoenix mine production from July 2024 to June 2034 at an average production rate of 5.966 million lbs U_3O_8 per year; and b) Gryphon mine production from September 2030 to March 2037 at an average production rate of 7.648 million lbs U_3O_8 per year;
- Estimated metallurgical process uranium recoveries of 98.5% and 98.2% for Phoenix and Gryphon mill feeds, respectively;
- A Base case uranium pricing scenario as follows: a) Phoenix – based on UxC's Q3-2018 Uranium Market Outlook Report Composite Midpoint spot price projection, in constant dollars, ranging from USD$29.48 to USD$45.14 per pound U_3O_8 during the Phoenix mine production period, translated to CAD using an exchange rate of 1.30 CAD/USD; b) Gryphon – based on a price of USD$50.00 per pound during the Gryphon mine production period, translated to CAD using an exchange rate of 1.30 CAD/USD;
- Project capital costs of $1,131.813 million (100% basis) as shown in Table 21-1;
- Project operating costs of $996.623 million (100% basis) as shown in Table 21-21; and
- Saskatchewan revenue-based royalties and surcharges applicable to uranium revenue, as follows: a) a basic royalty of 5.0% of uranium revenue; b) a resource credit of 0.75% of uranium revenue (which partially offsets the basic royalty); and c) a resource surcharge of 3.0% of the value of uranium revenue.

No inflation or escalation of revenue or costs has been incorporated.

22.2 Canadian Royalties Applicable to Wheeler River

The province of Saskatchewan imposes royalties on the sale of uranium extracted from ore bodies in the province in accordance with Part III of The Crown Mineral Royalty Regulations (the "Regulations") pursuant to The Crown Minerals Act (the "Act"). The uranium royalty regime currently in effect in Saskatchewan has three components:

i. Basic Royalty: Computed as 5% of gross revenues derived from uranium extracted from ore bodies in the province;
ii. Resource Credit: Reduction in the basic royalty equal to 0.75% of gross revenues derived from uranium extracted from ore bodies in the province; and
iii. Profit Royalty: Computed as 10% to 15% of net profits derived from the mining and processing of uranium extracted from ore bodies in the province.

Under the current system, each owner or joint venture participant in a uranium mine is a royalty payer. Individual interests are consolidated on a corporate basis for the computation and reporting of royalties due to the province.

Gross revenue, for purposes of the Basic Royalty and Resource Credit, is determined in accordance with the Regulations and allows for reductions based on specified allowances. In computing gross revenue Denison has included the transport to convertor cost as a specified allowance allowed under the Regulations. Net profit, for the Profit Royalty, is calculated based on the recognition of the full dollar value of a royalty payer's production, exploration, capital and decommissioning costs, in most cases, incurred after January 1, 2013, subject to various expiry provisions. Net profits are taxed under the profit royalty at a rate of 10% for net profits up to and including CAD$22.00 per kilogram (CAD$10 per pound) of uranium sold, and at 15% for net profits in excess of CAD$22.00 per kilogram. The CAD$22.00 threshold is applicable for 2013 (the base year) and is indexed in subsequent years for inflation.

Royalty payments are due to the province on or before the last day of the month following the month in which the royalty payer sold, or consumed, the uranium for the purposes of the basic royalty, and quarterly installments are required based on estimates of net profits in respect of the profit royalty.

22.3 Canadian Income and Other Taxes Applicable to Wheeler River

In 2018, taxable income of a Canadian resource company with a project located in the province of Saskatchewan is subject to federal tax at a rate of 15% and Provincial tax in Saskatchewan at a rate of 12% for a combined tax rate of 27%. This combined tax rate is applied to a company's taxable income for the year, which is calculated on a net basis after claiming certain allowable deductions.

Resource corporations in Saskatchewan are also subject to a uranium resource surcharge equal to 3% of the value of uranium resource sales from production in Saskatchewan, if any, during the year. As with the Basic Royalty and Resource Credit, the value of resource sales can be reduced by specified allowances – Denison has included the transport to convertor cost as a specified cost allowance for the purpose of the resource surcharge.

22.4 McClean Lake Toll Milling Revenue Applicable to Wheeler River

Denison's wholly owned subsidiary, DMI, holds a 22.5% interest in the McClean Lake joint venture project ("MLJV"). The MLJV is a joint venture between Orano Canada (formerly AREVA Resources Canada Inc. - 70% and operator), DMI (22.5%), and OURD (Canada) Ltd. (7.5%).

Participants in the MLJV receive their proportionate share of toll milling fees earned at the McClean Lake mill from toll milling carried out on behalf of any non-MLJV joint ventures or other third parties. In the case of Wheeler, where it has been assumed that Gryphon ore will be milled at the McClean Lake mill, DMI would receive 22.5% of the toll milling fees paid by the WRJV to the MLJV related to Gryphon ore processing, by virtue of its ownership in the MLJV.

22.5 Pre-tax Economic Analysis

22.5.1 Pre-tax Cash Flow Model – Base Case

Basis of the Model

The pre-tax base case cash flow model is based on the inputs noted in Section 22.1 and the following additional notes:

- The evaluation of the project is on a 100% ownership basis;
- No toll milling revenue or production credits applicable to MLJV participants is included;
- No Saskatchewan Profit Royalty is included;
- No provincial / federal tax calculations are included; and
- Net Present Value (NPV) calculations assume a discount rate of 8%.

Table 22-1. Wheeler River Project Pre-Tax Base Case Cash Flow Model
shows the Wheeler River project pre-tax base case cash flow model. Note: For presentation purposes, the post-production period (2038 to 2043) has been grouped into a single column.

Basis of Discount Rate

A discount rate of 8% was selected for assessing the time value of money for project economics. While the standard industry discount is 10%, a lower rate was selected based on the following rationale:

- Reference to current interest / lending rates are at relative lows;
- Project country risks (political stability, established taxation regime, extent of corruption and civil unrest) are considered low in Canada and in Saskatchewan;
- Minimal unexpected risks associated with operating a uranium mine in the eastern Athabasca Basin region in northern Saskatchewan due to the significant existing regional infrastructure and current mining / milling operations in the area;
- Relatively small scale of pre-production capital expenditures and short expected pre-production and pay back periods reducing inflationary exposure; and
- Assessment of project specific risks (geotechnical, hydrogeological) are already incorporated into the project economics through specific risk, schedule and contingency analysis and provisions throughout the financial modelling process.

Table 22-1. Wheeler River Project Pre-Tax Base Case Cash Flow Model

		Total	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038 To 2043
Key Physicals:																				
Mine Production																				
Mined Ore Tonnes - Phoenix	Tonnes	141,440	-	-	-	5,690	14,224	14,224	14,224	14,224	14,224	14,224	14,224	14,224	14,224	7,734	-	-	-	-
Mined Ore Grade - Phoenix	% U308	0.1913	-	-	-	0.1913	0.1913	0.1913	0.1913	0.1913	0.1913	0.1913	0.1913	0.1913	0.1913	0.1913	-	-	-	-
Mine Production - Phoenix	K lbs U308	59,662	-	-	-	2,400	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	3,262	-	-	-	-
Mined Ore Tonnes - Gryphon	Tonnes	1,257,463	-	-	-	-	-	-	-	-	-	13,183	70,751	136,258	283,972	264,784	214,436	218,242	55,837	-
Mined Ore Grade - Gryphon	% U308	0.0179	-	-	-	-	-	-	-	-	-	0.0149	0.0176	0.0195	0.0192	0.0187	0.0177	0.0157	0.0144	-
Mine Production - Gryphon	K lbs U308	49,712	-	-	-	-	-	-	-	-	-	432	2,742	5,850	12,038	10,928	8,382	7,574	1,767	-
Mill Feed																				
U308 Mill Feed - Phoenix	K lbs U308	59,662	-	-	-	2,400	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	3,262	-	-	-	-
U308 Mill Feed - Gryphon	K lbs U308	49,712	-	-	-	-	-	-	-	-	-	-	3,174	5,850	9,000	9,000	9,000	9,000	4,688	-
U308 Recovered (and Sold) in Period																				
U308 Recovered (and Sold) - Phoenix	K lbs U308	58,767	-	-	-	2,364	5,910	5,910	5,910	5,910	5,910	5,910	5,910	5,910	5,910	3,213	-	-	-	-
U308 Recovered (and Sold) - Gryphon	K lbs U308	48,817	-	-	-	-	-	-	-	-	-	-	3,117	5,745	8,838	8,838	8,838	8,838	4,604	-
Selling Price:																				
Revenue:																				
U308 Sales Price per lb - Phoenix	$USD / lb U308		$ 28.20	$ 28.19	$ 28.47	$ 29.48	$ 31.70	$ 34.21	$ 36.48	$ 37.72	$ 39.03	$ 40.19	$ 42.11	$ 43.70	$ 45.14	$ 44.94	$ 44.79	$ 44.79	$ 44.79	$ 44.79
U308 Sales Price per lb - Gryphon	$USD / lb U308		$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00	$ 50.00
U308 Sales Price per lb - Phoenix	$CAD / lb U308		$ 36.66	$ 36.65	$ 37.01	$ 38.32	$ 41.21	$ 44.47	$ 47.42	$ 49.04	$ 50.74	$ 52.25	$ 54.74	$ 56.81	$ 58.68	$ 58.42	$ 58.23	$ 58.23	$ 58.23	$ 58.23
U308 Sales Price per lb - Gryphon	$CAD / lb U308		$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00	$ 65.00
Project Economics:																				
Revenue																				
U308 Mineral Sales - Phoenix	K $CAD	2,969,458	-	-	-	90,588	243,551	262,818	280,252	289,826	299,873	308,798	323,513	335,747	346,799	187,692	-	-	-	-
U308 Mineral Sales - Gryphon	K $CAD	3,173,131	-	-	-	-	-	-	-	-	-	-	202,584	373,412	574,470	574,470	574,470	574,470	299,255	-
	K $CAD	6,142,589	-	-	-	90,588	243,551	262,818	280,252	289,826	299,873	308,798	526,097	709,160	921,269	762,162	574,470	574,470	299,255	-
Operating																				
Mining																				
Phoenix	K $CAD	(44,020)	-	-	-	(2,220)	(4,398)	(4,398)	(4,398)	(4,398)	(4,398)	(4,398)	(4,398)	(4,398)	(4,398)	(2,220)	-	-	-	-
Gryphon	K $CAD	(266,201)	-	-	-	-	-	-	-	-	-	(3,118)	(26,114)	(34,604)	(52,115)	(51,027)	(45,548)	(40,152)	(13,524)	-
Mining / Processing																				
Phoenix	K $CAD	(115,577)	-	-	-	(5,217)	(11,558)	(11,558)	(11,558)	(11,558)	(11,558)	(11,558)	(11,558)	(11,558)	(11,558)	(6,341)	-	-	-	-
Gryphon	K $CAD	(412,621)	-	-	-	-	-	-	-	-	-	-	(26,343)	(48,557)	(74,702)	(74,702)	(74,702)	(74,702)	(38,914)	-
Transport to Convertor	K $CAD	(22,593)	-	-	-	(496)	(1,241)	(1,241)	(1,241)	(1,241)	(1,241)	(1,241)	(1,896)	(2,448)	(3,097)	(2,531)	(1,856)	(1,856)	(967)	-
G&A-Site Support / Admin	K $CAD	(135,610)	-	-	-	(3,401)	(8,068)	(8,926)	(10,937)	(12,029)	(10,408)	(10,440)	(10,936)	(10,501)	(10,486)	(10,377)	(10,136)	(9,825)	(9,139)	-
	K $CAD	(996,623)	-	-	-	(11,334)	(25,265)	(26,123)	(28,134)	(29,226)	(27,605)	(30,754)	(81,244)	(112,065)	(156,356)	(147,197)	(132,242)	(126,534)	(62,543)	-
Sales Royalties / Non-Income Taxes																				
Sask. Resource Surcharge	K $CAD	(183,600)	-	-	-	(2,703)	(7,269)	(7,847)	(8,370)	(8,658)	(8,959)	(9,227)	(15,726)	(21,201)	(27,545)	(22,789)	(17,178)	(17,178)	(8,949)	-
Sask. Basic Royalty	K $CAD	(260,100)	-	-	-	(3,829)	(10,298)	(11,117)	(11,858)	(12,265)	(12,692)	(13,071)	(22,279)	(30,035)	(39,022)	(32,284)	(24,336)	(24,336)	(12,677)	-
	K $CAD	(443,700)	-	-	-	(6,532)	(17,567)	(18,964)	(20,228)	(20,922)	(21,651)	(22,298)	(38,005)	(51,237)	(66,567)	(55,073)	(41,515)	(41,515)	(21,626)	-
Pre-Tax Contribution before CAPEX	K $CAD	4,702,266	-	-	-	72,723	200,719	217,731	231,890	239,678	250,617	255,745	406,848	545,858	698,345	559,892	400,714	406,421	215,085	-
Pre-Tax Contribution before CAPEX - Phoenix	K $CAD	2,500,864	-	-	-	72,723	200,719	217,731	231,890	239,678	250,617	259,615	275,433	288,932	301,040	162,487	-	-	-	-
Pre-Tax Contribution before CAPEX - Gryphon	K $CAD	2,201,401	-	-	-	-	-	-	-	-	-	(3,869)	131,415	256,925	397,305	397,405	400,714	406,421	215,085	-
Capital Costs																				
CAPEX - Phoenix	K $CAD	(425,950)	(24,973)	(126,295)	(147,995)	(24,970)	(3,471)	(3,936)	(6,163)	(3,951)	(3,485)	(6,607)	(3,479)	(4,026)	(5,522)	(26,129)	(8,831)	(6,095)	(6,095)	(13,928)
CAPEX - Gryphon	K $CAD	(705,863)	-	-	-	-	(5,459)	(76,987)	(100,765)	(155,944)	(114,310)	(169,907)	(36,735)	(17,745)	(9,711)	(7,630)	(5,800)	(2,825)	(76)	(1,969)
	K $CAD	(1,131,813)	(24,973)	(126,295)	(147,995)	(24,970)	(8,929)	(80,923)	(106,928)	(159,895)	(117,795)	(176,513)	(40,214)	(21,771)	(15,234)	(33,759)	(14,631)	(8,919)	(6,171)	(15,897)
Pre-Tax Contribution after CAPEX	K $CAD	3,570,453	(24,973)	(126,295)	(147,995)	47,753	191,789	136,807	124,962	79,783	132,823	79,232	366,634	524,087	683,112	526,133	386,082	397,502	208,914	(15,897)
Pre-Tax Contribution after CAPEX - Phoenix	K $CAD	2,074,914	(24,973)	(126,295)	(147,995)	47,753	197,248	213,794	225,728	235,727	247,133	253,008	271,954	284,906	295,518	136,358	(8,831)	(6,095)	(6,095)	(13,928)
Pre-Tax Contribution after CAPEX - Gryphon	K $CAD	1,495,539	-	-	-	-	(5,459)	(76,987)	(100,765)	(155,944)	(114,310)	(173,776)	94,680	239,181	387,594	389,775	394,913	403,597	215,009	(1,969)

Project Financial Metrics		At Pre-Production		At Pre-Production	
Net Present Value / Internal Rate of Return		NPV	IRR	NPV	IRR
Pre-Tax Contribution after CAPEX					
Combined at F'2021 at 8%	K $CAD / %	$1,308,279	38.71%		
Phoenix at F'2021 at 8%	K $CAD / %	$ 930,446	43.31%		
Gryphon at F'2026 at 8%	K $CAD / %			$ 560,619	23.15%

22.5.2 Pre-Tax Indicative Economic Results – Base Case

The Wheeler River project pre-tax indicative base case economic results are as follows:

- NPV (8%) of $1,308 million when discounting project cash flows back to 2021;
- Internal rate of return ("IRR") of 38.7%; and
- Payback period of ~24 months from the start of uranium production in July 2024.

Note that this analysis considers all project pre-development costs (i.e. environmental assessment, feasibility study, etc.) as excluded capital costs (Section 21-5) and as such they are not factored into the above calculations.

22.5.3 Pre-Tax Sensitivities – Base Case

Basic Sensitivities

The base case results are summarized as follows (all in Canadian dollars):

Average uranium price	$57.10 per pound (average combined selling price of USD$43.92/lb U_3O_8 converted using exchange rate of 1.30 CAD/USD)
Average mill feed grade	3.55% U_3O_8
Average site operating cost	$9.26 per pound U_3O_8 (total Phoenix and Gryphon operating costs of $996.623 million divided by recovered pounds of 107.584 million)
Total project capital cost	$1,131.813 million

Denison has prepared a sensitivity analysis by varying these four inputs. Table 22-2 shows the impact on NPV (8%), in millions of dollars, of varying these input values on the base case pre-tax economic indicators. Figure 22-1 presents these sensitivities in graphical format. As with most mining projects, the most sensitive parameter is the commodity price. Mill feed grade (% U_3O_8) is the next most sensitive parameter.

Table 22-2. NPV Sensitivity

	NPV (8%) Sensitivity ($M)						
Variable	-30%	-20%	-10%	0%	10%	20%	30%
Capex	1,509	1,442	1,375	1,308	1,241	1,174	1,107
Opex	1,423	1,385	1,347	1,308	1,270	1,231	1,193
Uranium Price	600	836	1,072	1,308	1,544	1,781	2,017
U_3O_8 Grade	650	875	1,094	1,308	1,532	1,756	1,979

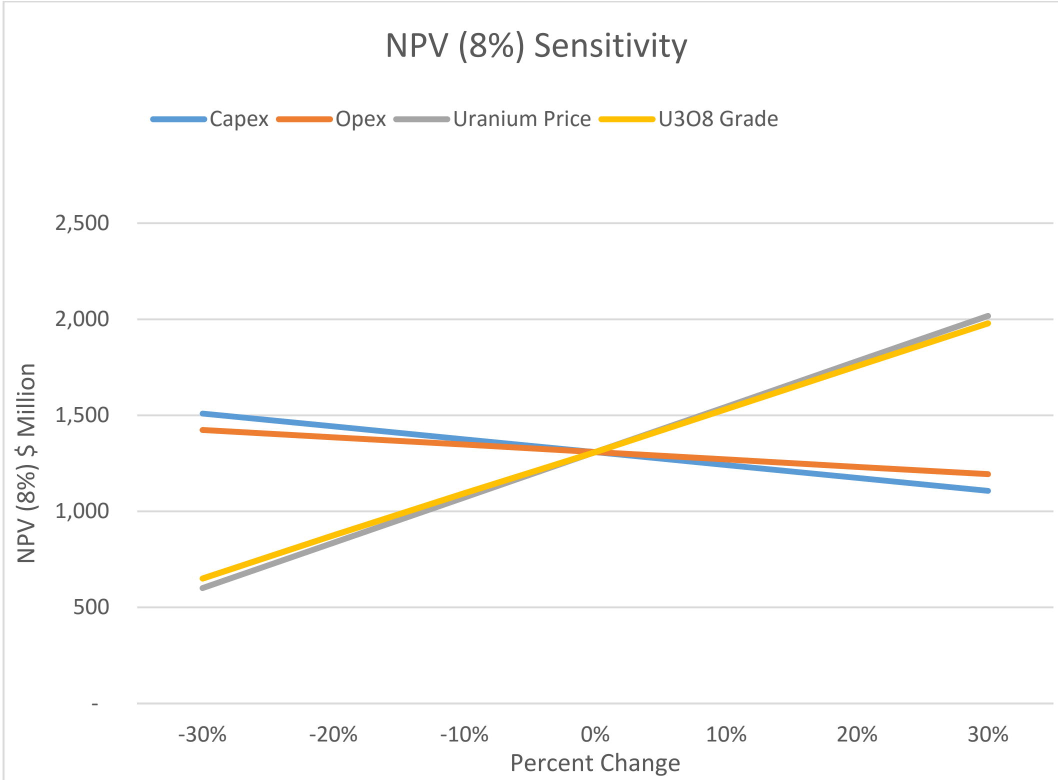


Figure 22-1. NPV Sensitivity

22.5.4 Price Sensitivity – High Case

The project economic results are quite sensitive to the price of uranium. To illustrate the potential for the project to benefit from rising uranium prices, the PFS considered an additional pricing scenario, the High Case, which used an estimated uranium selling price of USD$65.00/lb for all production.

The Wheeler River project pre-tax indicative high case economic results are as follows:

- NPV (8%) of $2,587 million when discounting project cashflows back to 2021;
- IRR of 67.4%; and
- Payback period of ~11 months from the start of uranium production in July 2024.

A summary of the economic results of the base and high case scenarios are illustrated in Table 22-3.

Table 22-3. Pre-tax Economic Results

Scenario	Pre-Tax Results	NPV	IRR	Payback
Base Case	(UxC spot price for Phoenix & fixed USD$50/lb U3O8 for Gryphon production)	$1,308 million	38.7%	~ 24 Months
High Case	All production at USD$65/lb U3O8	$2,587 million	67.4%	~ 11 Months

22.6 Post Tax Economic Analysis

22.6.1 Post Tax Cash Flow Model – Base Case

The post-tax base case cash flow model is specific to Denison's ownership interest in the WRJV and Denison's specific facts and circumstances as it relates to: a) tax pools it has available to it to reduce taxable income for both Saskatchewan Profit Royalties as well as Canadian Federal and Provincial income taxes, and b) benefits that accrue to it from its interest in the MLJV.

Effective September 24, 2018 Denison held a 63.30% ownership interest in the WRJV. Under two publicly announced agreements, Denison can increase its ownership in the WRJV up to 90% as follows:

- In January 2017, Denison agreed to fund additional WRJV expenses in 2017 and 2018 to increase its interest in the WRJV up to approximately 66.16% at the end of fiscal 2018. Under this agreement, DMI would be the owner of this WRJV interest; and
- In September 2018, Denison agreed to increase its interest in the WRJV by acquiring Cameco's remaining interest (expected to be approximately 23.84%) in exchange for the issuance of common shares of Denison Mines Corp ("DMC"). DMC would be the owner of this WRJV interest. Upon completion of the transaction Denison's ownership would increase to 90%.

To illustrate the economic impact of the change in ownership, Denison has provided post-tax base case cash flow model summaries at 63.3% (its current ownership interest) and 90% (its pro-forma maximum ownership interest). The post-tax base case cash flow model is based on the inputs noted in Section 22.1 with the following additional items:

- Adjustments for Denison's share of project development costs (refer to section 21) and the associated impact on Denison's estimated tax pools;
- The economic benefits of Denison's 22.5% share of MLJV's toll milling fees;
- The impact of the Saskatchewan Profit Royalty applicable on uranium production;
- Denison's expected Federal and Provincial income taxes payable; and

Discounting for NPV calculations remains at 8% and the impact of Denison's NPI interest (refer to section 4) has not been included as the impact is estimated to be immaterial.

The following assumptions were used in computing the Federal and Provincial income tax payable, as well as Saskatchewan Profit Royalty amounts owing by Denison in the model:

- All estimated applicable tax deductions currently available in DMC and DMI at June 30, 2018, and those which will arise in the future related to the Wheeler project will be available for use as a deduction against income generated from the Wheeler River project;
- Tax deductions attributable to DMI were used to reduce taxable income from the Wheeler River project up to DMI's maximum ownership interest in the WRJV of 66.16% - tax deductions attributable to DMC were used to reduce taxable income attributable to Wheeler River for Denison ownership interest in the WRJV in excess of 66.16%;
- The currently enacted tax laws and the proposed tax law amendments at the time of this PFS are those that will apply during the life of the Wheeler River project (as well as the existing interpretations and assessing practices of the applicable taxing authority), and that substantially all of the income from the project will be taxed using a combined Canadian Federal and Saskatchewan income tax rate of 27.0% (Federal – 15% / Saskatchewan – 12%);

- Non-capital losses will continue to have a loss carry forward period of 20 years for income tax purposes and 10 years for Saskatchewan Profit Royalty purposes; and
- For Saskatchewan Profit Royalty computations, the $10.00 profit per pound U_3O_8 threshold demarcating the 10% and 15% net profit taxation tiers has been indexed up to Q4-2017 to approximately $10.56 per pound U_3O_8 and then held constant throughout the life of the Wheeler River project.

Table 22-4 contrasts the results of the Wheeler River project base case pre-tax cash flow model and the post-tax cash flow model as it applies to Denison's current ownership interest and pro-forma maximum ownership interest.

Table 22-4. Base Case Cash Flow Model – Pre-Tax vs Post Tax Comparison

Item Description	Base Case Pre-Tax Summary	Base Case Post Tax Summary	Base Case Post Tax Summary
CAD$ millions			(Pro-Forma)
Project Percentage	100.0%	63.30%	90.00%
Gross Uranium Revenue	**6,142.6**	**3,888.3**	**5,528.3**
Toll Milling Fees	Excl.	4.8	1.3
Operating Costs	(996.6)	(630.9)	(897.0)
Operating Costs – Toll Milling Credits	Excl.	8.2	11.7
Saskatchewan Revenue Royalties, Surcharges	(443.7)	(280.9)	(399.3)
Operating Cash Flow	**4,702.3**	**2,989.5**	**4,245.0**
Capital Costs	(1,131.8)	(716.4)	(1,018.6)
Capital Costs – Project Development	Excl.	(13.5)	(19.2)
Contribution before Taxes	**3,570.5**	**2,259.6**	**3,207.2**
Saskatchewan Profit Royalties	Excl.	(341.0)	(421.9)
Canadian Federal / Provincial Income Taxes	Excl.	(497.2)	(685.5)
Net Contribution	**3,570.5**	**1,421.4**	**2,099.8**
NPV (8%) at fiscal 2021	1,308.3	506.4	755.9
IRR	38.7%	31.7%	32.7%

Net contribution represents the undiscounted cash flow impact applicable to the Wheeler project.

22.6.2 Post Tax Cash Flow Model – High Case

The sensitivity of the post-tax cash flow model to Capex, Opex, Uranium Price and U_3O_8 Grade is similar to that of the pre-tax cash flow model.

To illustrate the project's sensitivity to higher uranium prices, the high case post-tax cash flow model is based on a uranium selling price of USD$65.00/lb with all other assumptions used in the base case post-tax cash flow model held constant.

Table 22-5 contrasts the results of the Wheeler River project high case pre-tax cash flow model and the post-tax cash flow model as it applies to Denison's current ownership interest and pro-forma maximum ownership interest.

Table 22-5. High Case Cash Flow Model – Pre-tax vs Post Tax Comparison

Item Description	**High Case Pre-Tax Summary**	**High Case Post Tax Summary**	**High Case Post Tax Summary**
CAD$ millions			(Pro-Forma)
Project Percentage	100.0%	63.30%	90.00%
Gross Uranium Revenue	**9,090.9**	**5,754.5**	**8,181.8**
Toll Milling Fees	Excl.	4.8	1.3
Operating Costs	(996.6)	(630.9)	(897.0)
Operating Costs – Toll Milling Credits	Excl.	8.2	11.7
Saskatchewan Revenue Royalties, Surcharges	(657.5)	(416.2)	(591.7)
Operating Cash Flow	**7,436.8**	**4,720.4**	**6,706.1**
Capital Costs	(1,131.8)	(716.4)	(1,018.6)
Capital Costs – Project Development	Excl.	(13.5)	(19.2)
Contribution before Taxes	**6,305.0**	**3,990.5**	**5,668.3**
Saskatchewan Profit Royalties	Excl.	(617.5)	(776.8)
Canadian Federal / Provincial Income Taxes	Excl.	(889.9)	(1,254.2)
Net Contribution	**6,305.0**	**2,483.1**	**3,637.3**
NPV (8%) at fiscal 2021	2,587.7	1,006.2	1,483.8
IRR	67.4%	53.8%	55.7%

Net contribution represents the undiscounted cash flow impact applicable to the Wheeler project.

23 Adjacent Properties

There are no properties adjacent to the Denison Wheeler River property that are considered relevant to this technical report.

24 Other Relevant Data and Information

24.1 Project Execution Plan

The project execution strategy is focused on delivering the project on budget, scope, and schedule. In order to accomplish these goals, the philosophy of the execution generally follows:

- Use of external firms: Simply put, Denison is not in the business of nor does it specialize in the design and construction of surface, underground, or processing facilities. As such, Denison will seek out and partner with specialty firms to provide these services for project development. Denison has already engaged with reputable engineering firms and personnel for the PFS and will aim to continue building these relationships in future steps of project development. This will minimize both transition risk between stages and the re-learning of unique aspects of the project.
- Staged execution of the project: The project will be implemented in two stages, with the Phoenix deposit initiated first, followed by the Gryphon deposit. Within each of the two projects, further sequencing of activities has been established to identify critical path activities and to organize the project in separate phases. Staged construction of the two deposits simplifies project execution and reduces overall risk associated with project construction.
- Materials procurement: At all times, a formal tendering process will be used to select suitably qualified vendors for materials and supplies. As a remote site, Denison will seek to maximize pre-fabrication of buildings and equipment with transportation to site as pre-assembled units. This will reduce construction costs and schedules, project complexity, and the number of personnel on site.
- Construction procurement: At all times, a formal tendering process will be used to select suitably qualified vendors for construction. Denison will target lump sum and/or unit rate contracts to provide a degree of inherent cost control on the project. Denison has already received interest in the project from local contractors and it is expected that, during the feasibility study, procurement documents will be prepared in advance for early engagement of contractors. This will be especially important to ensure Denison is allowed time to initiate discussions, communications, and relationships between contractors and northern communities.
- Constructability reviews: Denison has already engaged contractors in the design phase to provide input and guidance on constructability of the project. Third party reviews and optimization of project design will continue in future stages to further add value to the project.
- Operations: The project is designed to be operated by employees once construction activities are complete. Saskatchewan has a long history in mining operations and established training programs for young people just starting out. Given the small-scale nature of the operation, it is expected that a small management team and workforce will be the most efficient method to operate the site. The combination of the Phoenix and Gryphon mine schedules and the low operating costs offers an attractive career for employees with lower risk of closures during market downturns.

24.1.1 Phoenix Deposit

The Phoenix project is broken down into several key phases. The first is the pre-construction phase, followed by construction and operations.

Pre-development Phase

Following receipt of environmental approvals, the preparatory phase will include initiation of licensing activities, organization of the project execution team, preparation of key project documents, and procurement of equipment, materials, and labour. These activities will be initiated during the last stages of the feasibility study should results continue to support advancement of the project.

Construction Phase

Following receipt of licensing approvals, construction activities on site will commence. Construction of Phoenix infrastructure has several key areas which make up the bulk of the capital costs.

Site Preparation: Clearing and leveling of the surface facilities will be contracted out to a suitable contractor. Lidar surveys of the area for PFS level estimation and contracting at unit rates will minimize the risk of capital cost overruns.

Wellfield and Freeze Hole Drilling: Denison has been drilling on the property since 2004 and has successfully drilled over 250 holes and 110,000 meters into the deposit and surrounding sandstone. This experience and knowledge will be applied to the drilling of the freeze and wellfield holes. Suitably qualified and experienced contractors will be overseen by Denison personnel to complete drilling activities. This organization, currently in use by Denison, has been successfully implemented and has resulted in some of the lowest cost drilling operations in the Athabasca Basin.

ISR Plant Construction: While the plant is likely the most complex construction activity for the project, when compared to other projects, it is relatively simple, with limited numbers of vessels and piping. Furthermore, due to the degree of isolation of the plant from other site facilities, construction of this facility can be prioritized with minimal impact to other facilities. Denison has already received interest in lump sum construction of this plant and will seek to engage interested parties for constructability reviews. Most of the equipment and materials inside the plant are small in nature, enabling the shipment of tanks and other vessels pre-assembled.

Other Surface Infrastructure: Other surface infrastructure includes camp buildings, the operations centre, the airport terminal building, and various other smaller infrastructure. With the exception of the operations centre, all other buildings are expected to be pre-fabricated buildings to reduce the costs of construction on site.

Commissioning of the facilities is expected to be supported by engineering and/or supplies vendors along with the assistance of the construction teams. This will ensure constructed facilities adhere to the designs and specifications set forth and will reduce turnover challenges for the operations team.

Project and construction management during the capital development phase of the project will be managed by a small dedicated project management team. During the construction phase, Denison will provide general and administrative services to operate the site and support the contractors in construction (i.e. room and board, flights, general supplies, freight haulage, etc.). It is expected that a mix of employees, contractors, and engineering service providers will support site construction efforts.

Operations Phase

The capital construction period will end with the first production of yellowcake from the ISR plant. Operations for the ISR deposit is planned to last 11 years. Denison anticipates operating the site with employees, with limited numbers of external contractors.

24.1.2 Gryphon Deposit

After Phoenix enters production, construction activities will be initiated at Gryphon. The Gryphon project is broken down into several key phases.

Pre-development Phase

The Gryphon development will benefit from Phoenix being an operating site, with various departments providing support (logistics, procurement, safety and environment, etc.). The established, fixed infrastructure and personnel will also reduce complexity of the project. It will be more in line with a brownfield site, without the disadvantages of working around production activities. The Gryphon site is 3 km from Phoenix and as such, it will be a separate site with limited interaction from producing operations at Phoenix.

The established site operations and regulatory approvals will enable a shorter pre-development construction period. During this time, procurement and detailed engineering will be completed for the shaft construction. Otherwise, the rest of the engineering and procurement will be completed as the mine is developed.

Construction Phase

Following receipt of licensing approvals, construction activities on site will commence. Construction of Gryphon infrastructure has several key areas which make up the bulk of the capital costs. Similar to Phoenix, construction of surface facilities, excavation of the shaft, and construction of underground facilities will be tendered out to contractors. Standard mine development will be completed by Denison employees.

Blind Boring: Shaft excavation is an independent process with contractors being completely self-sufficient. Excavations of shafts using this technique is typically completed under civil construction standards, which include lump sum contracts. Mobilization of the contractor for the 5.0 m diameter production shaft will be initiated. A composite concrete/steel liner will be installed to shaft bottom and grouted in place. Once blind boring is complete on the production shaft, the equipment will be relocated to the ventilation shaft for excavation, followed by lining.

Construction of Surface Facilities: During the blind boring of the shafts, construction of surface facilities will be initiated so as to eliminate these activities from the project critical path. Once shaft excavation is complete, installation of the headframe will be completed to allow for installation of permanent infrastructure in the production shaft down to the 500 m level. The contractor will excavate the 500 Level shaft station, providing sufficient room to enable development gear to be reassembled underground and development to commence once the shaft work is complete. The shaft crew will complete the equipping to shaft bottom and excavate/equip the loading pocket and ore pass to the 500 Level.

Mine Development and Construction: Initial mine development and construction of underground infrastructure will be completed by suitably qualified mining contractors. Unit rate contracts for mine development and lump sum contracts for well defined infrastructure projects will be employed. Once initial mine development is completed and development proceeds down the main ramp, standard mine development activities will be assumed by Denison employees. Once the production shaft is commissioned, a single development crew will commence off-shaft development, with a focus on accessing the ventilation shaft for flow through ventilation and secondary egress. Development will also progress towards the first production area in the E Zone mining block on the 582 Level.

Project and construction management during the capital development phase of the project will be managed by a small dedicated project management team, supported by the existing services and personnel operating Phoenix. Denison will continue to operate site, providing general and administrative services and support to the construction (i.e. room and board, flights, general supplies, freight haulage, etc.).

24.2 Construction Schedule

The Wheeler River project carries an overall duration of approximately 67 months from the commencement of feasibility and initiation of the environmental assessment process through first production at Phoenix in late July 2024. This is followed just over six years later, with first production at Gryphon in September of 2030.

The project has been scheduled as two independent phases, with the higher-grade Phoenix deposit starting first, followed by Gryphon construction commencing just after the first year of commercial production at Phoenix.

Pre-construction activities begin in January 2019 with the initiation of the environmental assessment exercise, which has been scheduled to take three years to receive full project approvals in January of 2022. The project feasibility study also starts in January of 2019 and is expected to take two years to complete. The year of lag post feasibility and pre-environmental approvals will be used for detailed engineering of critical path activities, procurement of long lead time items, and the selection of appropriate contractors for early construction packages.

The critical path for physical construction flows through the establishment of the ISR processing plant and the creation of the frozen curtain over the Phoenix orebody to initiate ISR production.

Physical site work commences in the early spring of 2022, following environmental approvals, and is currently scheduled to begin in May of that year once physical conditions have recovered from winter. The initial earthworks construction will focus on preparing suitable roads from the main access highway into the site, specifically to the ISR plant and the two ends of the Phoenix deposit where the ground freezing drilling will occur. These two sites will remain the focus of levelling and grading activities into the autumn period. All of this work will be supported by temporary camps and utilities while permanent facilities are established.

Two streams of work carry on in parallel to support the critical path goal of establishing ISR production, the first being ground freezing above the Phoenix orebody and the second being construction of the ISR processing plant.

Ground freezing requires the establishment of a pattern of freeze wells drilled across the orebody and of refrigeration units and corresponding electrical and mechanical services to each. Freeze well drilling is expected to commence in the summer of 2022 and proceed through that autumn. In parallel, the ground freezing units will arrive at site and be physically installed, site electrical infrastructure will be installed, with the main powerline being constructed during this first summer, and the main electrical substation will be constructed. Electrical distribution to the ground freezing plants is completed by February of 2023, which is in sync with the completion of the ground freezing plant construction and connections into the freeze wells. This facilitates the commencement of ground freezing in February of 2023, which is expected to require 14 months to freeze an appropriate freeze curtain around the orebody by late April 2024.

ISR plant construction begins immediately following earthworks at the ISR plant site. Foundations begin in October 2022, with the pre-engineered building being installed in December 2022, and mechanical, electrical, and other installation work commencing in March of 2023 to have the plant complete by March of 2024. A three-month commissioning period begins post-construction and leads to first ISR production in late July of 2024. The ISR wellfield phase one drilling is carried out over the previous winter and is completed just ahead of ISR commissioning.

The summer of 2023 begins again with the second phase of earthworks, when the balance of the site is completed, facilitating the completion of the remaining infrastructure and utility systems.

The operations center is completed in January of 2024, which will allow the operations team to be in place ahead of commissioning to complete all preparedness for operations activities in the suitable building, and will create a permanent maintenance facility ahead of operations commencement. The permanent camp is completed in a similar time frame, along with basic services such as permanent communications and fire systems.

The balance of the infrastructure items, such as storage areas, incinerator, and security checkpoints, are completed at about the time of commissioning and will complete the construction at Phoenix.

Phoenix milestones are summarized in Table 24-1 below.

Table 24-1. Project Construction Milestones - Phoenix

Milestone	Completion Date
Environmental Assessment Start	January 1, 2019
Feasibility Start	January 1, 2019
Feasibility Complete	January 1, 2021
Environmental Assessment Complete	January 1, 2022
Earthworks Start	May 1, 2022
Electrical Complete	February 27, 2023
Ground Freezing Start	February 27, 2023
Wellfield Drilling Start	January 1, 2024
Commissioning of Phoenix Facilities Complete	January 5, 2024
Wellfield Complete	April 18, 2024
Ground Freezing Complete	April 22, 2024
Commissioning of ISR Complete	July 21, 2024
Phoenix First Production	July 21, 2024

The Gryphon deposit has been scheduled such that physical construction begins approximately one year after Phoenix ISR production commences.

Gryphon will have an independent feasibility study, which will last two year and will commence in July 2023.

The critical path for Gryphon development flows through the establishment of the two mine shafts and the subsequent underground development to the orebody, which facilitates first production.

Engineering and procurement for the blind boring contractor is assumed to start immediately after the approval of the feasibility study in July of 2025 and continues for 9 months into the spring of 2026. Production shaft collar work begins at this time, along with setup of drilling equipment and training of personnel. Blind boring of the shaft and the ensuing liner installation begins in July 2026 and continues through December of 2027.

Installation of the physical plant required for shaft equipping begins in April 2027 during shaft lining and there is an approximate 4-month period post-lining where the headframe is installed and preparations are completed for shaft equipping. Shaft equipping is completed in February 2029 and the production shaft is commissioned at the same time.

The ventilation shaft collar construction commences in October 2027 and blind boring of the ventilation shaft commences in December 2027 following drilling and lining of the production shaft. Boring and lining is completed in March 2029. The ventilation shaft is commissioned in September 2029 following construction of the headframe and hoist house, and equipping.

The two shafts are connected underground for ventilation purposes by November 2029, and development of infrastructure and construction of facilities occurs in the ensuing four months. Underground infrastructure is complete on the 500 Level by September 2030. Underground development continues through the construction period and first ore production occurs in September 2030.

Gryphon surface construction is scheduled to be developed in parallel with shaft sinking periods and to be completed at the time that underground development commences. Gryphon road work and earthworks is completed in November 2026. Physical systems, such as fuel, firewater, and propane, are completed in September 2028.

Of concern is the establishment of the site water treatment plant at Gryphon. This facility is commissioned in November of 2028 and ready for the end of shaft sinking activities.

As Gryphon ore will be trucked via highway trucks, the extension of highway 512 will coincide with the start of Gryphon construction and is completed in September 2030.

Gryphon milestones are summarized in Table 24-2 below.

Table 24-2. Project Construction Milestones - Gryphon

Milestone	Completion Date
Feasibility Start	July 28, 2023
Feasibility Complete	July 17, 2025
Surface Engineering Complete	January 12, 2026
Mining Engineering Complete	April 12, 2026
Mobilization to Site	April 13, 2026
Blind Boring of Production Shaft Complete	December 11, 2027
Production Shaft Headframe and Surrounding Infrastructure Complete	May 28, 2028
Shaft Surface Infrastructure Complete	November 27, 2028
Initiation of McClean mill upgrade engineering followed by construction	January 1, 2029

Milestone	Completion Date
Production Shaft Equipping Complete	February 20, 2029
Blind Boring of Ventilation Shaft Complete	March 9, 2029
Ventilation Shaft Headframe and Surrounding Infrastructure Complete	June 7, 2029
Ventilation Shaft Equipping Complete	September 3, 2029
McClean mill upgrades complete	December 2031
Gryphon First Production	September 17, 2030

24.3 Project Opportunities and Risks

During the completion of the PFS, risk and opportunity assessments were completed for all major areas of work including Phoenix mining and processing, site infrastructure, Gryphon mining and surface infrastructure, McClean Lake mineral processing, environmental and regulatory, capital and operating costs, financial modelling, and market analysis, among others.

Risk assessments were completed by a combination of QPs, consulting engineers, and Denison project personnel using a standardized scoring system. Risks and opportunities were evaluated based on likelihood and consequence/benefit to arrive at a quantitative score. The highest ranked factors are classified as critical, meaning if the event occurs, it could have a material impact on the project. Lower ranked factors are classified as high, moderate, or low as the quantitative score decreases.

Following the ranking, action plans (opportunities) and mitigations plans (risks) were identified and developed to address the item in future work. The action and mitigation plans and associated budgets are incorporated into the next steps in project development. A brief description of some of the higher ranked opportunities and risks follows.

24.3.1 Project Opportunities

During the study, a total of 32 opportunities were identified, with the majority of them being material to the project (Figure 24-1). Action plans have been established for all opportunities and are incorporated into future project development budgets. A brief summary of the material opportunities follows.

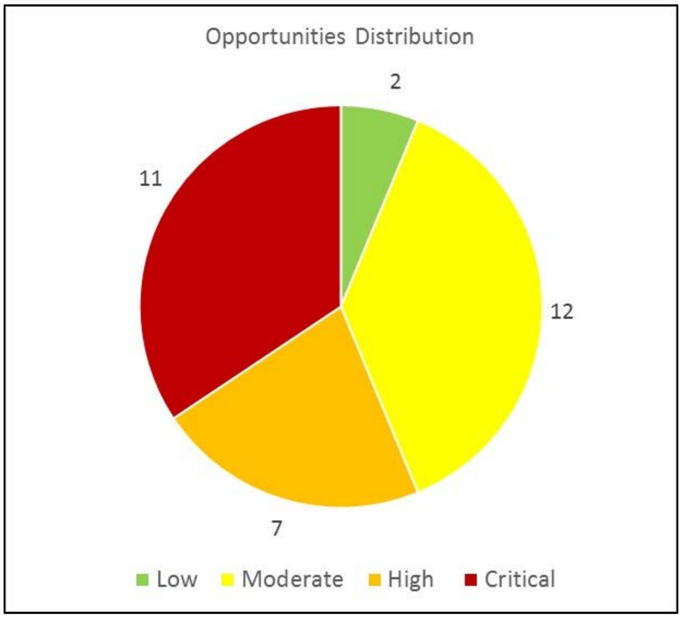


Figure 24-1. Project Opportunities Ranking Distribution

1. Increase in total Production at Phoenix: ISR metallurgical testing achieved over 90% resource extraction with testing halted prematurely while pregnant leaching solution concentrations remained above 5 g/l. With economic cut-off grades far below this level, there may be potential for additional resource extraction above the 85% assumed in the PFS. In addition, the current mineral resource estimate at Phoenix is based on a 0.8% U_3O_8 cut-off grade based on previously assumed conventional mining practices. Using an ISR extraction method is significantly less costly and as a result, a lower cut-off grade can be utilized. Furthermore, there are other areas at Phoenix (Zone C, Zone D, mineralized zones between Zone A and B) which are known to contain mineralization but have not been drilled or quantified to a mineral resource level of confidence due to the negligible impact these would have on an underground conventional mining operation. However, this mineralization may prove to be attractive under an ISR extraction technique.

2. Phoenix production increase: The current production plan is based on the assumption that 10 recovery wells will be producing 10 g/L solution for an overall production level of 6 M lbs/yr. Mineral processing test work has demonstrated solution average grades of 12 g/L and grades as high as 27 g/L. Additionally, there is a total of 94 recovery wells planned for the Phoenix Zone A and Zone B mineral resources. Based on the above information, should the solution grades be higher than the assumptions used in this study, or if additional recovery wells are operated, there is potential for production levels to increase above the 6 M lbs/yr plan.

3. Wellfield drilling: The project design assumed contractor drilling of the wellfields using a reverse circulation drill rig for all aspects of the well drilling and installation. In the future, evaluation of owner supplied equipment and/or utilization of multiple drill rigs for installation could significantly reduce costs and schedule for wellfield construction.

4. Gryphon ore sorting: During the metallurgical testing, it was determined that radiometric sorting of the ore may be possible. This would significantly reduce the quantity of ore to be transported to the mill for processing and would have material reductions in transportation costs, milling costs, tailings storage requirements, etc.

5. Processing Gryphon ores at Wheeler River: Due to the mining sequence, the Phoenix ISR plant will be constructed and operating well before Gryphon is developed. The current plan for processing of Gryphon ores is to ship them to McClean Lake, which requires significant capital and operating costs, including construction of a 50 km extension of Highway 914, McClean Lake mill upgrades, toll milling and tailings storage fees, mill operating costs, etc. Future work will evaluate the construction of a front end to the ISR mill (i.e. grinding and leach circuit) to process Gryphon ores to reduce capital and operating costs.

6. Rare earth metals: Both the Phoenix and Gryphon deposits contain levels of rare earth elements. The PFS focused on the production of uranium and did not consider recovery of other valuable elements. However, from metallurgical test work, it is known that rare earth metals may be recovered in the leaching process. Future work will evaluate the recovery of other metals from the pregnant leach solution.

7. Gryphon sill development: Current ground support designs in the ore sills require bolts and screen. Later in the development design stage, shotcrete was included in the design for radiological exposure reduction. In future project stages, optimization of ground support will allow for elimination of ground support duplication, saving significant time and money during sill development phases.

8. Phoenix freeze wall spacing: Currently, the schedule allows for a total of 19 months to drill freeze holes at 5 m centres and to establish a 10 m freeze wall surrounding the deposit. The critical path for production is estimated at 30 months. As freezing is not on the critical path for Phoenix production, reduction of the capital costs by increasing freeze hole spacing to 6 or 7 m (reducing the total meters of drilling) and/or purchasing smaller freeze plants (requires longer time to freeze) will enable material reductions in the capital costs.

24.3.2 Project Risks

During the study, a total of 66 risks were identified, with the majority of them being rated as low to moderate level risks (Figure 24-2). Mitigation plans have been established for all risks and are incorporated into future project development budgets. A brief summary of the material risks follows.

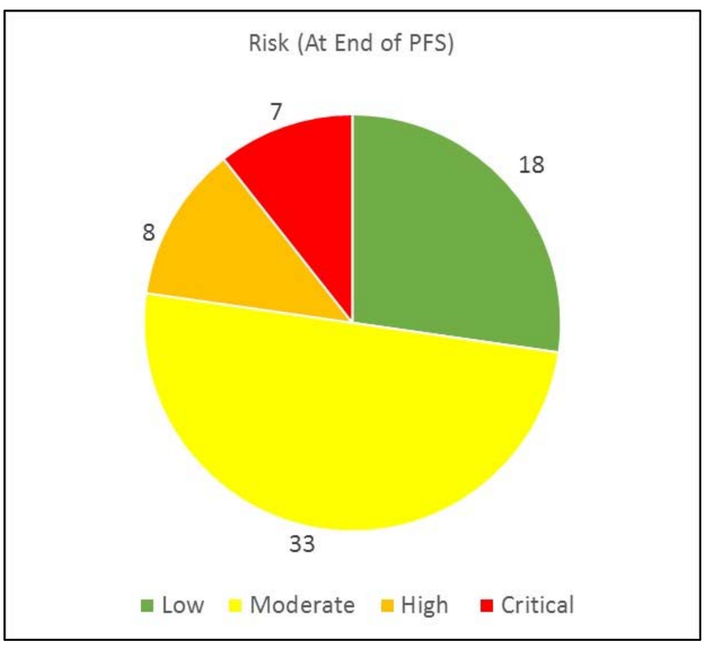


Figure 24-2. Project Risks Ranking Distribution

1. Regulatory approvals: The design assumes that federal and provincial project approvals will be granted in certain time periods and without major impact to the project. There is no guarantee that such approvals will be received or that they will be received in the time period assumed.

2. Gryphon toll milling agreements: Processing of Gryphon ores would require a toll milling agreement between the WRJV and the MLJV. There is no guarantee that such an agreement will be completed or that the terms of the agreement, toll milling capacities, and /or associated fees would be at the levels assumed within this PFS.

3. Gryphon toll milling: Impact of future Cigar Lake grades on process design. Data was requested from Denison on future production grades from Cigar Lake, however Hatch was informed that this data was not available. Lower grades (higher tonnage) may increase the type and/or size of equipment required to process both feeds. However, if lower grades do occur, additional costs may not solely rest with Denison, as this would need to be reviewed according to the terms of the existing Cigar Lake toll milling JV agreement. The quality of future Cigar Lake feed grades will have a material impact on the results of Hatch's analysis.

4. Toll milling: Test work has not yet been completed on comingled samples (Cigar Lake ore and Wheeler River ore). There is potential for impact on recovery (i.e. if a metallurgical interaction were to be observed in comingled leaching, resulting in lower recovery), capital costs (i.e. if an increased residence time were to be required to maintain recovery in a comingled circuit, and in turn a larger circuit is required), and operating costs (i.e. if higher acid addition were to be required in a comingled circuit), among others.

5. Inability for mining solution to move through Phoenix deposit at rates required: The PFS assumed the operation of ten recovery wells (out of 94 total recovery wells) and solution grades of 10 g/L to produce 6 Mlbs/yr. At these quantities, total solution to be recovery would be 500 L/min out of the wellfield. In order to produce this quantity of solution, the orebody must possess a certain level of permeability. While hydrogeological testing to date indicates that the required permeability is within the range of field test results, there is potential for the overall permeability to be less than current field test results. Should this risk occur, operation of additional wellfields or higher-grade solutions would be required in order to meet annual production targets.

6. Gryphon toll milling: The current McClean Lake operating license from the Canadian Nuclear Safety Commission (CNSC) is valid to 2027. The current approval to operate from the Saskatchewan Ministry of the Environment is valid until 2023. There is a risk that McClean Lake may not receive or may be delayed in receiving future licenses, permits, and approvals, which would impact the ability to process Wheeler River ore at McClean Lake.

7. Gryphon toll milling: The McClean Lake mill is an operating facility. Completing the required modifications to process Wheeler River materials will require detailed execution planning and construction planning. Construction may need to occur over multiple years in order to utilize planned mill shutdowns to complete tie-ins and other critical tasks. As a result, there is a potential risk to project schedule and cost if execution planning is not carefully completed and managed.

8. Gryphon blind boring: The main production shaft requires a high degree of accuracy in terms of verticality to ensure conveyances are able to travel through the shaft. Application of traditional blind boring technology may not be able to produce a vertical shaft within specifications. In order to ensure verticality, a pilot hole or other measures may be required to guide the blind boring to the end target.

9. Underestimation of capital costs: Wheeler River is exposed to the same risks of capital cost increases as any major mining project. While best practices have been utilized to reduce this risk and deliver accurate cost estimates, there is no guarantee that actual capital costs may be within this range.

25 Interpretations and Conclusions

25.1 General Conclusions

In December 2017 Denison retained several engineering consulting organizations to complete a prefeasibility study for the Wheeler River project, which includes the Phoenix and Gryphon orebodies.

This technical report provides a summary of the results and findings from assessments of geological modelling, hydrogeology, rock mechanics, mineral processing and alternatives, infrastructure design, underground mine design, environmental management and permitting, capital and operating cost estimates, and project economic analysis. The level of detail in the investigations of each of these sections is suitably defined to be consistent with what is normally expected from a prefeasibility study of this nature.

The results of the PFS indicate that the Wheeler River project has positive economic conditions under the base case assumptions considered (Section 22). The results should be considered reliable to guide further decision making by Denison on future next steps in the development of the project. This would normally be a definitive feasibility study.

25.2 Geology and Mineral Resources

The Phoenix mineral resource consists of two separate lenses known as Zone A and Zone B located at the Athabasca unconformity approximately 400 m below surface within a 1.1 km long, northeast trending mineralized corridor. Both lenses contain a higher-grade core within a lower grade mineralized envelope and extend along the unconformity roughly overlying the northeast trending WS basement fault. Some mineralization also occurs on the northwest side of the WS Fault but commonly at a slightly lower elevation.

Mineral resources for Phoenix, based on 196 diamond drill holes totalling 89,835 m, were estimated by RPA at a cut-off grade of 0.8% U_3O_8. On a 100% basis, Indicated Resources total 166,000 t at 19.1% U_3O_8 containing 70.2 million lb U_3O_8. Inferred Resources total 9,000 t at 5.8% U_3O_8 containing 1.1 million lb U_3O_8.

The Gryphon deposit is located three kilometres northwest of Phoenix. The Gryphon uranium deposit occurs within southeasterly dipping crystalline basement rocks of the Wollaston Supergroup below the regional sub-Athabasca Basin unconformity. The deposit is located from 520 m to 850 m below surface and has an overall strike length of 610 m, dip length of 390 m and varies in thickness between two metres and 70 m, depending on the number of mineralized lenses present. The mineralized lenses are controlled by reverse fault structures which are largely conformable to the basement stratigraphy and dominant foliation. The A, B and C series of lenses comprise stacked, parallel lenses which plunge to the northeast along the G-Fault, which occurs between hangingwall graphite-rich pelitic gneisses and a more competent pegmatite-dominated footwall. A ubiquitous zone of silicification, the Quartz Pegmatite Assemblage, straddles the G-Fault, and the A, B and C series of lenses occur in the hangingwall of, within, and in the footwall of the Quartz-Pegmatite Assemblage respectively. The D series lenses occur within the pegmatite-dominated footwall along a secondary fault zone, the Basal Fault, or within extensional relay faults which link to the G-Fault. The E series lenses occur along the GFault, up-dip and along strike to the northeast of the A and B series lenses, within the upper basement or at the sub-Athabasca unconformity. Mineralization within the Gryphon deposit lenses is dominated by massive, semi-massive or fracture-hosted uraninite associated with an alteration assemblage comprising hematite, dravitic tourmaline, illite, chlorite and kaolinite. Secondary uranium minerals, including uranophane and carnotite, and sulphides are trace in quantity.

Current mineral resources for Gryphon, based on 214 diamond drill holes totalling 120,351 m, were estimated by RPA at a cut-off grade of 0.2% U_3O_8. On a 100% basis, indicated mineral resources total 1,643,000 t at 1.7% U_3O_8 containing 61.9 million lb U_3O_8. Inferred mineral resources total 73,000 t at 1.2% U_3O_8 containing 1.9 million lb U_3O_8. The Gryphon deposit is a growing, high-grade uranium deposit that belongs to a select group of large basement-hosted uranium deposits in the eastern Athabasca Basin, which includes Eagle Point mine and Millennium deposit, and Rio Tinto's Roughrider deposit. The Gryphon deposit remains open in numerous areas with significant potential for future resource growth. Priority target areas include: (1) Along strike to the northeast of the E series lenses, where both unconformity and basement potential exists; (2) Down plunge of the A and B series lenses; (3) Along strike to the northeast and southwest of the D series lenses; and (4) Within the currently defined D series lenses, where additional high-grade shoots may exist.

CIM Definitions (2014) were followed for classification of mineral resources, and in RPA's opinion, the estimation methodology is consistent with standard industry practice and the Wheeler River property mineral resource estimate is considered to be reasonable and acceptable.

25.3 Hydrogeology

Hydrogeological conditions at the Wheeler River deposits were assessed during drilling programs in 2015 through 2018. Data from the hydraulic testing, pressure transducer systems, water levels surveys, water chemistry, and lab testing of core samples were combined with geological modelling and structural interpretation to build an understanding of the hydrogeological system at both the Phoenix and Gryphon deposits.

Phoenix

The hydrogeological system surrounding the Phoenix deposit has been assessed as it pertains to the proposed ISR mining option. The mine design, with the encapsulating freeze zone and underlying basement formations provide a controlled groundwater system that will greatly simplify control of ISR fluids due to the hydraulic containment. Additionally, operational and closure monitoring will be simplified.

Within the ISR volume, testing to date, although limited, indicates that fluid flow through the mineralised portion is expected to be viable at rates required for designed production. Additional work on in-situ testing is planned to augment the current in-situ hydraulic tests, which were designed for more conventional design evaluation, and core testing. This ISR specific testing will be carried out as part of the Feasibility level program.

Gryphon

The Gryphon deposit has an extensive data set for a deposit of this depth at Pre-Feasibility level of study, with data covering the overlying Athabasca formations, the regional unconformity, and the basement complex. This data was used to model potential inflows to the basement hosted deposit and underground workings, with results comparing very closely to similar mines in the Athabasca Basin. Additional test work will be carried out during Feasibility level programs but aimed more at confirmatory testing of specific structural targets and areas related to specific mine design aspects.

Potential for inflow from the overlying unconformity and Athabasca formations was assessed both numerically and benchmarked against other mines in the Basin. Based on the low hydraulic conductivity of the unconformity in all tests to date at Gryphon and lack of identified connection between the unconformity and the mining zones, this geological feature does not appear to present the same risk at some other Basin locations. Inflow control and risk mitigation from the overlying sedimentary units is considered to be feasible through mine design (avoidance of thin crown pillar) and mining practice (assessment and mitigation through probe and grout) in potential areas of suspected steep angle structure, etc.

High inflow events, if they were to occur, were not numerically modelled, but have been based on design and operational criteria from other mines in the Basin.

25.4 Geotechnical

Rock mass quality throughout the Gryphon deposit typically ranges from predominantly “FAIR” to “GOOD” using established rock mass classification terminology (RMR and Q-Systems). Within the Basement units the intact rock strength can vary between R0 (very weak) to R5 (very strong). ‘Typical’ fresh basement is classified as strong rock (R3, 50-100 MPa). A standard ground support pattern of bolts and screen has been designed to control rock movement during operations. However, during operations there are likely to be localized areas that may have deteriorated ground conditions and may require additional ground support.

Recent hydrogeological assessment completed by SRK indicates relatively low hydraulic conductivity rock mass conditions within the basement units at Gryphon. For geotechnical engineering purposes the rock masses have been considered wet but dewatered (i.e. not subject to significant water pressure or flows). In the event that local water control grout campaigns may be required, rock discontinuities generally appear amenable to cementitious grout injection.

The upper portion of the mining is located ~25 m below the unconformity and below the paleo weathering profile. However, it is recommended to complete this and other high risk development under probe and grout cover programs.

The PFS mine plan proposes two underground mining methods, longitudinal and transverse long hole stoping with cemented rock backfill (CRF). A 15 m level spacing is proposed with longitudinal stopes averaging 5.9 m wide, 17 m along strike. Stope dimensions were analyzed using the empirical open stope design methodology known as Mathews-Potvin or the Stability Graph Method (Hutchinson & Diederichs 1996). Application of the method indicates the deposit is amenable to the planned longitudinal and transverse long hole stoping.

25.5 Phoenix Mining

ISR recovery of the Phoenix orebody has been shown to be an economically attractive and physically viable alternative for the extraction of this resource.

The technology employed in the wellfield drilling is well understood and is in place in numerous jurisdictions globally where it has been shown to operate successfully. The permeability modelling has shown that even under conservative assumptions the orebody is amenable to insitu conditions required for ISR operations.

Testing to date, although limited, indicates that fluid flow through the mineralised portion is expected to be viable at rates required for designed production. Additional work on in-situ testing is planned to augment the current in-situ hydraulic tests, which were designed for more conventional design evaluation, and core testing. This ISR specific testing will be carried out as part of the Feasibility level program.

In the absence of the required impermeable lithological unit above the orebody horizon, the ground freezing design is a practical risk mitigation approach that will keep the orebody isolated from the regional groundwater which will serve to maximize product recovery, minimize dilution and eliminate environmental impacts.

It is expected that the planning and implementation of operations will benefit directly from the existing operational practices in place at other sites, suitably modified and factored for the extremely high grades present at Phoenix.

25.6 Gryphon Mining

Based upon the information currently available, ground conditions in the mine are, in general, anticipated to be reasonable. The main geotechnical risks for the project is related to mining near the unconformity; hence the permanent 25 metre crown pillar and the presence of faults which may intersect the area. The faults will affect access development, requiring appropriate ground control measures to be implemented. There are also certain areas of the hanging and footwall with unfavourable stope wall conditions that may lead to localised, increased waste dilution and/or ore loss.

The preliminary hydrological assessment indicated initial water inflows of 258 m3/hr (58 m3/hr from process water and 200 m3/hr from groundwater) for the underground mine. If higher rates are encountered, there may be adverse impacts on the mine production schedule, particularly the lateral development advance rates. The dewatering system will be designed to handle emergency inflows of 1,650 m3/hr (anticipated inflows through unconformity plus 10% extra) for the underground mine. If higher rates are encountered, the designed dewatering system may not be sufficient, and the mine may be forced to be abandoned until the inflows can be controlled.

The current ventilation design is based upon local legislation and comparative case study identification. If the proposed ventilation system becomes insufficient due to recommendations from Arcadis (pending), or changes in regulations, the ability to expand the volume of air reporting underground is limited.

The lateral development advance rates used in the PFS are in accordance with good industry standards, but there is a risk that these are not consistently achieved in practice if the hydrology or geo-mechanical assumptions are understated. In particular, the timely excavation of the declines and level access development in the early stages of mine development is critical to attaining the ore production schedule.

25.7 Phoenix Mineral Processing

There are limited options for validating the successful operation of an ISR operation at this level of project definition. Denison Mines retained the services of knowledgeable uranium ISR development service providers in the U.S. Specific metallurgical tests, namely an agitation (bottle roll) and a column leach test were performed with high grade core material collected from the Phoenix deposit.

Metallurgical test results from both the agitation and column leach tests indicate that over 90% of the uranium resources can be extracted. Testing also highlighted the possibility of cumulative buildup of contaminants in the uranium bearing solution with detrimental effect on the direct uranium precipitation process. An iron removal step was introduced and tested with success eliminating the accumulation of dissolved iron and other impurities in the lixiviant enabling the production of a high-quality uranium product. The uranium recovery process, simple in nature, consists of adjusting the pH, adding hydrogen peroxide and re-fortifying the BLS for re-injection in the wellfield. The overall uranium recovery rate has been estimated at 98.5% and most losses reporting to the iron removal waste stream.

Demonstrating the ability to restore the groundwater to pre-mining quality is critical to the planning of an ISR mining option. Preliminary test work completed after the column leach test indicates neutralization of the ground after low pH mining can be completed. Results indicate the concentration of some elements does exceed pre-mining baseline levels but over 60% of the elements monitored were below baseline levels. Additional testing and optimization is required to improve restoration performance and modelling of post mining ground water quality and potential environmental impacts. Groundwater restoration activity will necessitate the use of reverse osmosis as a water treatment process. The permeate will be re-injected in the ground and the concentrate stream evaporated. Salts will be collected in tote bags and de-ionized water combined with the permeate stream generating no effluent to be disposed of.

25.8 Gryphon Mineral Processing

The results of the metallurgical test work program indicate that the Gryphon deposit is amenable to recovery utilizing the existing McClean Lake mill flowsheet. Moreover, Gryphon is amenable for processing at similar conditions to those currently used in the McClean Lake mill. Overall process recovery based on metallurgical test work to date has been estimated at 98.4% (co-processed with Cigar Lake ore). Gryphon production levels align well with known available capacity in the McClean Lake mill. Production scenarios do not exceed the McClean Lake mill's currently licenced capacity of 24 Mlbs/a U_3O_8 production. Gryphon ore be processed in conjunction with Cigar Lake Phase 2 production. Cigar Lake Phase 2 production, which is not fully defined, is assumed to be 15 Mlbs/a U_3O_8, allowing for Gryphon ore processing at 9 Mlbs/a U_3O_8.

Processing the Gryphon deposit will require modifications to the McClean Lake mill. These modifications include: expansion of the leaching circuit, the addition of a filtration system to complement the Counter Current Decantation (CCD) circuit capacity, the installation of an additional tailings thickener, and expansion of the acid plant. Upgrades are also required throughout the mill to permit production at the full 24 Mlbs/a U_3O_8 licenced capacity. Total capital costs for Gryphon modifications are $101.7M CAD, excluding Owners' costs. The operating cost for Gryphon co-milling is estimated to be $5.71/lb U_3O_8.

25.9 Environment

The project will be required to complete a joint environmental assessment in accordance with provincial and federal legislation. This process will require approximately 24 to 36 months following the approval of a project description. A thorough, comprehensive environmental database has been collected to support the anticipated environmental assessment.

Based on the existing understanding of the proposed project, there are no environmental fatal flaws identified and there is no reason to assume the project could not successfully complete an environmental assessment which could be acceptable to the federal and provincial regulatory regimes and the project's stakeholders.

25.10 Social Considerations and Stakeholder Engagement

Denison initiated a thorough Stakeholder Engagement Management Plan in 2016. This plan was developed in accordance with provincial, federal and international guidance. Since its initiation over 20 face-to-face meetings have taken place between Denison and stakeholder communities, First Nations and Metis leadership, community economic development representatives and community residents. These engagements have allowed for the inclusion of Traditional knowledge into the design of the environmental baseline programs as well as portions of the engineering designs presented within this technical report.

26 Recommendations

A review of each of the areas of the PFS has produced the following suite of recommendations for future work programs to further develop the design base and to address the project risk factors and realize identified opportunities.

This report section summarizes all these principal recommendations that have arisen from the PFS. Where possible costs have been included, in the exceptions to this the costs have been included in the capital and/or operating cost estimates for the project.

26.1 Phoenix

26.1.1 Mine Hydrogeological

Sufficient data is available for assessment of the overlying Athabasca formations and basement outside of the proposed freeze envelope, so any additional testing would be to evaluate structure that could impact operational and closure flow systems in the surrounding rock mass. This would entail additional packer based hydraulic testing and groundwater sampling targeting structure where possible.

Within the frozen shell and orebody, small scale (short test zone) tests in the sandstone and orebody itself should be carried out, with the objectives of determining:

- Small scale hydraulic parameters (hydraulic conductivity, storability, and connectivity) of the rock and orebody.
- Cross hole testing from test screens within the orebody to simulate lixiviant flow between wells. These tests should target zones ranging from high fracture frequency to the more competent ore to compare fluid flow across different portion of the orebody. Testing should also target different grade portions to work in with evaluation of ISR production rates.
- Installation of additional VWP transducer strings should proceed any cross-hole tests to allow for pressure response monitoring around and within the orebody.

Additional lab work to test for “chemical erosion” of fractures within the ore should be carried out if feasible. This effect would have significant impact on lixiviant flow through the orebody as lixiviant flow would increase with time and would be a major difference on hydraulic flow within the high-grade Phoenix deposit compared to low-grade uranium deposits currently in production.

26.1.2 Ground Freezing

The geometry and well spacing of the freeze curtain design could be optimized with a view to reducing the number of required wells. Careful review of the project schedule should be compared to the varying lead times to achieve freeze thicknesses against available schedule time.

26.1.3 Metallurgical Testing

It is recommended to simulate the planned ISR operation in a large-scale setting. Using a pipe diameter of a minimum of 6” and a length similar to the expected well spacing in the field, ground rock would be packed within the pipe. The material particle size distribution and packing methodology would a close approximation of appropriate in situ permeability value.

Lixiviant would be pumped through the pipeline on a continuous basis. The solution would be subjected to ISR recovery processes as defined during the PFS metallurgical testing programs. The pilot plant installation would remove iron and other metals prior to the uranium direct precipitation step. Yellowcake product would be characterized and accumulated over time. The BLS stream would be re-fortified and re-injected in the pipeline.

Once the uranium resource has been successfully extracted from the material, ground restoration activities would be initiated. It would consist of completing ground water sweeps, forced neutralization and periods of soaking. The lixiviant displaced by the groundwater would be recovered until uranium concentration is deemed sufficiently dilute. The water exiting the pipeline would feed an RO unit where permeate water re-used as part of the restoration process and concentrate stream would be characterized.

26.1.4 Uranium Recovery and Processing

During feasibility there are opportunities to consider the capture additional rare earth elements that were not quantified during the PFS, this could have the impact of improving revenue and project economics.

Bench testing has indicated that the possibility exists of higher uranium solution concentration levels above the 10 g/L assumed in the PFS. During the feasibility study attention should be paid to the capacity of the plant to be able to potentially produce at a higher level should concentrations be higher than expected without having to reduce flowrates, effectively increasing overall uranium production levels.

The possibility of processing Gryphon ore at the Phoenix ISR plant should be examined as the costs for modifications to the McClean Lake mill are material and operating and transportation costs are also significant. A front-end leaching circuit would be required, and would be evaluated against the cost of the base case currently included in the PFS.

26.2 Gryphon

26.2.1 Mine Geotechnical

The following points summarize the requirements and recommendations for further advancement of the project in subsequent project stages.

- Review core logging and data collection procedures. Recommend two data collection systems, RMR76 or RMR89 and Q. A geotechnical database quality control review should be completed to ensure a quality assured design data set moving forward. Continue to collect point load test data to expand the intact rock strength data set.
- To advance the project through to feasibility level additional drilling is recommended. Holes that target the highest grade and widest lenses of economic ore is preferred. The conditions are likely to represent the poorer conditions and in a mine block that is at the heart of the mines production.
- Additional laboratory testing is recommended to confirm intact properties which will result in more accurate assessments of both intact and rock mass properties, and subsequently more accurate analysis and numerical simulations. Test work should focus on altered basement in the HW and ore zone of economic mineralization. Testing should also include some direct shear testing of representative discontinuities from drill core to aid in defining rock mass strength properties.
- Develop a series of geotechnical models to support more detailed underground mine design. Such models will allow the development of a higher confidence overall geotechnical domain model.
    - 3D clay alteration models, by intensity.

- 3D RQD models. This will be useful in defining the spatial representation of the various rock mass classes and assessing HW stability and HW over break limits. It will also be used to develop the 3D geotechnical domain model.
- 3D RMR and intact rock strength models when more data becomes available. These will serve, along with the RQD model, to fully define the approximate boundaries of the 3D geotechnical domains to provide location specific stope stability assessments and a more accurate numerical model.

26.2.2 Mine Hydrogeological

Feasibility hydrogeological testing at Gryphon should target structural features (E-W sub vertical features) that have not been well tested by current drilling due to drill orientation bias. This work would be to determine what level of risk these structures could impose as high inflow features as they would continue down into the basement and through the underground development. Planned probe and grout should be feasible to reduce risk in development areas, but occurrence in open stoping where mitigation would be more difficult should be assessed.

If these features are found to be more transmissive and penetrate the basement within the mine envelope, a series of cross hole pumping tests would be recommended.

26.2.3 Mining

The following points are the recommendations for future work on the Gryphon underground design in subsequent studies, these are purposefully described as specific recommendations, with specific actions accordingly.

- Investigate opportunities to recover additional resource material via more selective mining methods, varying cut-off grades appropriately, in addition to planned longhole stopes;
- Investigate opportunities to reduce time to production by adopting top down mining or cut and fill in the upper areas of the mine;
- Review options to create additional mining fronts (increased equipment, manpower, ventilation, etc.) to recover tonnes as quickly as possible, and to stockpile at mill, shorten the mine life, but with deferred revenues; additionally, also review options to create additional mining fronts to mine “selectively” while maintaining the target production rates. This could potentially provide high-grade initial tonnes but reduce overall recovery with additional sill pillars;
- Evaluate the opportunity to eliminate services in the ventilation shaft, other than secondary egress. Services include backfill lines, fibre optics, and redundant dewatering lines. Currently the proposed secondary egress utilizes fixed guides. Without services in the shaft there is a better potential to utilized rope guides, which will reduce capital costs and LOM operating costs;
- Further review the potential to optimize the delivery of shotcrete to the underground operations, via slickline or borehole;
- Since the greatest risk to encountering major inflows of water is during the initial off-shaft development, i.e. closest to unconformity, further evaluate the opportunity to install dewatering system earlier in the mine life via borehole pumps or at shaft bottom should be evaluated. This would allow the entire dewatering system to be readily available prior to initial off-shaft development;
- Review ultimate depth of shaft(s) and the potential to decrease ramp up time to full production, reduced trucking vs increase shaft cost, and limitations of blind boring; and

- Review opportunities to decrease ore development costs including: extend heights of sub-levels, eliminate duplication of ground support in ore headings with both shotcrete and bolting requirements, or reducing size of drifts with jumbo slash recovery as stopes are mined.

26.2.4 Metallurgical Testing

To further validate the performance of processing Wheeler River ores at the McClean Lake mill, it is recommended that further test work be conducted in the next study phase. Additional test work is recommended for all process circuits to provide further metallurgical characterization of the ore and provide definition to support detailed engineering design. This test work should include:

- Comingled leach tests using Wheeler and Cigar Lake ores.
- Comingled settling and filtration tests using Wheeler and Cigar Lake ores.
- Tailings aging test work and evaluation of stability in TMF.
- Variability comminution test work.
- Gryphon ore bulk material property test work and size distribution.
- Additional variability testing is recommended on both Phoenix and Gryphon ores to further explore any variation in processing response throughout the extremes of the deposit. It is recommended that testing be completed on unique samples from each lithological group, and well as non-composited samples from the high and low-grade extremes of the deposits.
- Due to the high clay content in Gryphon ores, rheology testing is recommended to support agitation and pumping design.
- Final neutralization tests should be repeated with higher reagent addition to demonstrate regulatory limit radium removal.

For the processing scope outlined in Section 17, Hatch recommends a budget of $1.5M to complete metallurgical test work to the end of a feasibility study.

26.2.5 Metallurgical Design and Implementation

The following items are recommended for future work in a feasibility study:

- Detailed review and study of the existing calciner to confirm it is capable of 24 Mlbs/a U_3O_8 production. This would include a review of the current ADU product feed to the calciner (including element and moisture content), physical design loading considerations, and a review of the associated off-gas system.
- Minor element deportment and water balance assessment and confirmation that there are no impacts on the existing Water Treatment Plant (WTP).
- A detailed execution plan and construction schedule should be completed for the project. This will improve confidence in schedule and installation costs given the brownfield nature of the mill.
- For the processing scope outlined in Section 17 (Recovery Methods), Hatch recommends a budget of $3.5M to complete engineering design to the end of a feasibility study.

26.3 Environmental, Regulatory and Community Relations

Continuation of the collection of environmental data throughout the next phases of engineering is recommended to increase the already robust understanding of the baseline conditions of the project and its immediate and regional environment.

Continued engagement with the existing stakeholders throughout the advancement of the project is recommended. This continued engagement will allow for the integration of additional Traditional knowledge, as it becomes available, into the engineering designs of the project as they are advanced to the next level.

26.4 Recommendations Cost Summary

Table 26-1 summarizes the expected costs to address the assembled collection of recommendations for the Wheeler EA and for Phoenix feasibility study. Table 26-2 summarizes the recommended work for the development of the Gryphon deposit. These costs have been included in the capital and/or operating costs of the project as presented in this report.

Table 26-1: EA and Phoenix Future Work Cost Estimates

Environmental Assessment and Field Work / Test Work	
Environmental Assessment and Project Description	$ 2,000,000
Hydrogeological Testing	$ 500,000
Freeze Cap Evaluation	$ 1,500,000
ISR Pilot Plant	$ 1,000,000
Subtotal Field Work/Test Work	**$ 5,000,000**
Phoenix Feasibility Study	
Geological Modelling	$ 150,000
Hydrogeological Modelling	$ 150,000
Freeze Cap Modelling	$ 150,000
Well Field Design	$ 250,000
ISR Plant Design	$ 1,500,000
Surface Infrastructure Design	$ 1,250,000
Class 3 Cost Estimation	$ 1,000,000
Execution Plan	$ 250,000
Manage/Assemble Feasibility Study	$ 575,000
Subtotal Feasibility Study	**$ 5,275,000**
Phoenix Total	**$ 10,275,000**

Table 26-2: Gryphon Future Work Cost Estimates

Field Work / Test Work	
Additional Shaft Test Holes	$ 300,000
Geotechnical and Hydrogeological Drilling	$ 200,000
Metallurgical Test Work	$ 1,500,000
Subtotal Field Work/Test Work	**$ 2,000,000**
Feasibility Study	
Geological Modelling	$ 150,000
Geotechnical Modelling	$ 150,000
Hydrogeological Modelling	$ 150,000
Shaft Design	$ 950,000
Mine Design	$ 550,000
Underground Infrastructure Design	$ 750,000
Processing Plan Upgrade Design	$ 750,000
Surface Infrastructure Design	$ 500,000
Class 3 Cost Estimation	$ 1,000,000
Execution Plan	$ 250,000
Mineral Processing Feasibility study	$ 3,500,000
Manage/Assemble Feasibility Study	$ 575,000
Subtotal Feasibility Study	**$ 9,275,000**
Gryphon Total	**$ 11,275,000**
Wheeler River Grand Total	**$ 21,550,000**

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28 Certificates of Qualified Persons

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CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018

I, Mark William Liskowich do hereby certify that:

1) I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office at suite 205, 2100 Airport Drive, Saskatoon Saskatchewan;
2) I am a graduate of the University of Regina, Saskatchewan, where, in 1989 I obtained a BSc degree in Geology through the Dept. of Sciences. I have practiced my current profession continuously since 1989. My principal experience is in the areas of environmental, permitting and social management of mineral exploration and mining projects;
3) I am a professional geoscientist registered with the Association of Professional Engineers and Geoscientists of Saskatchewan - PGeo License (#10005) and the Professional Engineers and Geoscientists Newfoundland and Labrador (#09424);
4) I have not personally inspected the subject project site;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;
6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a contributing of this report and responsible for sections 1.1, 1.2.2, 1.2.13, 1.2.14, 1.2.16, 1.2.17, 2, 3, 4.4, 4.5, 13.1.8, 17.2.6, 18.18, 18.19, 20, 21.1, 21.2, 21.3.1, 21.3.6, 21.4, 21.4.6, 21.4.7, 21.5, 23, 24.3, 25.1, 25.9, 25.10, 26.3 and 26.4 and accept professional responsibility for these sections of this technical report;
8) I have had prior involvement with the subject property; I was responsible for the environmental and social sections of the 2016 preliminary economic assessment;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;
10) SRK Consulting (Canada) Inc. was retained by Denison Mines Corp. to prepare a preliminary economic assessment audit of the Wheeler River uranium project in 2016 and to review the incorporation of such report in this technical report. The preceding report was based on a site visit, a review of project files and discussions with Denison Mines Corp. personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corp.; and
12) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Signed and Sealed"

____

Mark Liskowich, PGeo,
Principal Consultant
Dated October 30, 2018

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018.

I, Mark Hatton do hereby certify that:

1) I am a Senior Project Manager with Stantec with business address of 1-1760 Regent Street, Sudbury ON P3E 3Z8;
2) This certificate applies to the technical report entitled "*Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada*" (the Technical Report), dated September 24 2018;
3) I am a graduate of Queen's University, ON, where, in 1992 I obtained a BSc degree in Applied Science. I am a member in good standing of the Association of Professional Engineers of Ontario, registration 90433939;
4) I have not visited the Property;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;
6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a contributing author for this report and responsible or partially responsibility for sections 1.2.5, 1.2.9, 1.2.14, 1.2.16, 1.2.17, 2.1, 2.2, 2.3, 3.0, 15.1, 15.3, 16.1, 16.1.2, 16.5, 16.6, 18.6, 18.7, 18.8, 18.22, 21.0, 21.1, 21.2, 21.4, 21.4.1, 21.4.2, 21.4.3, 21.4.4, 21.4.7, 21.5, 21.5.3, 24.1.2, 24.3, 25.6, 26.2.3, and 26.4, and accept professional responsibility for these sections or portions of these sections of this technical report;
8) Prior to this study, I have had no past involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;
10) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corp.; and
11) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

["signed and sealed"]
Mark Hatton, P.Eng. Stantec
Senior Project Manager
Dated October 30, 2018

HATCH

CERTIFICATE OF QUALIFIED PERSON

I, William McCombe, P.Eng., do hereby certify that:

1. I am a Senior Metallurgist, of Hatch Ltd. ("**Hatch**"), a corporation with a business address of 2800 Speakman Drive, Mississauga, Ontario, L5K 2R7.

2. I am an author of a technical report entitled "Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada", with an effective date of September 28, 2018 (the "**Technical Report**").

3. I am a graduate of Queen's University, Kingston, Ontario, Canada in 2005 with a degree in B. Sc. Mining Engineering (Mineral Processing).

4. From 2005 to present I have been actively employed as an engineer in the area of extractive metallurgy. My relevant experience for the purpose of this Technical Report is:

- Senior Metallurgist at Hatch Ltd. from 2005 to present, with extensive experience in metallurgical test work program analysis, engineering design, management, and project execution on a number of projects and commodities for various clients.
- I was the lead process engineer (Senior Metallurgist) and commissioning manager for the Cigar Lake Hydrogen Mitigation Project (leach circuit re-design and restart) at the McClean Lake JEB Mill.

5. I am a member, in good standing, of PEO in the Province of Ontario, member #100098890, and with APEGS in the Province of Saskatchewan, member #28650.

6. I have read the definition of "qualified person" set out in *National Instrument 43–101 Standards of Disclosure for Mineral Projects* ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" within the meaning of NI 43-101**.**

7. I have not visited the Wheeler River site. I have visited the McClean Lake JEB Mill multiple times as a consultant of Orano, and visited the site specifically for the Wheeler River Project on June 5-6, 2018.

8. I am responsible for sections 1.2.11, 13.2, 17.2 (except 17.2.1 and 17.2.6), 21.4.5, 21.5.4, 25.8, 26.2.4 and 26.2.5, and co-authored 1.2.14, 1.2.16, 1.2.17, 2.1, 2.2, 2.3, 2.4, 3, 21.1, 21.2, 21.4, 21.4.7, 21.5, 24.3, and 26.4 of the Technical Report.

9. I am independent of the issuer, Denison Mines Corp., applying all of the tests in Section 1.5 of NI 43-101.

10. As a Senior Metallurgist, I have not had prior involvement with the Wheeler River property that is the subject of the Technical Report.

11. I have read NI 43-101 and the parts of Technical Report that I am responsible for have been prepared in compliance with that Instrument.

12. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th day of October, 2018, in Mississauga, Ontario.

["signed and sealed"]
William McCombe, P. Eng.
Senior Metallurgist
Hatch Ltd.

Sheridan Science and Technology Park, 2800 Speakman Drive, Mississauga, Ontario, Canada L5K 2R7
Tel: +1 (905) 855 7600 www.hatch.com

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018

I, Douglass H Graves do hereby certify that:

1) I am a Principal with the firm of Woodard & Curran with an office at Bozeman, Montana, USA;
2) I graduated with a Bachelor of Science degree in Watershed Sciences from Colorado State University in 1975. I am also a graduate with a Bachelor of Science degree in Civil Engineering from Montana State University in 1982.
   I have worked as a consulting Engineer for 40 years. My experience has encompassed infrastructure design, mine construction oversight, cost estimating and control, economic analyses, feasibility studies, equipment selection, design, construction management and mine closure/reclamation for numerous metal mining operations, conventional uranium and uranium ISR facilities. I have either been responsible for or the engineer of record for the design and/or construction of five uranium ISR central processing facilities, two uranium ISR satellite plants and numerous technical and financial evaluations for other uranium processing facilities in Wyoming, Colorado, Texas and New Mexico. I have also been responsible for or the engineer of record for numerous metal and uranium mine decommissioning and reclamation projects over the past 35 years. Some of the mining properties I have been involved with include:

Lance Uranium ISR Projects	Hansen Uranium
Lost Creek Uranium	Jab-Antelope Uranium
Moore Ranch Uranium	Climax Molybdenum
Nichols Ranch Uranium	Henderson Molybdenum
Ludeman Uranium	Bagdad Copper
Ross Creek Uranium	Sierrita Copper
Willow Creek Uranium	Globe Copper
Churchrock Uranium	Morenci Copper

3) I am a Professional Engineer in Wyoming, Montana, Colorado, South Carolina, Arizona, Idaho, Michigan, Oklahoma and Missouri, a P. Eng. in Alberta and Saskatchewan, Canada, a Registered Member of the Society for Mining, Metallurgy and Exploration (SME), Mining Associates of Wyoming (MAW), Montana Mining Association (MMA), American Exploration and Mining Association (AEMA);
4) I did not personally inspect the subject project;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;
6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a contributing author for this report and responsible for part or all of sections 1.2.5, 1.2.8, 1.2.10, 1.2.16, 1.2.17, 2.1, 2.2, 2.3, 3.0, 13.0, 13.1, 15.1, 15.2, 16.1, 16.1.1, 16.4, 16.6, 17.1, 24.3, 25.5, 25.7, 26.1.3, 26.1.4 and 26.4 and accept professional responsibility for these sections of this technical report;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;
10) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corp.; and
11) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and sealed Douglass H. Graves

Douglass H. Graves, P.E., Professional Engineer Wyoming PE 4845 and SME Registered Member 4149627

Dated October 30, 2018

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018.

I, Mark Mathisen do hereby certify that:

1) I am a Principal Geologist with RPA (USA) Ltd. with an office at Suite 505, 143 Union Boulevard, Lakewood, Colorado, USA 80228;
2) I am a graduate of Colorado School of Mines in 1984 with a B.Sc. degree in Geophysical Engineering;
3) I am a Registered Professional Geologist in the State of Wyoming (No. PG-2821), a Certified Professional Geologist with the American Institute of Professional Geologists (No. CPG-11648), and a Registered Member of SME (RM #04156896). I have worked as a geologist for a total of 22 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Mineral Resource estimation and preparation of NI 43-101 Technical Reports.
   - Director, Project Resources, with Denison Mines Corp., responsible for resource evaluation and reporting for uranium projects in the USA, Canada, Africa, and Mongolia.
   - Project Geologist with Energy Fuels Nuclear, Inc., responsible for planning and direction of field activities and project development for an in situ leach uranium project in the USA. Cost analysis software development.
   - Design and direction of geophysical programs for US and international base metal and gold exploration joint venture programs.
4) I have personally inspected the subject project on March 23 to 25, 2015, September 12, 2016, and September 21 to 22, 2017;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;
6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a contributing author for this report and responsible for sections 1.2.1 – 1.2.4, 2.1, 2.2, 2.4, 4.1–4.3, 5-12, 14 and 25.2 and accept professional responsibility for these sections of this technical report;
8) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;
9) I have prepared previous technical reports on the subject property, including technical reports on an updated Mineral Resource estimate for the Wheeler River Property dated November 25, 2015 and March 15, 2018;
10) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River Uranium Project or securities of Denison Mines Corp.; and
11) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed *"Mark Mathisen"*
Mark Mathisen, CPG
Principal Geologist

Dated October 30, 2018

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018

I, William Roscoe do hereby certify that:

1) I am a Principal Geologist with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7;

2) I am a graduate of Queen's University, Kingston, Ontario, in 1966 with a Bachelor of Science degree in Geological Engineering, McGill University, Montreal, Quebec, in 1969 with a Master of Science degree in Geological Sciences and in 1973 a Ph.D. degree in Geological Sciences;

3) I am registered as a Professional Engineer in the Province of Ontario (Reg. #39633011) and in the Province of British Columbia (Reg. # 30343). I have worked as a geologist for a total of 50 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Thirty-six years' experience as a Consulting Geologist across Canada and in many other countries
   - Preparation of numerous reviews and technical reports on exploration and mining projects around the world for due diligence and regulatory requirements, including estimation and auditing of mineral resources
   - Senior Geologist in charge of mineral exploration in southern Ontario and Québec
   - Exploration Geologist with a major Canadian mining company in charge of exploration projects in New Brunswick, Nova Scotia, and Newfoundland

4) I have personally inspected the subject project on June 16, 2014;

5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;

6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;

7) I am a contributing author for this report and responsible for sections 1.2.1 – 1.2.4, 2.1, 2.2, 2.4, 4.1 – 4.3, 5-12, 14 and 25.2 and accept professional responsibility for these sections of this technical report;

8) I have prepared previous technical reports on the subject property, including technical reports on an updated Mineral Resource estimate for the Phoenix deposit dated December 31, 2012 and June 17, 2014; a technical report on a Mineral Resource estimate for the Wheeler River Property dated November 25, 2015; mineral resources estimate in preliminary economic assessment audit of the Wheeler River uranium project in 2016 and to review the incorporation of such report in this technical report;

9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;

10) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corp.; and

11) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed *"William Roscoe"*
William Roscoe, P.Eng.
Principal Geologist

Dated October 30, 2018

RPA 55 University Ave. Suite 501 | Toronto, ON, Canada M5J 2H7 | T +1 (416) 947 0907 **www.rpacan.com**

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018

I, Gordon Graham do hereby certify that:

1) I am a Vice President with the firm of Engcomp with an office at Saskatoon,SK,Canada;
2) I am a graduate of Queen's University, where, in 1988 I obtained a BASc degree in Mining Engineering through the Faculty of Applied Science. I have practiced my current profession continuously since 1988. My principal experience is in the areas of mine engineering, mine operations management and mining project development;
3) I am a professional engineer registered with the Association of Professional Engineers and Geoscientists of Saskatchewan – P.Eng License No.: 39771;
4) I have not visited the Wheeler River site;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;
6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a contributing author for this report and responsible for sections 1.2.12, 17.2.1, 18 with the exception of sections, 18.5, 18.6, 18.7, 18.8, 18.18, 18.19, 18.22, 24.1, 24.1.1, and 24.2 and a contributing author to sections 1.2.16, 1.2.17, 2.1, 2.2, 2.3, 2.4, 24.3 and 26.4 accept professional responsibility for these sections of this technical report;
8) I have had prior involvement with the subject property; as an engagement to complete a scoping level study of large diameter petroleum drilling as a primary recovery methodology for the Phoenix deposit;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;
10) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corp.; and
11) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

["signed and sealed"]
Gordon Graham,
Vice President, Mining, Engcomp
Dated October 30, 2018

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018

I, Geoffrey Allan Wilkie do hereby certify that:

1) I am a Specialist Cost Consultant with the firm of Engcomp with an office at Saskatoon, Saskatchewan, Canada;
2) I am a graduate of the University of British Columbia, BC, where, in 1986. I obtained a BASc degree in Civil Engineering through the Dept. of Applied Science. My principal experience is in the area of cost estimating;
3) I am a professional engineer registered with the Association of Professional Engineers and Geoscientists of Saskatchewan – Peng. License No.: 11116;
4) I have not personally inspected/visited the subject project;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;
6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a contributing author for this report and responsible for sections 17.2.1, 21.1, 21.2, 21.3.1 to 21.3.5, 21.4, 21.4.7, 21.5, 21.5.1, 21.5.2, 21.6, 24.3 and 26.4 and accept professional responsibility for these sections of this technical report;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;
10) Engcomp was retained by Denison Mines Corp. to prepare a prefeasibility level engineering, design and cost estimates for the On-Site Surface Infrastructure aspects of the Wheeler River uranium project in 2018 and to incorporate such engineering, designs and estimates in this technical report;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corp.; and
12) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and Sealed

Name, Geoffrey Allan Wilkie, P.Eng.
Title, Cost Consultant Specialist
Dated October 30, 2018

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018

I, Greg Newman do hereby certify that:

1) I am a President with the firm of Newmans Geotechnique Inc. with an office at Saskatoon;
2) I am a graduate of the University of Saskatchewan, SK, where, in 1992 I obtained a BSc degree in Mechanical Engineering through the faculty of Engineering. I am also a graduate of the University, where I obtained a M.Sc degree in Geotechnical Engineering in 1995. I have practiced my current profession continuously since 1992. My principal experience is in the areas of artificial ground freezing;
3) I am a professional geoscientist registered with the Association of Professional Engineers and Geoscientists of Saskatchewan - P.Eng. License No.: 9054;
4) I have not personally inspected the project site;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;
6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a contributing author for this report and responsible for sections 1.2.14, 1.2.16-17, 2.1-2.3, 3, 16.4.2-3, 18.5, 21.1-21.3, 21.4.7, 24.3, 25.3, 26.1.2 and 26.4 and accept professional responsibility for these sections of this technical report;
8) I have had no prior involvement with the subject property, except as a consultant on a very early scoping study as part of the SRK 2016 task (see item 10 below);
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;
10) SRK Consulting (Canada) Inc. was retained by Denison Mines Corp. to prepare a preliminary economic assessment audit of the Wheeler River uranium project in 2016 and to review the incorporation of such report in this technical report;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corp.; and
12) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

[Signed] Greg Newman

Greg Newman, P.Eng.
President, NGI
Dated October 30, 2018

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018

I, J. Roland Tosney, B.E., M.Sc., P.Eng., do hereby certify that:

1) I am a Senior Mining Geotechnical Engineer with the firm of North Rock Mining Solutions Inc. with an office at 15 South Point Lane Corman Park SK S7T1C1;
2) I am a graduate of the University of Saskatchewan, SK, where, in 1998 I obtained a B.E. degree in Geological Engineering, and in 2001 I obtained a M.Sc. degree in Mining Geotechnical Engineering, both through the College of Engineering. I have practiced my current profession continuously since 1998. My principal experience is in the areas of open pit and underground mining geotechnics and rock mechanics;
3) I am a professional engineer registered with the Association of Professional Engineers and Geoscientists of Saskatchewan – P.Eng. License No.: 29820;
4) I have personally inspected the subject project on September 26, 2017;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;
6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a contributing author for this report for sections 1.2.16, 1.2.17, 2.1, 2.2, 2.3, 2.4, 21.4.7, 24.3 and 26.4, and responsible for sections 1.2.7, 16.3, 25.4, and 26.2.1 and accept professional responsibility for these sections of this technical report;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;
10) North Rock Mining Solutions Inc. was retained by Denison Mines Corp. to prepare a preliminary mining geotechnical assessment of the Gryphon underground component of the Wheeler River project in 2017 and to review the incorporation of such report in this technical report. The preceding report was based on a site visit, a review of project files and discussions with Denison Mines Corp. personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corp.; and
12) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

[SIGNED]

J. Roland Tosney,
Senior Mining Geotechnical Engineer
Dated October 30, 2018

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018

I, Michael D. Royle do hereby certify that:

1) I am a Principal Hydrogeologist with the firm of SRK Canada (INC) with an office at Suite 22100, 1066 West Hastings Street, Vancouver BC;
2) I am a graduate of the University of British Columbia, BC, where, in 1987 I obtained a BSc degree in Geology through the Dept. of Sciences. I am also a graduate of the University of New South Wales, NSW, Australia, where I obtained a M.App.Sc. degree in Hydrogeology and Groundwater Management. I have practiced my current profession continuously since 1989. My principal experience is in the areas of mining hydrogeology;
3) I am a professional geoscientist registered with the Association of Professional Engineers and Geoscientists of Saskatchewan - PGeo License No.: 30586;
4) I have personally inspected the subject project on June 21 to 24, 2016;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;
6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a contributing author for this report and responsible for sections 1.2.6, 1.2.16, 1.2.17, 2.1, 2.2, 2.3, 2.4, 16.2, 21.3.1, 21.4.7, 24.3.1, 24.3.2, 25.3, 26.1.1, 26.2.2, and 26.4 and accept professional responsibility for these sections of this technical report;
8) I have had prior involvement with the subject property as listed in item 10 below;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;
10) SRK Consulting (Canada) Inc. was retained by Denison Mines Corp. to prepare a preliminary economic assessment audit of the Wheeler River uranium project in 2016 and to review the incorporation of such report in this technical report. SRK was also retained in 2015 through 2018 to design and implement detailed hydrogeological site characterisation and assessment of the Wheeler River project to support ongoing exploration and the project PFS. The preceding report was based on a site visit, a review of project files and discussions with Denison Mines Corp. personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corp.; and
12) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

[signed and sealed]

Michael D. Royle
Principal Hydrogeologist
Dated October 30, 2018

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Prefeasibility Study Report for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date September 24, 2018

I, Michael Selby do hereby certify that:

1) I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. with an office at Sudbury;
2) I am a graduate of the Queen's University, ON, where, in 2001 I obtained a BSc degree in Mining Engineering through the Faculty of Applied Science. My principal experience is in the areas of underground hard rock mining;
3) I am a professional engineer registered with the Association of Professional Engineers and Geoscientists of Saskatchewan - License No.: 30781;
4) I have personally inspected the subject project on September 26, 2017;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101;
6) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a contributing author for this report and responsible for sections 1.2.15, 4.4, 19, 22 and 26.4 as well as contributing coauthor of sections 1.2.2, 1.2.16, 1.2.17, 2.1 to 2.4, 3, 24.3 and accept professional responsibility for these sections of this technical report;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith and Form 43-101F1;
10) SRK Consulting (Canada) Inc. was retained by Denison Mines Corp. to prepare a prefeasibility study of the Wheeler River uranium project. The preceding report was based on a site visit, a review of project files and discussions with Denison Mines Corp. personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corp.; and
12) As at the effective date of this report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

["signed and sealed"]
Michael Selby, PEng
Principal Consultant
Dated October 30, 2018