# Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada

**Report Prepared for**

## Denison Mines Corporation

**Report Prepared by**

SRK Consulting (Canada) Inc.
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Cover: Exploration drilling at Wheeler River.

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# Table of Contents

# List of Tables

# List of Figures

# 1 Summary

## 1.1 Introduction

The Wheeler River uranium project is an advanced exploration stage joint venture owned 60% by Denison Mines Inc. a wholly owned subsidiary of Denison Mines Corp. (Denison), 30% by Cameco Corporation (Cameco), and 10% by JCU (Canada) Exploration Company Ltd. 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. Including its interest in the Wheeler River project, which hosts the high grade Phoenix and Gryphon uranium deposits, Denison's exploration portfolio consists of numerous projects covering over 354,000 hectares in the eastern 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 deposit and a 61.55% interest in the J Zone deposit on the Waterbury Lake property. Both the Midwest and J Zone deposits are located within 20 kilometres of the McClean Lake mill.

The Wheeler River property has been explored since the late 1970s but in late 2004 Denison entered an agreement with the joint venture partners to earn into a majority 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 graphitic pelite basement rocks. 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). As drilling defined further mineralization, subsequent resource estimates were made on the Phoenix deposit in December 2012 and June 2014 by Roscoe Postle Associates (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 3 km northwest of the Phoenix deposit. A maiden resource estimate was completed for the Gryphon deposit in November 2015 by RPA and an updated NI 43-101 Technical Report was issued for the Wheeler River project.

In September 2015, Denison commissioned SRK and other consultants to prepare a National Instrument 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.

## 1.2 Geology

The uranium deposits in the Athabasca Basin occur below, across, and immediately above the sub-Athabasca unconformity, which can lie within a few metres of surface at the rim of the basin, to over 1,000 m deep near its centre. A water bearing sandstone unit lies above the unconformity. The deposits are formed by extensive hydrothermal systems that can result in large accumulations of uranium mineralization when reduced and oxidized fluids interact at the unconformity – particularly where the unconformity is intersected by post-Athabasca brittle fault zones.

The Phoenix deposit is located at the unconformity between the Athabasca Basin and basement rocks, approximately 400 m below surface. Mineralization and alteration have been traced over a strike length of approximately 1 km and are coincident with a significant steeply dipping fault zone. A total of 253 boreholes have reached the target depth, delineating two distinct zones (A and B) of high grade uranium mineralization lying horizontally at the unconformity.

Unlike the Phoenix deposit, mineralization at Gryphon occurs from 580 m to 850 m below surface and is centred approximately 220 m below the unconformity. The deposit consists of a set of parallel, stacked, elongate lenses that are broadly conformable with the basement geology and associated with a significant fault zone (G-Fault) that separates a thin unit of quartzite (Quartz-Pegmatite) from an overlying graphitic pelite (Upper Graphite). The lenses dip moderately to the southeast and plunge moderately to the northeast. The Gryphon deposit is approximately 450 metres along plunge, 80 metres across plunge and varies in thickness, between 2 and 20 metres, depending on the number of mineralized lenses present.

## 1.3 Mineral Resources

National Instrument 43-101 compliant mineral resource estimates for the Wheeler River project have been prepared by RPA as disclosed in the November 2015 report titled "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada." (RPA, 2015)

The effective date of the resource estimate is September 25, 2015. The Phoenix cut-off grade of 0.8% $U_3O_8$ is based on internal conceptual studies by Denison and a price of US$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 on mines operating in the Athabasca Basin and a price of US$50/lb $U_3O_8$.

For the Phoenix and Gryphon deposits, total Indicated mineral resources are estimated at 166,400 t at an average grade of 19.14% $U_3O_8$ containing 70.2 million pounds of $U_3O_8$. Total Inferred mineral resources are estimated at 842,600t at an average grade of 2.37% $U_3O_8$ containing 44.1 million pounds of $U_3O_8$.

## 1.4 Mineral Resources within PEA Design Plan

This PEA is based on the Indicated and Inferred mineral resources of the Phoenix and Gryphon deposits. SRK's methodology for estimating the mineralization to be included in the mine production plan included:

- Selecting mining methods
- Cut-off grade of 0.4% $U_3O_8$ was estimated for longhole mining at Gryphon, refer to report Section 16.3.2. A cut-off grade of 2% $U_3O_8$ was used as a guide for jet bore mining at Phoenix, refer to report Section 16

- Mineralization wireframes were evaluated at a zero cut-off grade
- Wireframes were clipped to remove low grade areas below the cut-off grade
- The final wireframes were evaluated in Gemcom to determine in situ tonnes and grades
- Factors for external dilution and mining recovery were applied

Table 1-1 shows the Wheeler River mineral resources within PEA design plan (MR within PEA).

**Table 1-1: Wheeler River Mineral Resources within PEA Design Plan**

| Deposit | Kilo-Tonnes | Grade % $U_3O_8$ | Mlbs $U_3O_8$ | Source |
|---|---|---|---|---|
| Phoenix | 232.0 | 12.30 | 63.0 | Indicated mineral resources |
| Phoenix | 7.8 | 6.27 | 1.1 | Inferred mineral resources |
| Gryphon | 975.0 | 1.90 | 41.0 | Inferred mineral resources |

SRK notes that this PEA is preliminary in nature. MR within PEA are sourced partially from Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

## 1.5 Hydrogeology and Mine Geotechnical

The following hydrogeological and geotechnical characteristics of the project were considered in the mining study:

- The Phoenix deposit is located at the unconformity and is subject to high pressure water in the overlying sandstone.
- The Gryphon deposit is located in basement rocks and is considered protected from the water bearing sandstone.
- Non-routine water inflows could be as high as 1,500 $m^3$/h.
- At Phoenix, the geotechnical assessment indicates very poor rock mass conditions in the immediate hangingwall. Ground conditions within the deposit are generally poor to fair. Basement rock development will generally be in fair to good rock mass conditions.
- Generally, fair to good ground conditions are expected for the Gryphon deposit, with localized zones of lower quality rock mass attributed to fault structures.
- Phoenix requires ground freezing to mitigate high water pressures and to help strengthen the poor hangingwall rock mass conditions.
- Geotechnical conditions at Gryphon indicate conventional mining methods are applicable.

## 1.6 Mining

### 1.6.1 Mining Methods

Jet bore system (JBS) mining was selected for the high grade Phoenix Zones A and B1, similar to the mining method utilized at the Cigar Lake mine. This mining method requires freeze wall protection in a tent configuration (Figure I).

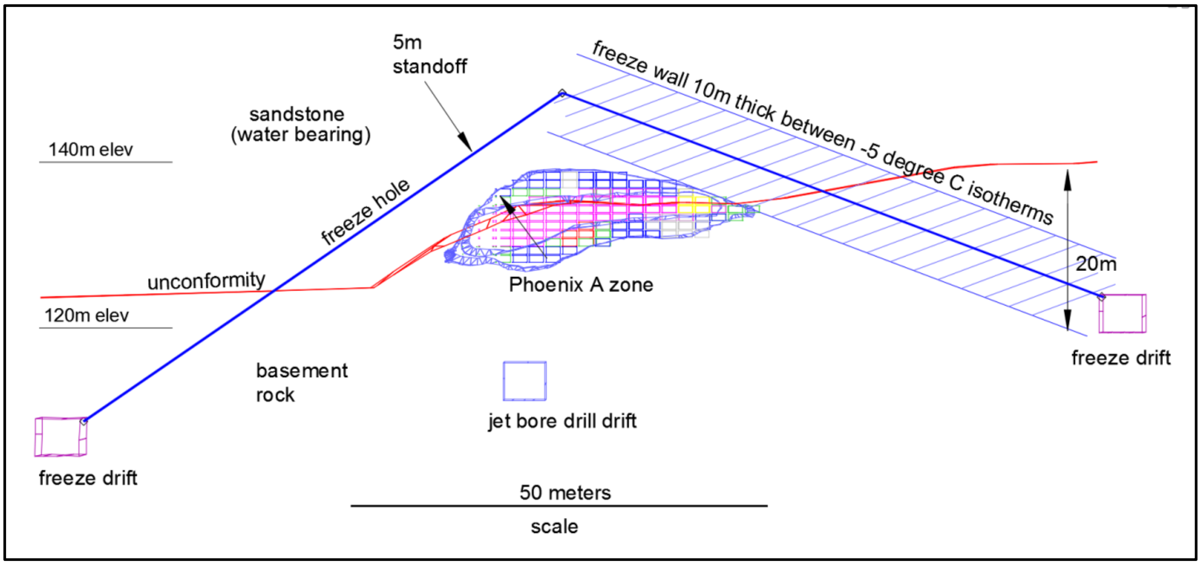



**Figure 1-1: Phoenix A Zone Cross-Section Showing Tent Freeze Wall Arrangement (Looking NE)**

The JBS mining method requires an access drill drift within basement (waste) rock below the mineralization (Figure 1-1). A pilot hole is drilled up into the deposit equipped with a rotating high pressure water jet capable of cutting the surrounding mineralization. A slurry of water and loose broken rock flows by gravity out of the cavity created, down into a receiving car next to the jet bore machine. At the Cigar Lake mine, the JBS method has successfully excavated cavities in the range of 4 to 7 m in diameter. Mined out cavities will be filled with concrete that withstands the force of the water jet when an adjacent cavity is mined. The JBS method allows for mine operators to carry out their work in a protective environment to ensure exposure to high grade mineralization is minimized for all personnel.

Conventional longhole open stoping with backfill is planned for the Gryphon deposit. No freeze wall protection is needed due to the location of the deposit well below the unconformity in basement rock. (Figure 1-2)

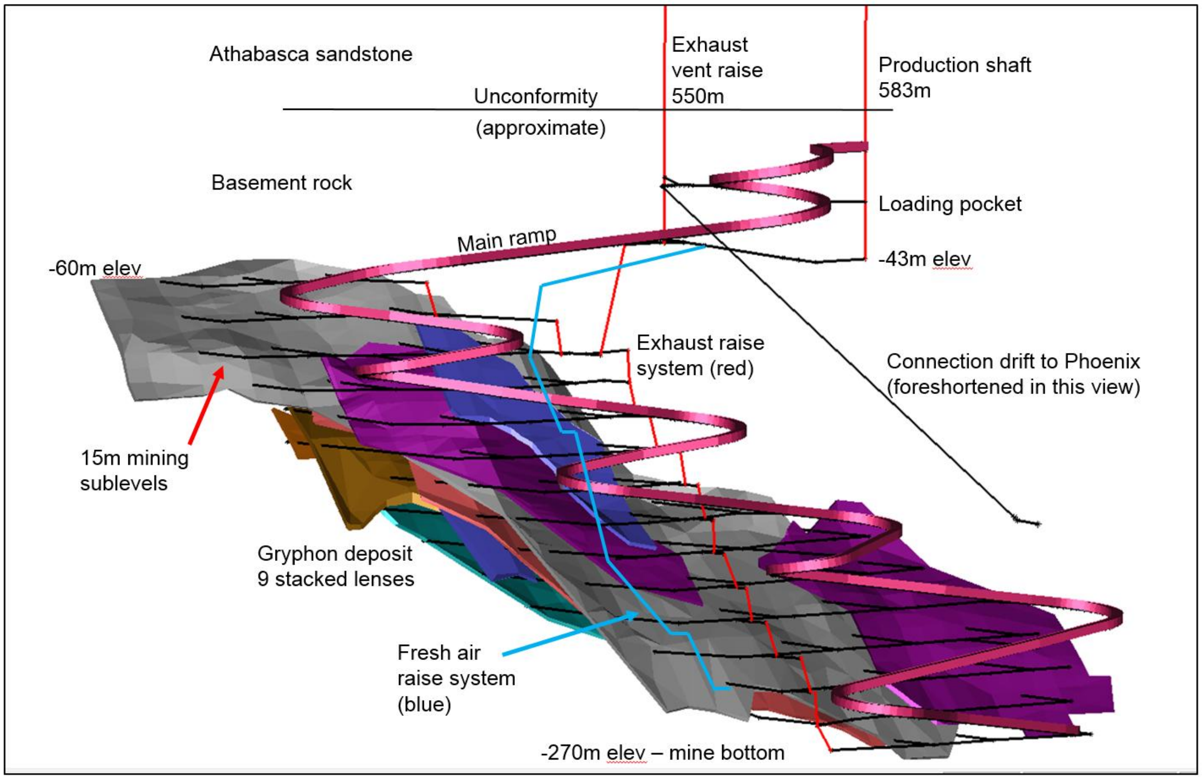



**Figure 1-2: Isometric View - Gryphon 3D Mine Model (Looking N)**

Table 1-1 shows the relative distribution of the planned mining methods.

**Table 1-2: Relative Distribution of Mining Methods**

| Mining Method | Deposit | Mining Method Distribution by Tonnes | by Pounds $U_3O_8$ |
|---|---|---|---|
| Jet Bore System | Phoenix | 20% | 61% |
| Longhole Stoping | Gryphon | 80% | 39% |

## 1.6.2 Other Mining Methods Considered

The geometry at the Phoenix Zones A and B1 is also well suited for a blind raise boring mining method. This method was successfully tested at the McArthur River mine, but it was not incorporated into its life-of-mine (LOM) plan.

This method was not selected for the Phoenix deposit for the following reasons:

- On an overall basis it was considered less productive than the JBS method
- Increased lateral development requirements

Potential productivity improvements to the blind boring method may be possible by blasting into the cavity using longhole drilling. This was not considered as part of this study.

### 1.6.3 Conceptual 3D Mine Model

Several different configurations are possible when considering how to provide underground mining access to the Gryphon and Phoenix deposits. An important aspect of the design approach was to maximize synergy between the two deposits. The distance between the two, at roughly 3 km, is such that the question must be answered as to whether it is best to connect them underground, or to develop them with separate accesses from surface.

Aspects considered by SRK in the mine access design process included:

- Minimizing capital costs
- Maximizing synergy between the two deposits, including ability to move workers, materials and equipment from surface and between deposits
- Providing sufficient air flows without exceeding rule-of-thumb air velocities
- Moving the mobile mining fleets underground
- Providing services to each deposit including, mine dewatering, electrical power, second exit
- Providing additional services for Phoenix including, brine piping for freeze walls, high grade uranium slurry transport and slick line for concrete for JBS backfilling

The design approach selected connects the two deposits underground with a 2.8 km (line distance) connection drift driven from Gryphon to Phoenix where it is positioned safely in the basement rock below the deposit (Figure 1-3). For Gryphon, the mine design includes a full service production shaft and a bare ventilation exhaust raise to support underground development and production. Heated fresh air will be delivered through the shaft with return air up the ventilation raise. Later in the mine life with Gryphon mining completed, Phoenix will receive fresh air from Gryphon through the connection drift and Phoenix exhaust air will be routed to surface through an additional ventilation raise at Phoenix.

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 appear to offer competitive costs and construction schedules. The method includes a comprehensive surface pre-grouting program followed by blind boring with the shaft full of water. After dewatering, a concrete liner will be installed over the full length and grouted into basement rock. The main advantage is virtually eliminating the risk of unexpected shaft water inflow during shaft construction.

Table 1-3 shows the estimated LOM lateral development requirements.

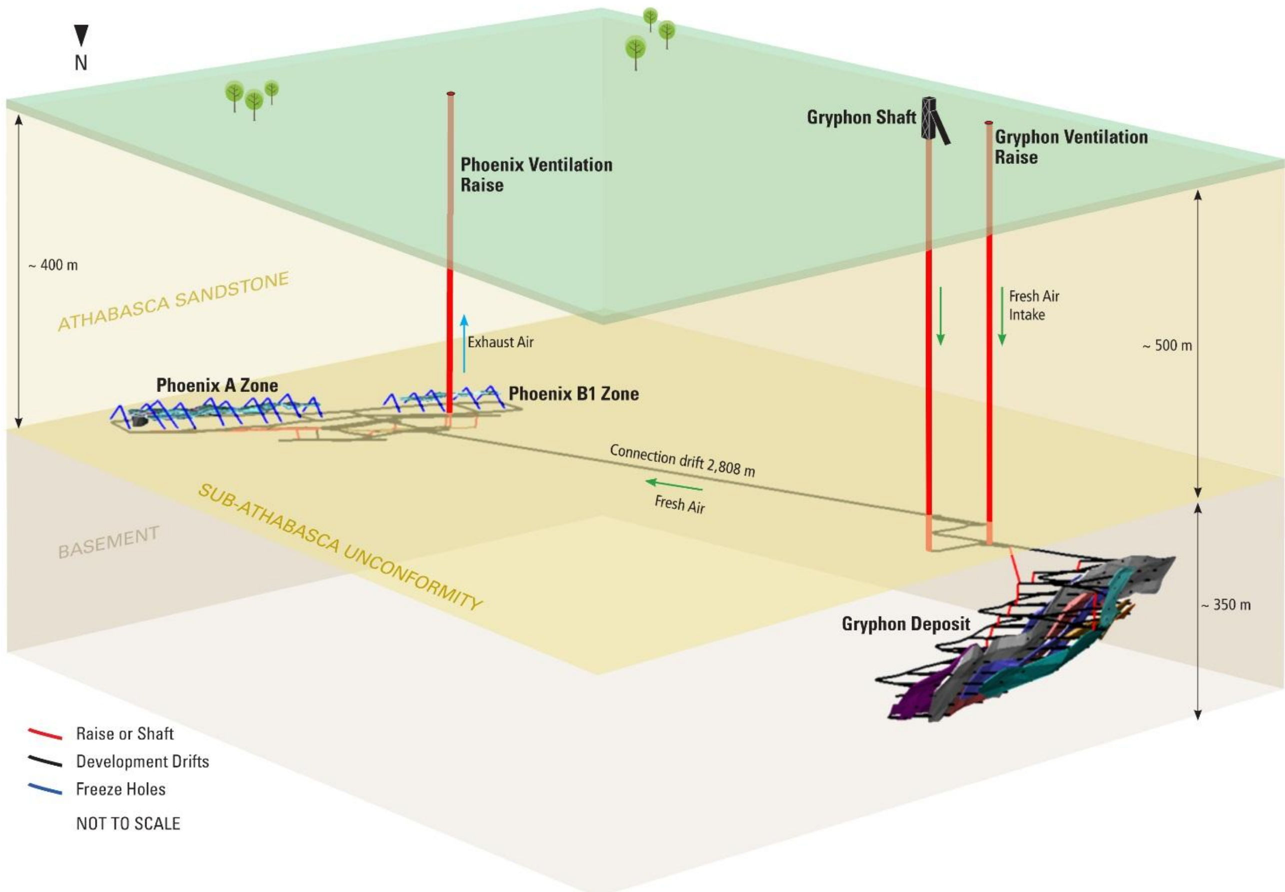



**Figure 1-3: Isometric View - Connection Drift - Phoenix Phase Air Flows (Looking S)**

**Table 1-3: LOM Lateral Development Estimate**

| Lateral Development | Gryphon (m) | Phoenix (m) | Total (m) |
|---|---|---|---|
| Connection Drift | 3,239 | | 3,239 |
| Other Capitalized Development | 6,588 | 6,310 | 12,898 |
| **Total Capitalized** | 9,827 | 6,310 | 16,137 |
| **Expensed Development** | 4,160 | 4,651 | 8,811 |
| **Total Lateral Development** | **13,987** | **10,961** | **24,948** |

### 1.6.4 Production Schedule

The nominal production rates selected for this study are:

- Gryphon – 7 year mine life, at 6.0 Mlbs $U_3O_8$ per year (399 t/d)
- Phoenix – 9 year mine life, at 7.0 Mlbs $U_3O_8$ per year (73 t/d)

SRK defined a five-year pre-production period from January 2021, when the project is assumed to be permitted, until it reaches commercial production in December 2025. The project production period is 16 years from January 2026 to the end of 2041.

### 1.6.5 Underground Infrastructure and Services

Planned underground infrastructure and services include:

- Definition drilling - The Phoenix deposit is well drilled. For the Gryphon deposit, SRK planned an additional 7,800 m of NQ underground drilling to tighten the average pierce point spacing in the mineralized lenses to 25 x 25 m.
- Waste rock handling - Waste rock not needed for backfill will be trucked to a truck dump near the Gryphon shaft and hoisted to surface in one of the skips. The surface site layout includes an area designated for waste rock storage.
- Low grade conventional mineralization handling at Gryphon - This material will be hoisted using the other skip and the other side of the loading pocket. Low grade material will be fed to the loading pocket from a separate mineralization handling system.
- High grade material handling at Phoenix - Broken mineralization and water from the jet boring unit will be crushed underground and fed into a small ball mill. The high grade slurry produced will be pumped to surface through a steel pipeline installed in the Phoenix ventilation raise.
- Freeze wall infrastructure - Phoenix underground freeze infrastructure will include a heat exchanger for the chilled brine and an underground brine circulation system from the heat exchanger to the freeze holes. Freeze holes will be drilled to lengths of approximately 75 m at a 4 m spacing from two dedicated freeze drifts.
- Mine ventilation - Ventilation estimates were based on comparisons to other Athabasca Basin uranium mines and were selected to ensure the planned mine development would be adequately ventilated. SRK estimated the required mine ventilation at 302 cms for Gryphon, and 240 cms for Phoenix.
- Mine dewatering - The system is designed for a capacity of 2,250 $m^3$/h. The main sumps and pumps will be located at the Gryphon mine. Phoenix mine water will be transferred to the Gryphon main sumps, largely by gravity, through pipe lines installed in the connection drift.
- Electrical power distribution - Power is expected to be sourced from the Provincial power grid and will feed a main 13.8 kV substation located on surface near the Gryphon shaft, which will then be fed underground through the Gryphon shaft and Phoenix ventilation raise.
- Equipment maintenance - Fully serviced multi-bay underground maintenance shops will be constructed near the Gryphon shaft and at the Phoenix mine for servicing equipment.
- Refuge stations - Five permanent refuge stations are planned as well as three portable units that can be moved with development crews.

## 1.7 Mineral Processing

This PEA is based on the assumption that mill feed from Wheeler River will be trucked to an existing uranium mill in northern Saskatchewan for processing under a custom milling agreement. Preliminary process test work was completed for the Phoenix deposit in 2014, and for the Gryphon deposit in 2015. The results were used to support process design criteria suitable for the Wheeler River feeds at a regional acid leach mill.

At this time, custom milling at the JEB uranium mill on the McClean Lake site is considered the most likely scenario due to capacity constraints (in production and tailings management) at other regional milling facilities. Pursuing this option requires the construction of a new 45 km section of haul road between the McArthur River mine site and the Cigar Lake mine site to connect existing roads that otherwise run from the McClean Lake mill to the Key Lake mill. The cost estimate for this haul road is included in the project capital.

The production plan for the Gryphon and Phoenix deposits aligns well with making use of available capacity at the McClean Lake mill while co-milling with anticipated feeds from Cigar Lake mine. The expected peak mill production rate of up to 24 M pounds per year (lb/yr) $U_3O_8$ could occur while co-milling Cigar Lake Phase 1 high grade and Gryphon deposit low grade feeds, matching the intended total license capacity of the mill.

The current scope of mill modifications approved for construction at McClean Lake is focused on enabling the full capacity of 18 M lb $U_3O_8$/yr milling of high grade Cigar Lake Phase 1 feed through the #2 leach circuit, while a notional 4 M lb $U_3O_8$/yr of co-milling capacity exists in the #1 leach circuit for a total leach capacity of 22 M lb $U_3O_8$/yr. In the expected mill operating scenario there is no constraint to production of 18 M lb $U_3O_8$/yr of Cigar Lake feed through the #2 leach circuit, whereas production capacity constraints are identified for the Gryphon deposit feed due to tonnage restrictions in the #1 leach circuit.

In order to co-mill the full tonnage of the Gryphon deposit feed with the Cigar Lake Phase 1 feed, expansion of the #1 leaching circuit and solid/liquid separation circuits' capacities are required. The McClean Lake #1 leach circuit currently has insufficient retention capacity to provide the estimated leach time. One or two additional tanks would be required to augment the existing capacity to efficiently process the Gryphon deposit feed.

The counter current decantation (CCD) circuit used for solid-liquid separation at McClean Lake is anticipated to be a bottleneck in mill production. A conventional approach to wash poorly settling solids is pressure filtration. For the base case to reach full Cigar Lake Phase 1/Gryphon co-milling capacity within the design recovery rate, two new pressure filters are proposed to supplement the existing CCD thickener circuit. The proposed solid-liquid separation operation is as follows:

- Cigar Lake leach residue slurry from the primary thickener underflow feeds to a new dedicated high grade pressure filter. The washed cake is sent directly to tailings neutralization.
- Gryphon leach residue slurry is split into coarse and fine fractions using a hydrocyclone, and then:
    - The coarse fraction is sent to the existing CCD thickener circuit. This way, CCD tonnage is reduced to an acceptable rate and settling performance is improved at the same time.
    - The fines fraction is sent to a new low grade pressure filter. The washed cake is sent directly to tailings neutralization.

To co-mill the full tonnage of the Phoenix zone feed with the Cigar Lake Phase 2 feed, some minor re-configurations of the slurry receiving, leaching, and solid/liquid separation circuits are required. After the pregnant solution is separated from the leached solids residue, the downstream circuits (clarification, SX, carbon columns, precipitation, calcining, packaging, crystallization) are assumed from stated expansion plans to be capable of processing 24 M lb $U_3O_8$/yr.

The metallurgical test results indicate the Gryphon and Phoenix deposits are suitable for processing through the McClean Lake mill. Overall uranium process recovery has been estimated at 97.0% for Gryphon (due to lower grade), while Phoenix recovery is estimated at 98.1%.

## 1.8 Surface Infrastructure

Planned surface infrastructure at the Gryphon site includes:

- Production shaft, hoist house and headframe, and ventilation raise
- Main fresh air fans and mine air heater
- Fully serviced camp
- Mine buildings including administration office, change house, maintenance shop, warehouse, emergency services building, and laboratories

- Electrical sub-station supplied by a new overhead power supply line
- Back-up diesel power generators
- Water supply
- Water management ponds and water treatment plant
- Waste rock storage facilities for special waste, potentially acid generating (PAG) waste, and clean waste
- Fuel storage facility
- Backfill preparation plant

Planned surface infrastructure at the Phoenix site includes:

- Ventilation raise collar with main exhaust air fans
- Freeze plant infrastructure
- High grade slurry load out facility

## 1.9 Environmental and Permitting

- There are no recognized environmental fatal flaws associated with this project. All potential environmental impacts can be successfully mitigated through the implementation of industry best practices. The most significant environmental concern associated with the project will be the management of routine and non-routine mine water effluent.
- The project will require completion of a federal and provincial environmental assessment. This assessment will be completed as a joint environmental assessment. It is estimated the assessment will require approximately 24 to 36 months to complete following the submission of a detailed project description.

## 1.10 Capital and Operating Costs

Capital costs are expressed in 2015 Canadian dollars to a bottom line accuracy of +/- 40%. Initial capital costs are based on the five-year period from January 1, 2021 through to December 31, 2025. Sustaining capital costs are for the period from January 1, 2026 through to the end of 2041.

The Wheeler River project total capital cost estimate is $1,103 million including a contingency of 26% as shown in Table 1-4, comprising $560 million initial capital and $543 million sustaining capital.

**Table 1-4: Wheeler River Project Capital Cost Estimate**

| Capital Costs Area | Initial $M | Sustaining $M | Total $M | Direct $M | Indirect $M |
|---|---|---|---|---|---|
| Owners Costs | $25 | $0 | $25 | $0 | $25 |
| Surface Infrastructure | $167 | $7 | $174 | $138 | $36 |
| Mine | $219 | $335 | $554 | $469 | $85 |
| Plant Feed Handling & Processing | $18 | $60 | $78 | $51 | $27 |
| Decommissioning | $0 | $40 | $40 | $32 | $8 |
| Subtotal | **$429** | **$442** | **$871** | **$690** | **$181** |
| Contingency | $131 | $101 | $232 | $178 | $54 |
| **Total Capital ($M)** | **$560** | **$543** | **$1,103** | **$868** | **$235** |

Operating costs have been estimated at $19.28 per pound $U_3O_8$ for the Gryphon deposit and $29.90 per pound $U_3O_8$ for the Phoenix deposit. Table 1-5 shows the composition of the projected operating cost estimates.

**Table 1-5: Wheeler River Project Operating Cost Estimate**

| | \$/lb $U_3O_8$ | |
|---|---|---|
| **Operating Costs Area** | **Gryphon** | **Phoenix** |
| Mining | \$3.45 | \$17.45 |
| Surface Transportation | \$1.63 | \$0.85 |
| Processing | \$8.03 | \$6.03 |
| Toll Milling Fee | \$2.00 | \$2.00 |
| General & Administration | \$4.17 | \$3.57 |
| **Total** | **\$19.28** | **\$29.90** |

# 1.11 Indicative Economic Results

The PEA considers two pricing scenarios because of the long lead time to production (2026). Uranium price estimates were sourced from the Ux Consulting Company, LLC. (UxC) (Refer to Section 22.5.2 for details).

(1) A Base case scenario using a long-term contract price of US\$44.00/lb as of March 28, 2016.
(2) A Production case price sensitivity using a long-term contract price of US\$62.60/lb for the year 2026 (based on UxC's Uranium Market Outlook Q1 2016) when the project production period begins.

An exchange rate of 1.35 CAD/USD was selected based on Bloomberg long term projections as of February 2016.

## 1.11.1 Pre-tax Indicative Economic Results

### Base Case

The Wheeler River project (100% basis) indicative pre-tax base case economic results include:

- An internal rate of return (IRR) of 20.4%
- A net present value (NPV) at 8% discounting of \$513 million
- A pay-back period of approximately three years (from the start of production)
- The break-even price for the project is estimated at approximately US\$34/lb $U_3O_8$

### Production Case

Using a uranium price of US\$62.60/lb, with all other variables held constant, the project's NPV at 8% discounting increases to \$1,420 million, the IRR increases to 34.1%, and the pay-back period decreases to approximately 18 months (from the start of production)

## 1.11.2 Post-tax Indicative Economic Results

### Base Case

Denison's 60% ownership interest in the Wheeler River project yields the following indicative post-tax base case economic results:

- An internal rate of return (IRR) of 17.8%
- A net present value (NPV) at 8% discounting of \$206 million

### Production Case

Using a uranium price of US\$62.60/lb, with all other variables held constant, the project's post-tax NPV to Denison, at 8% discounting, increases to \$548 million and the IRR increases to 29.2%.

SRK notes that this PEA is preliminary in nature, it includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

## 1.12 Risks and Opportunities

The Wheeler River project risks include:

- The inclusion of Inferred mineral resources in the plant feed estimate.
- The possibility of an unexpected ground water inflow causing loss of production and increased costs.
- The JBS method has been developed specifically for the Cigar Lake deposit and there is a risk the method will not perform as well at Phoenix due to different orebody characteristics.
- The possibility that it may take longer than planned to obtain full project regulatory approval, delaying the start of construction on the site.
- This study is based on custom milling the Wheeler River plant feed at the McClean Lake mill, an existing uranium processing plant in northern Saskatchewan. There is a risk that sufficient plant capacity or tailings capacity may not be available for the Wheeler River feed, delaying the project or requiring additional capital to fund further modifications to the existing plant or the construction of a new processing plant.
- The composite samples used for the metallurgical testing of the Gryphon and Phoenix deposits do not reflect the potential variability of the processing plant feed, and uranium milling recoveries of 97.0% for Gryphon and 98.1% for Phoenix may not be consistently achieved.
- Capital and operating cost estimates developed as part of this study are at a scoping level, and there is a risk that actual costs will be higher than those estimated.

The Wheeler River project opportunities include:

- Wheeler River is Denison's flagship exploration property. There are many high priority exploration target areas, the most important of which consist of unconformity and basement targets in the Gryphon area. Future exploration may discover additional mineralization that could become part of the Wheeler River mining plan. During the winter 2016 program, drill testing within 200 metres north and northwest of the Gryphon deposit returned numerous high-grade intersections which have been reported in the Company's press releases. These results are not included in the current resource estimate or PEA.
- Annual production is constrained by available mineral process capacities. Opportunities to increase capacity may allow for increased mine production from Wheeler River.
- It is likely that continuous improvements made by currently operating uranium mines will benefit the Wheeler River project. One area of possible benefit could be in the approved handling methods for high grade uranium.

## 1.13 Conclusions and Recommendations

The results of the PEA indicate that the Wheeler River project has a positive economic return at the base case assumptions considered. The results are considered sufficiently reliable to guide Denison's management in a decision to further develop the project. This would typically involve the preparation of a preliminary feasibility study (PFS).

Assessment of each area of investigation completed as part of this PEA suggests recommendations for further investigations to improve the preliminary designs and to mitigate risks. The key recommendations arising from this study are described below.

- Denison has already planned exploration drilling at Wheeler River for 2016 to focus on numerous targets in the vicinity of the Gryphon deposit. In addition, the PEA results warrant an infill drilling program should be undertaken on the Gryphon deposit to bring the Inferred mineral resource into Indicated status.
- Targeted geotechnical drilling is required with associated laboratory strength testing. Structural models at Phoenix and Gryphon should be updated considering the additional data. A geotechnical database quality control review should be completed to screen and to compile a robust geotechnical data set for use in mine design.
- Further hydrogeological investigation should focus on hydraulic testing of permeable structures. Shallow hydrogeological testing should focus on areas of proposed shafts and raises, and should include testing of pumping wells and observation wells. Deep testing should include vibrating wire piezometer installation and other deep down-hole hydrogeology tests. A 3D numerical groundwater model should be constructed to estimate groundwater inflows for various stages of mine life.
- A preliminary feasibility mining study should be undertaken once infill drilling has been completed at Gryphon. Alternative methods should be investigated for shaft sinking and development of the required ventilation raises. Locations for the shaft and raises should be selected based on field investigation and consideration of the geotechnical/structural model. Further investigation is recommended into the technical aspects of applying the JBS at Phoenix.
- The design of surface water storage ponds and water treatment plant should be refined as estimates of mine water chemistry and flow become available. The existing surface hydrology data and suitability of the monitoring network should be reviewed. Long term meteorological data should be obtained for storm water management design. A water balance for the two mining sites should be determined.
- Pre-feasibility level process engineering design and cost estimation should be undertaken for the Wheeler River site’s underground and surface plant feed handling facilities and for JEB mill modifications, based upon updated design criteria derived from the recommended test programs:
    - Perform optimization test work on Gryphon and Phoenix deposits for grinding, leaching and CCD circuits’ performance.
    - Re-confirm production of on-spec yellowcake. Test effluent and tailings treatment.
    - Perform test work to investigate potential for hydrogen evolution from the Gryphon and Phoenix deposits.
- Should commercial negotiations proceed with the McClean Lake joint venture in respect of toll milling, design capacities should be validated for each of the downstream mill circuits (clarification, SX, carbon columns, precipitation, calcining, packaging, crystallization) and required equipment upgrades should be identified.
- Tailings characterization is recommended in conjunction with further metallurgical testing. The process solutions, final effluent, and the final tailings slurry (solids and liquids) should be analyzed for a complete suite of major and trace elements, and mineralogical characterization should also be completed on the tailings solids. Tailings slurry should be subjected to an anoxic aging test to simulate changes that are likely to occur over the short to medium-term.
- Waste rock characterization is recommended using a staged approach, with static testing (acid base accounting tests) on a moderate number of samples from each deposit area, then kinetic testing – including both laboratory and field based tests on a representative subset of samples. This will determine requirements for segregation, storage and handling of the waste rock.
- A detailed stakeholder engagement plan should be initiated to support the advancement of the project’s engineering and regulatory requirements. Comprehensive environmental and social baseline studies should be initiated to characterize the aquatic and terrestrial environment, heritage and archeological aspects of the project.

# 2 Introduction

The Wheeler River uranium project is an advanced exploration stage joint venture owned 60% by Denison, 30% by Cameco Corporation (Cameco), and 10% by JCU (Canada) Exploration Company Ltd. 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. Including its interest in the Wheeler River project, which hosts the high grade Phoenix and Gryphon uranium deposits, Denison's exploration portfolio consists of numerous projects covering over 354,000 hectares in the eastern 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 deposit and a 61.55% interest in the J Zone deposit on the Waterbury Lake property. Both the Midwest and J Zone deposits are located within 20 kilometres of the McClean Lake mill.

The Wheeler River property has been explored since the late 1970s but in late 2004 Denison entered an agreement with the joint venture partners to earn into a majority 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 graphitic pelite 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-kilometre-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. The conceptual study was primarily based on comparable operations with minimal site specific assumptions made. The study did not complete any mining method analysis. As drilling defined further mineralization, subsequent resource estimates were made on the Phoenix deposit in December 2012 and June 2014 by Roscoe Postle Associates (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 3 km northwest of the Phoenix deposit. A maiden resource estimate was completed for the Gryphon deposit in November 2015 by RPA and an updated NI 43-101 Technical Report was issued for the Wheeler River project.

In September 2015, Denison commissioned SRK and other consultants to prepare a National Instrument 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.

## 2.1 Basis of Technical Report

This PEA technical report is based on the following sources of information:

- November 17, 2010, SRK Consulting technical report, "Technical Report on the Phoenix Deposit (Zones A&B) - Wheeler River Project, Eastern Athabasca Basin, Northern Saskatchewan, Canada" (SRK, 2010)
- Publicly available technical reports prepared by Cameco including November 2, 2012 "McArthur River Operation, Northern Saskatchewan, Canada" (Cameco, 2012a) and February 24, 2012 "Cigar Lake Project, Northern Saskatchewan, Canada" (Cameco, 2012)
- Technical report prepared by Roscoe Postle Associates (RPA), November 25, 2015, "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada" (RPA, 2015)
- Technical report prepared by Amec Foster Wheeler, "Wheeler River Preliminary Economic Assessment – Process Aspects" January 21, 2016 (Amec, 2016)
- Technical and cost information provided by Denison
- Discussions with Denison technical and management personnel
- Inspection of the project area and drill core during a site visit
- Technical and cost information provided by Amec Foster Wheeler in the areas of metallurgy and mineral processing
- Technical and cost information provided by Mr. Greg Newman, President of Newmans Geotechnique Inc. (ground freezing experts)
- Technical information provided by the Saskatchewan Research Council (SRC)
- Additional information from public domain sources

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

- Roscoe Postle Associates Inc. (RPA) commissioned by Denison, responsible for report Sections 4 to 12, and 14, the summary of these sections in the Summary, and the Interpretation and Conclusions and Recommendations related to these sections
- Amec Foster Wheeler (Amec Foster Wheeler) commissioned by Denison, responsible for report Sections 13 and 17, parts of Section 21, the summary of these sections in the Summary, and the Interpretation and Conclusions and Recommendations from Sections 13 and 17
- Newmans Geotechnique Inc. (NGI) commissioned by SRK, responsible for report sections related to ground freezing including Sections 16.5.2, 16.8.3 and 18.4

This PEA technical report is based on Mineral Resource Statements for the Gryphon and Phoenix deposits prepared by RPA. It was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1 and is suitable for public disclosure.

The term "mineral resources within PEA design plan" (MR within PEA) is used in this PEA technical report to represent portions of the Gryphon and Phoenix uranium mineral resources that have had mining parameters applied to them including cut off criteria, external dilution and mining losses. MR within PEA are included in the Economic Analysis as uranium mill feed.

MR within PEA are sourced partially from Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

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

## 2.2 Qualified Persons

The compilation of this PEA technical report was undertaken by Mr. Ken Reipas, PEng, SRK. By virtue of his education, membership to a recognized professional association, and relevant work experience, Mr. Reipas is an independent Qualified Persons as this term is defined by National Instrument 43-101.

The following Qualified Persons have provided contributions for sections of this 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 Qualified Persons (QP) as this term is defined by National Instrument 43-101.

- Mr. Ken Reipas, PEng, SRK, mine design, mining costs, infrastructure, economics
- Mr. William E. Roscoe, PhD, PEng, RPA, geology and mineral resource estimation
- Mr. Bruce Murphy, FSAIMM, SRK, mine geotechnical
- Mr. Todd Hamilton, PGeo, SRK, hydrogeology
- Mr. Tom Sharp, PEng, SRK, water management and treatment
- Mr. Greg Newman, PEng, Newmans Geotechnique Inc, ground freezing
- Ms. Kelly Sexsmith, PGeo, SRK, waste rock geochemistry/management
- Mr. Mark Liskowich, PGeo, SRK, environmental, permitting, and social impact
- Mr. Lorne Schwartz, PEng, Amec Foster Wheeler, metallurgical and mineral processing
- Mr. Chuck Edwards, PEng, Amec Foster Wheeler, metallurgical and mineral processing

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

**Mr. Reipas** (SRK) is a Principal Consultant who has been employed by SRK since 2001 and has over 30 years of experience in mine engineering, mine production, and consulting. Prior to joining SRK, he worked at several open pit and underground mining operations in Canada involved in the bulk mining of iron, coal, gold, and base metals. Positions held included Chief Engineer and Mine Superintendent. Since 1997, his consulting projects have included technical studies, mine planning and reserves, mine operations assistance, and due diligence reviews.

**Dr. William E. Roscoe** (RPA) Principal Geologist, is Chairman Emeritus of RPA. He was a founding partner of RPA in 1985. Dr. Roscoe has extensive experience in mineral exploration, mineral resource and mineral reserve estimation, reviews using both conventional and geostatistical methods, valuation of mineral properties, supervision of exploration projects, and monitoring of exploration joint ventures. He has particular expertise in the preparation of NI 43-101 technical reports and assessment of advanced projects.

**Mr. Murphy** (SRK) is a Principal Consultant specializing in rock mechanics, which includes rock mass characterization leading to excavation and support design, mining method selection and the technical auditing of both open pit and underground operations. Prior to joining SRK in 2002, Mr. Murphy had been involved in mining operational rock mechanics since 1989; including hard rock open pit and underground operations, which include gold, copper and iron ore. He has operational/project experience in a number of underground mining operations/project, in Africa, Asia, North and South America.

**Mr. Hamilton** (SRK) is a Principal Consultant with 30 years of global experience including site characterization of groundwater flow systems; the design, implementation and optimization of dewatering systems; pore pressure input to slope stability studies; exploration and development of groundwater resources; delineation and remediation of groundwater contamination at landfills, industrial sites and mines; and design and impact prediction for wastewater disposal systems. He has extensive field experience in well installation, hydraulic testing, surface and underground hydrogeological mapping, and groundwater sampling and mapping.

**Mr. Sharp** (SRK) is a Principal Consultant and a civil/ environmental engineer with over 20 years of experience in mine water management, treatment, and chemistry, working as an operator and technical consultant. He has prepared, designed, and evaluated water treatment alternatives, site water and load balances, and water management plans for mine planning, operation, and closure. He also has developed and evaluated management and disposal alternatives for metallurgical residual products and by-products, water treatment sludge, and dredge spoils.

**Mr. Newman** (Newmans Geotechnique Inc.) has been involved with the design and implementation of artificial ground freezing systems since the early 90's.  He led the team that designed, developed, and implemented a system to drill and ground freeze an ore-body at Cameco's flagship McArthur River Mine.  In recognition for his work, Greg received Cameco's first outstanding achievement award for his design of a blow out prevention system for drilling freeze holes in unconsolidated ground, fractured rock, and squeezing clays.

**Ms. Sexsmith** (SRK) is a Principal Consultant in the GeoEnvironmental group, providing expertise in the characterization and prediction of acid rock drainage and metal leaching for new, developed, and closed mining properties. Experience includes design and supervision of geochemical test programs, development of conceptual waste management plans, prediction of water quality from mine components, and mine waste management. She has extensive experience with northern Saskatchewan uranium projects.

**Mr. Liskowich (**SRK**)** is an SRK Principal Consultant based in Saskatoon with 25 years of experience specializing in the environmental management of the mining industry. His career has included positions with exploration companies as an exploration geologist as well as positions with both the Saskatchewan and federal governments as an environmental mines inspector/consultation specialist. His experience has given him expertise in environmental management, environmental assessment, licensing, permitting, auditing, due diligence, and public and regulatory consultation. Throughout his career, he has had firsthand experience with each of Saskatchewan’s operating and decommissioned uranium mines.

**Mr. Lorne Schwartz** (Amec Foster Wheeler) has 20 years of process experience including testwork planning, execution and interpretation; studies of early-phase uranium projects; flowsheet development and optimization; and detailed design, commissioning and operations. He spent four years at McClean Lake, where he was involved in initial commissioning; five years at Rabbit Lake; and four years as Chief Metallurgist in Cameco’s head office, providing technical advice on uranium projects worldwide.

**Mr. Chuck Edwards** (Amec Foster Wheeler) has more than 45 years of experience, and has been involved in the design of all of the currently operating uranium facilities in Saskatchewan. He led the ore handling process design for the McArthur River and Cigar Lake projects. He led and managed the design for revision and expansion of the Key Lake mill for processing McArthur River ore, and process testing and engineering for optimizing the milling of the Cigar Lake plant feed. Since joining Amec Foster Wheeler in 2008, he has provided uranium processing consulting services, process design testwork, pilot plant design, processing plant design, and plant upgrade design to government and industrial clients in North America, Australia, and Africa.

Additional contributions to this PEA technical report were provided by:

- Mr. Mark B. Mathisen, CPG, (RPA), geology and mineral resource estimation

- Mr. Ross Greenwood, (SRK), mine geotechnical

## 2.3 Qualifications of SRK and SRK Team

The SRK Group comprises of more than 1,400 professionals, offering expertise in a wide range of resource engineering disciplines. The independence of the SRK Group 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 a large number of major international mining companies, the SRK Group has established a reputation for providing valuable consultancy services to the global mining industry.

## 2.4 Site Visit

The following SRK consultants visited the Wheeler River project site:

- Ross Greenwood, Senior Consultant – Geotechnical, from January 26 to 29, 2015. His personal inspection of the property included visits to the Phoenix and Gryphon deposit sites, inspection of drill core, a review of the drill program, a review of logging procedures and discussions with Denison technical staff.
- Ken Reipas, Principal Consultant – Mining, on January 29, 2015. His personal inspection of the property included visits to the Phoenix and Gryphon deposit sites, inspection of drill core, an assessment of site access and local infrastructure and discussions with Denison technical staff.
- Todd Hamilton, Principal Consultant – Hydrogeology, from June 8 to 10, 2015. His personal inspection included a review of the exploration drill program, organizing and conducting hydrogeological field investigations and discussions with Denison technical staff.

The following Amec Foster Wheeler consultant visited the McClean Lake mill site:

- Lorne Schwartz, Senior Engineering Specialist - worked for four years (1998 to 2002) at McClean Lake, where he was involved in initial mill commissioning.

## 2.5 Declaration

SRK's opinion contained herein and effective March 31, 2016 is based on information collected by SRK throughout the course of SRK's investigations. 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 is not an insider, associate or an affiliate 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

SRK has not performed an independent verification of land title and tenure information as summarized in Section 4 of this report. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties.

Mr. Reipas has relied on, and believes there is a reasonable basis for relying on, the following individual who has contributed the royalty and taxation information stated in this report, as noted below:

Mac McDonald, CFO Denison Mines Corp. for Sections 22.2, 22.3 and 22.6 (description of Saskatchewan royalties, description and application provincial/federal taxes, and post-tax economic results to Denison).

# 4 Property Description and Location

*This entire report section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada," Roscoe Postle Associates, November 25, 2015.*

## 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, which are operated by Cameco. 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. The estimated mineral resources contained in each deposit was last updated in the 2015 RPA technical report (RPA, 2015). 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 in the basement rocks, approximately 60 m to 350 m below the unconformity surface.

## 4.2 Land Tenure

The property comprises 19 contiguous claims held as a joint venture among Denison (60%), Cameco (30%), and JCU (Canada) Exploration Co. Ltd. (10%) with no back-in rights or royalties that need to be paid. The claims are shown in Figure 4-2 and listed in **Error! Reference source not found.**. Denison has been the operator of the property since November 10, 2004.

**Table 4-1: Land Tenure Details**

| Disposition # | Area (ha) | Annual Assessment ($) | Excess Credit ($) | Years Protected |
|---|---|---|---|---|
| S-97677 | 322 | 8,050 | 152,950 | 19 |
| S-97678 | 335 | 8,375 | 159,125 | 19 |
| S-97690 | 1,087 | 27,175 | 516,325 | 19 |
| S-97894 | 246 | 6,150 | 116,850 | 19 |
| S-97895 | 314 | 7,850 | 149,150 | 19 |
| S-97896 | 356 | 8,900 | 169,100 | 19 |
| S-97897 | 524 | 13,100 | 248,900 | 19 |
| S-97907 | 352 | 8,800 | 167,200 | 19 |
| S-97908 | 1,619 | 40,475 | 769,025 | 19 |
| S-97909 | 1,036 | 25,900 | 492,100 | 19 |
| S-98339 | 362 | 9,050 | 171,950 | 19 |
| S-98340 | 250 | 6,250 | 118,750 | 19 |
| S-98341 | 802 | 20,050 | 380,950 | 19 |
| S-98342 | 1,016 | 25,400 | 482,600 | 19 |
| S-98343 | 362 | 9,050 | 171,950 | 19 |
| S-98347 | 939 | 23,475 | 446,025 | 19 |
| S-98348 | 951 | 23,775 | 451,725 | 19 |
| S-98349 | 540 | 13,500 | 256,500 | 19 |
| S-98350 | 307 | 7,675 | 145,825 | 19 |

## 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, that 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 Saskatchewan mineral claim in good standing can be converted to a lease upon application. Leases have a term of 10 years and are renewable. A lease proffers 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.

## 4.4 Royalties and other Encumbrances

The property is subject to royalties levied by the Province of Saskatchewan (refer to Section 22.2). RPA is not aware of any other royalties due, back-in rights, or other encumbrances by virtue of any underlying agreements.

## 4.5 Permitting

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

RPA understands that Denison has all the required permits to conduct the proposed work on the property. RPA 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.

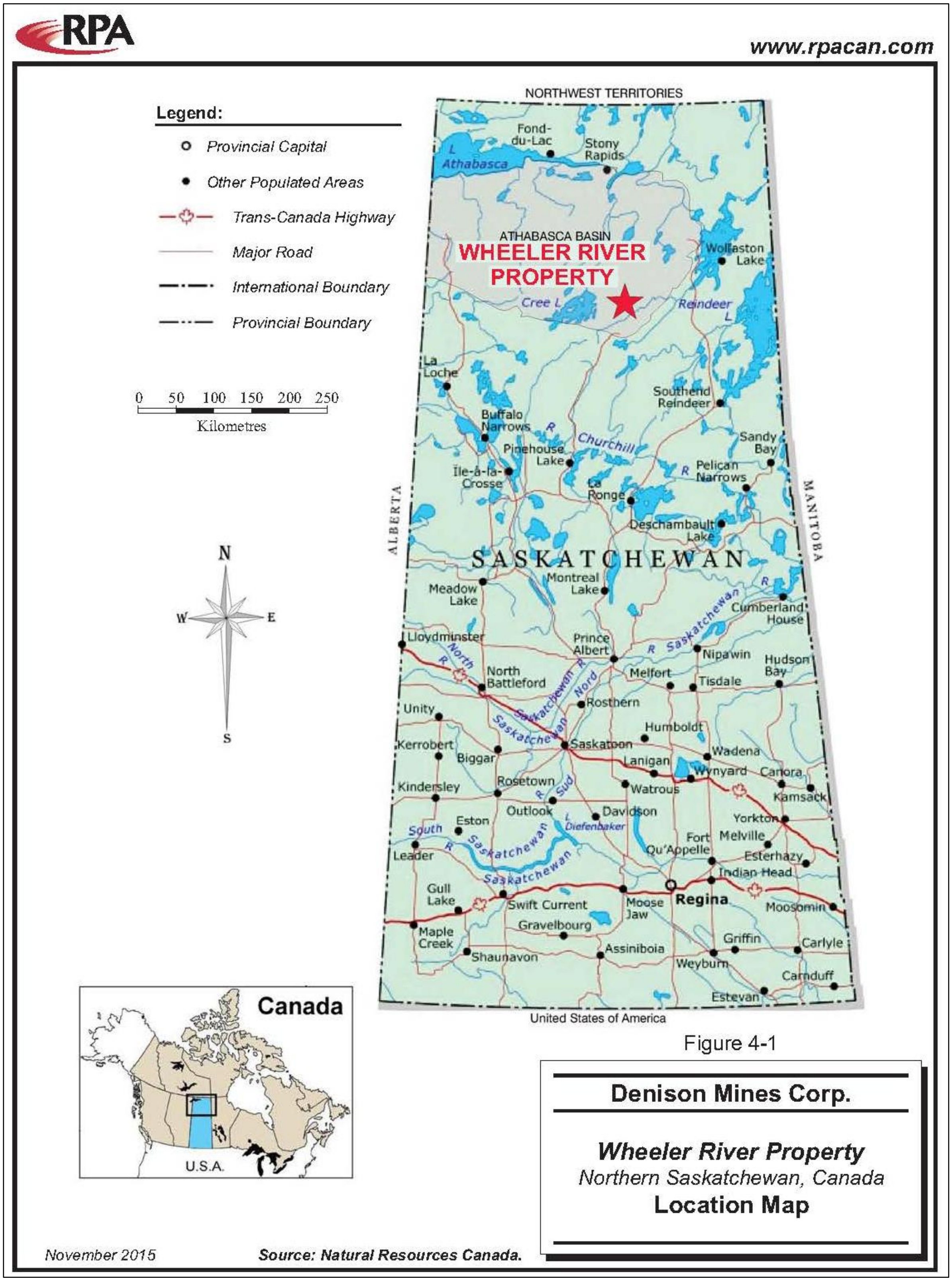



**Figure 4-1: Wheeler River Project Location Map**

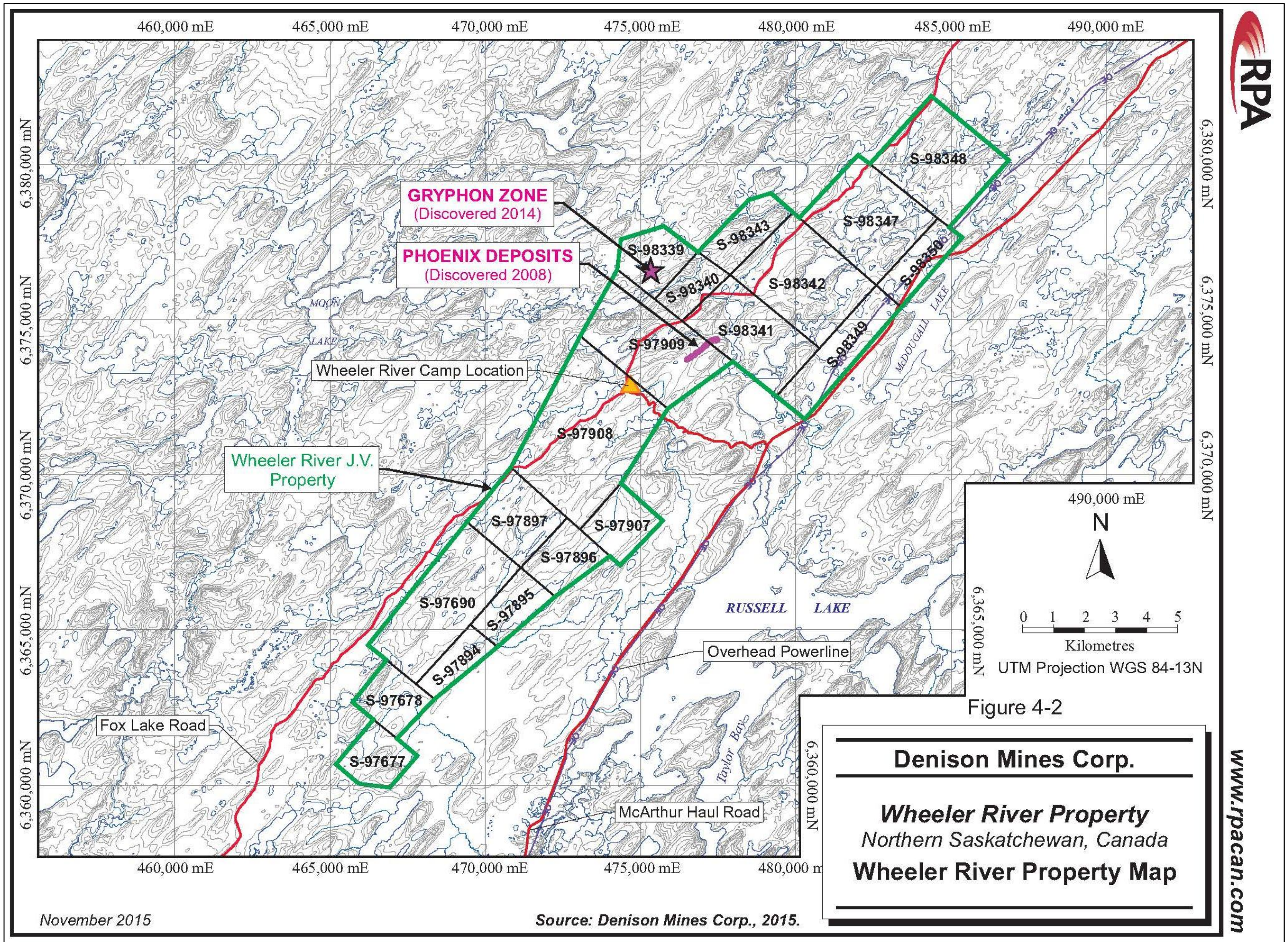



**Figure 4-2: Wheeler River Property Map**

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

*This entire report section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada," Roscoe Postle Associates, November 25, 2015.*

## 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 ore 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. Two airlines offer 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 off schedule. Contractor employees are generally on a longer work 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, owned and operated by Cameco, 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 m 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 m 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

*This entire report section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada," Roscoe Postle Associates, November 25, 2015.*

## 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 Wheeler River joint venture.

## 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 pelite units in the basement subcrop under the basal Athabasca sandstone. The graphitic pelite units are commonly intensely sheared and are highly conductive in contrast to the physically more competent adjoining rock types that include semipelite, 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 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 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 Wheeler River Joint Venture 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 Wheeler River Joint Venture 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.
- 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, desilification, 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 hanging wall 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 kilometre: 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.

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 fourteen 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. |
| 2008 | In 2008, three resistivity targets were drilled leading to the discovery of the Phoenix deposit. |
| 2008-2012 | During the period 2008 to 2012, drilling predominantly focused on defining the Phoenix deposits. |
| 2012-Present | Subsequent drilling has discovered 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 Consulting (Canada) Inc. (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). Both previous mineral resource estimates are superseded by the mineral resource estimate update in the 2014 Phoenix report, which incorporates additional drilling since 2012.

**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 lb $U_3O_8$ |
|---|---|---|---|
| Indicated | 152,400 | 15.6 | 52.3 |
| Inferred | 11,600 | 29.8 | 7.6 |

* Source: Roscoe, 2012

The current report includes the mineral resource estimate update documented in the 2014 Phoenix report as well as the initial mineral resource estimate for the Gryphon deposit. There are no previous mineral resource estimates for Gryphon.

## 6.4 Past Production

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

# 7 Geological Setting and Mineralization

*This entire report section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada", Roscoe Postle Associates, November 25, 2015.*

RPA notes that portions of the following geological descriptions are taken from internal Denison reports of 2009 to 2015.

## 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 (Figure 7-1 and Figure 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.

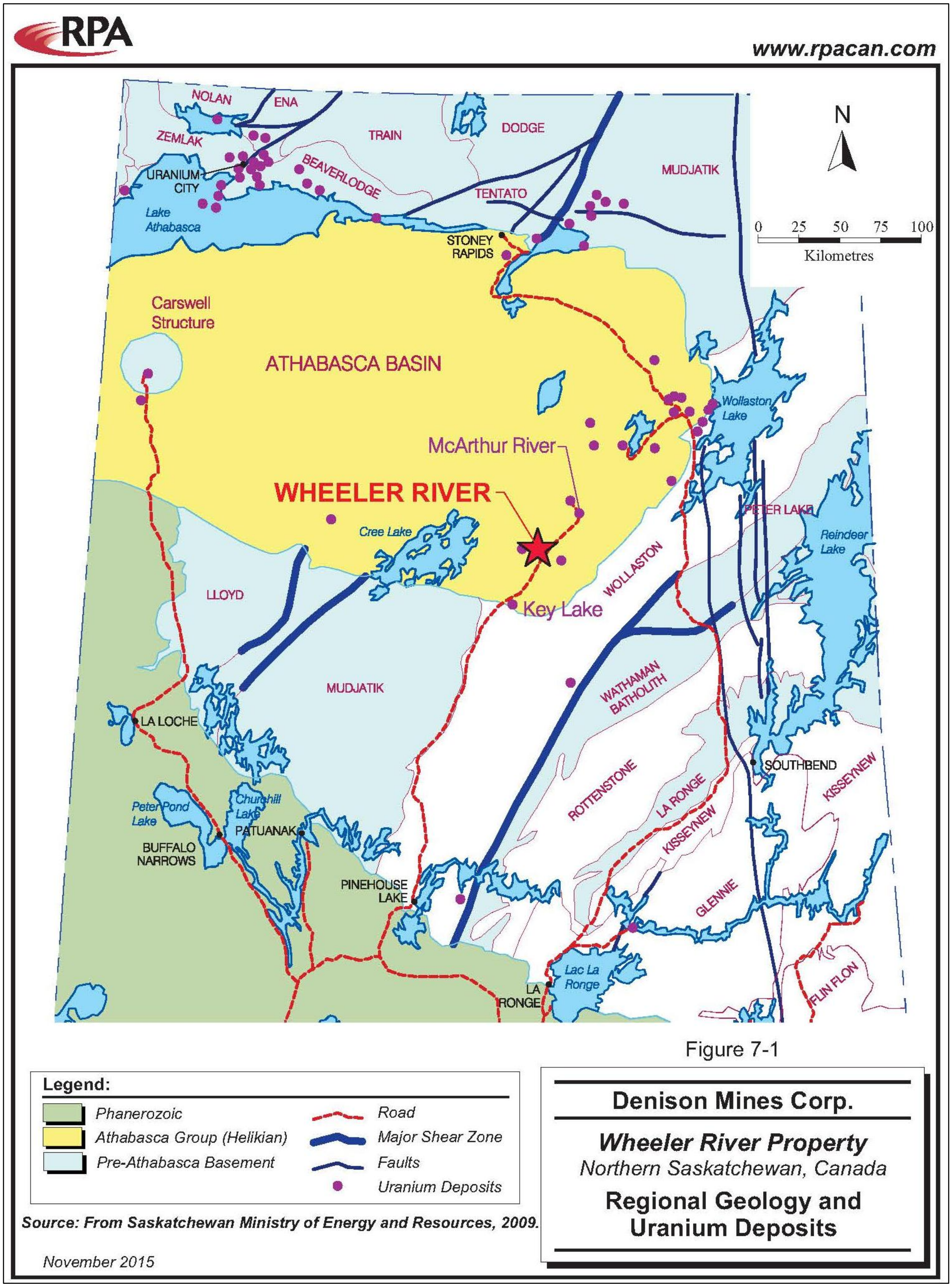



**Figure 7-1: Regional Geology and Uranium Deposits**

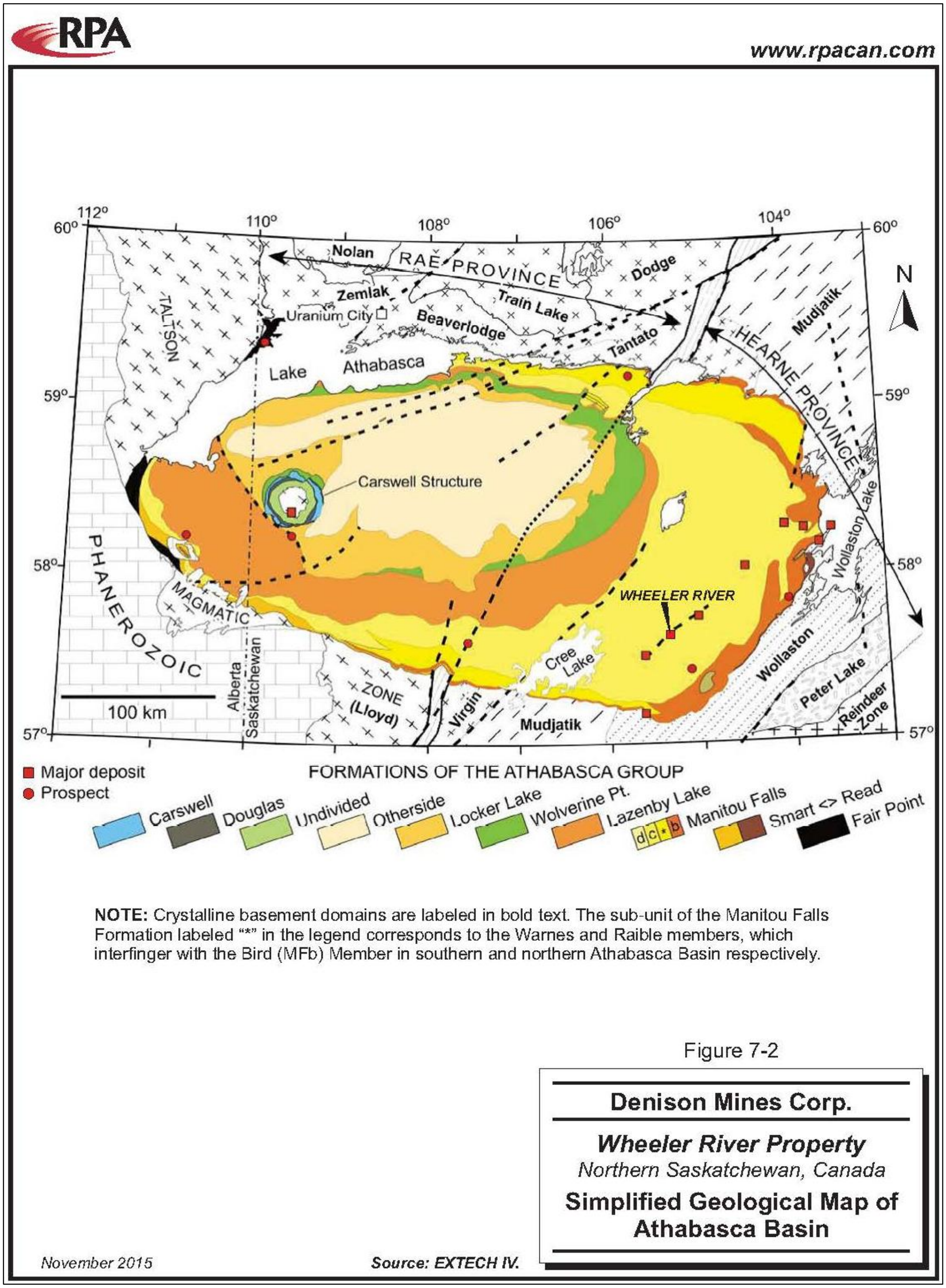



**Figure 7-2: Simplified Geological Map of Athabasca Basin**

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 metres 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, centimetres to metres 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).

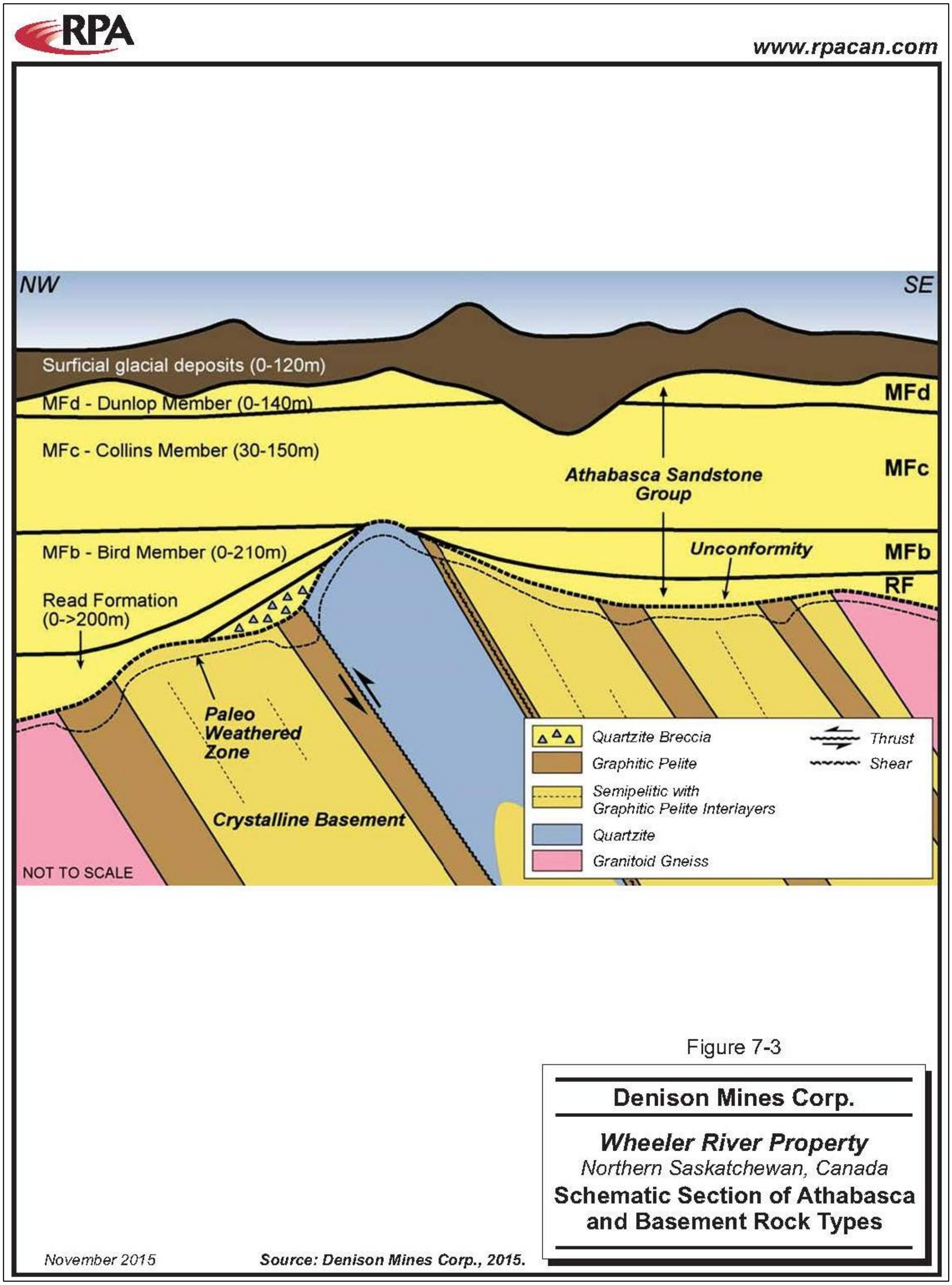



**Figure 7-3: Schematic Section of Athabasca and Basement Rock Types**

### 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 pelite 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 pelite), 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.

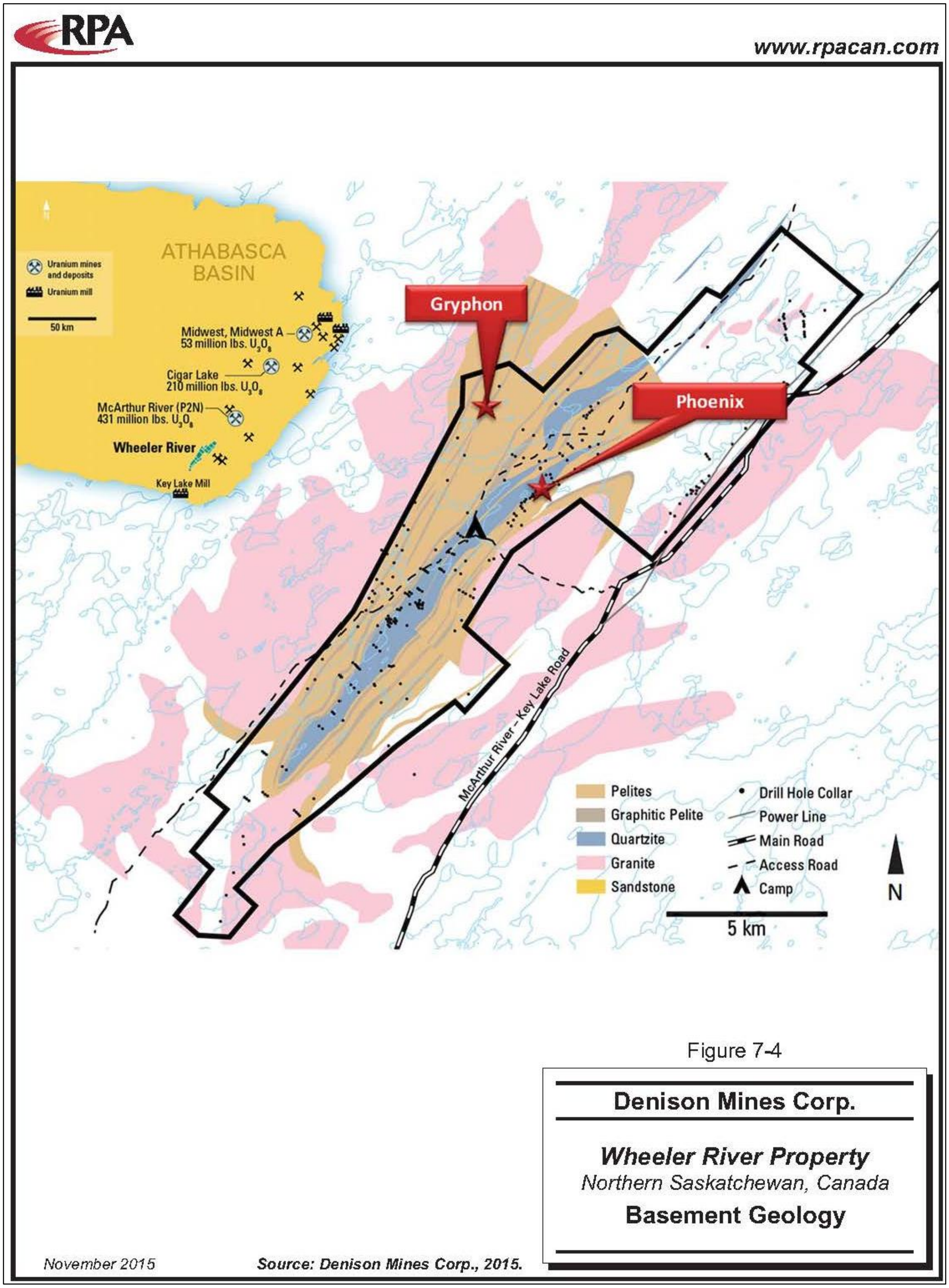



**Figure 7-4: Wheeler River Property Basement Geology**

### 7.3.5 Phoenix Deposit

The quartzite ridge, an interpreted impermeable and structural barrier forming the footwall to the mineralization (Figure 7-5), 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 pelite, which varies from seven metres to 60 m in thickness. This generally competent and unmineralized unit contains distinctive porphyroblastic garnets and acts as a marker horizon. Overlying the garnetiferous pelite is a graphitic pelite in which the graphite content varies from 1% to 40%. The graphitic pelite 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 pelite is a massive, non-graphitic, unaltered pelite 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 7-6).

### 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 (Figure 7-7):

1. Upper Graphite - The Upper Graphite is approximately 110 m thick, occurs furthest stratigraphically to the southeast, and is located in the hanging wall to the mineralization (Upper Lens). This pelitic 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 030° dipping at 50° to the southeast.

2. Quartz-Pegmatite Assemblage – Stratigraphically below the Upper Graphite is the Quartz-Pegmatite Assemblage. This unit is approximately 55 m thick and consists of several smaller (five metre to nine metre) discrete sub-units of alternating quartzite, quartz-rich pegmatite, pegmatite, and graphitic pelites.

3. Lower Graphite - Underlying the Quartz-Pegmatite Assemblage is the Lower Graphite. This pelitic unit is approximately 15 m thick and averages 10% to 15% graphite. It is well foliated and strikes approximately 030° and dips 45° to the southeast, and is located in the footwall to the Upper Lens mineralization. The Lower Lens mineralization occurs predominantly in this unit.

4. Basal Pegmatite – Stratigraphically below the Lower Graphite is the Basal Pegmatite. This is a pegmatitic to coarse grained granitic unit which is competent and relatively unaltered. Within the Basal Pegmatite, there are multiple minor (1 m to 2m) pelitic intervals.

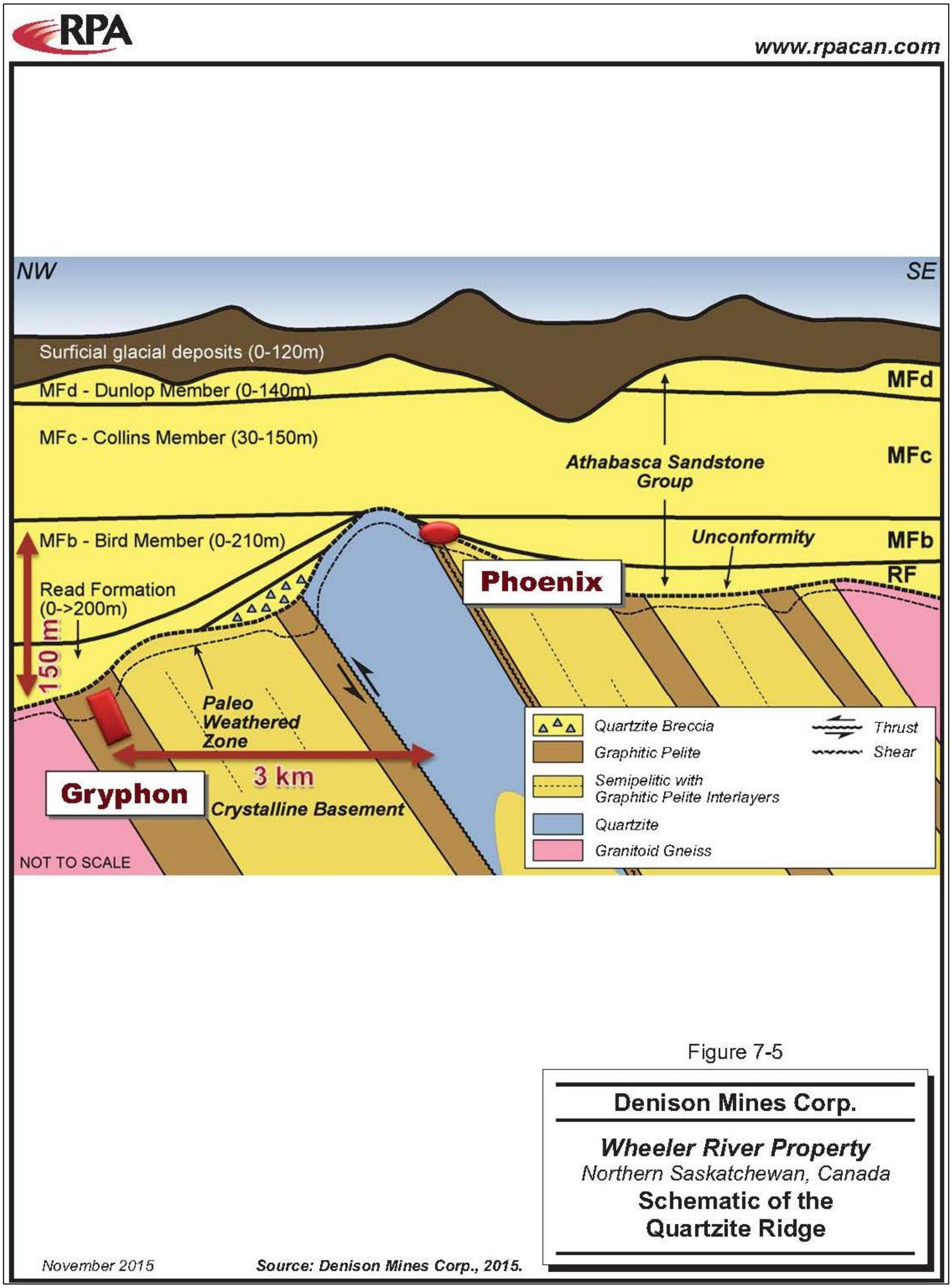



**Figure 7-5: Schematic of the Quartzite Ridge**

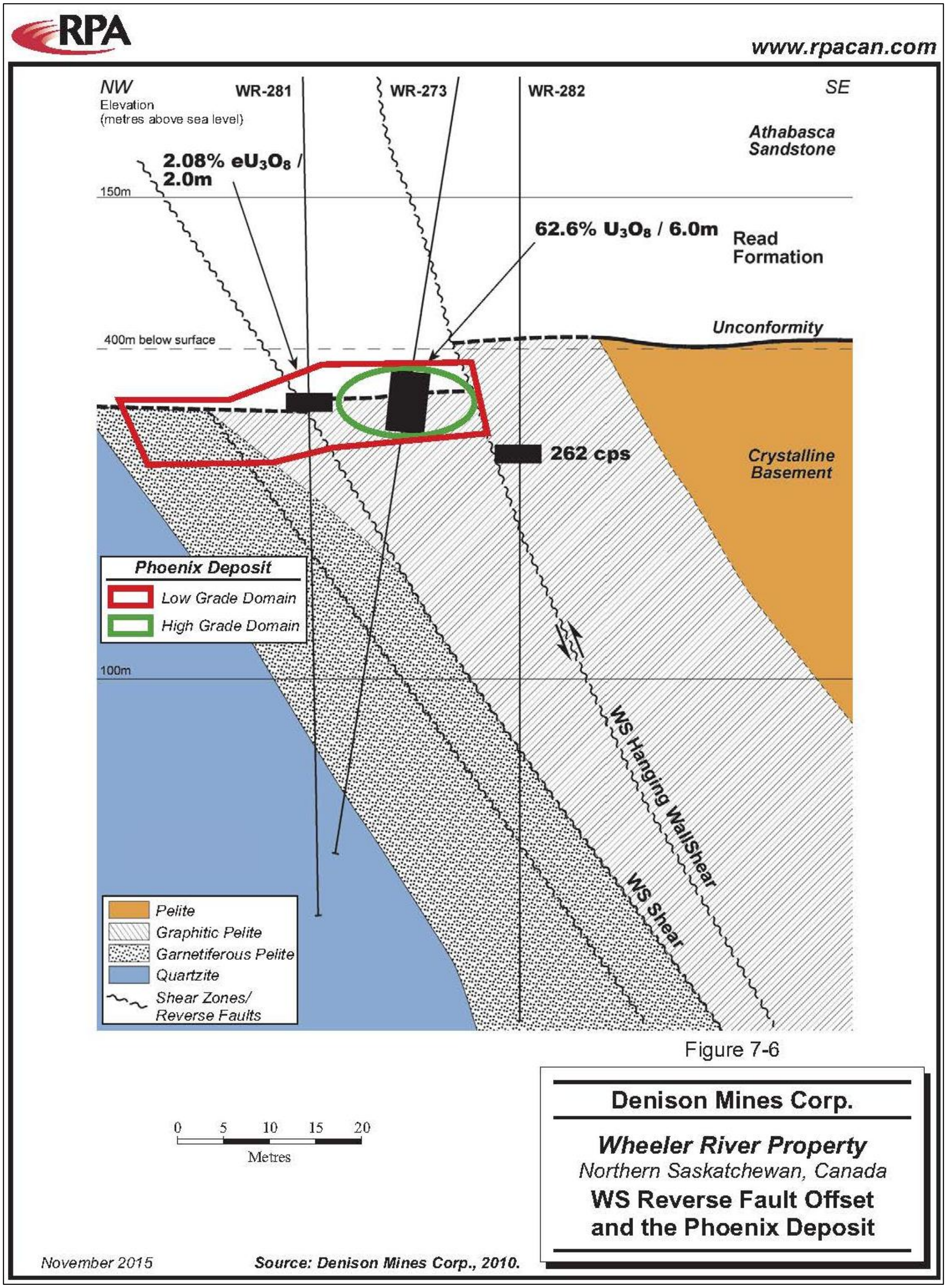



**Figure 7-6: WS Reverse Fault and the Phoenix Deposit**

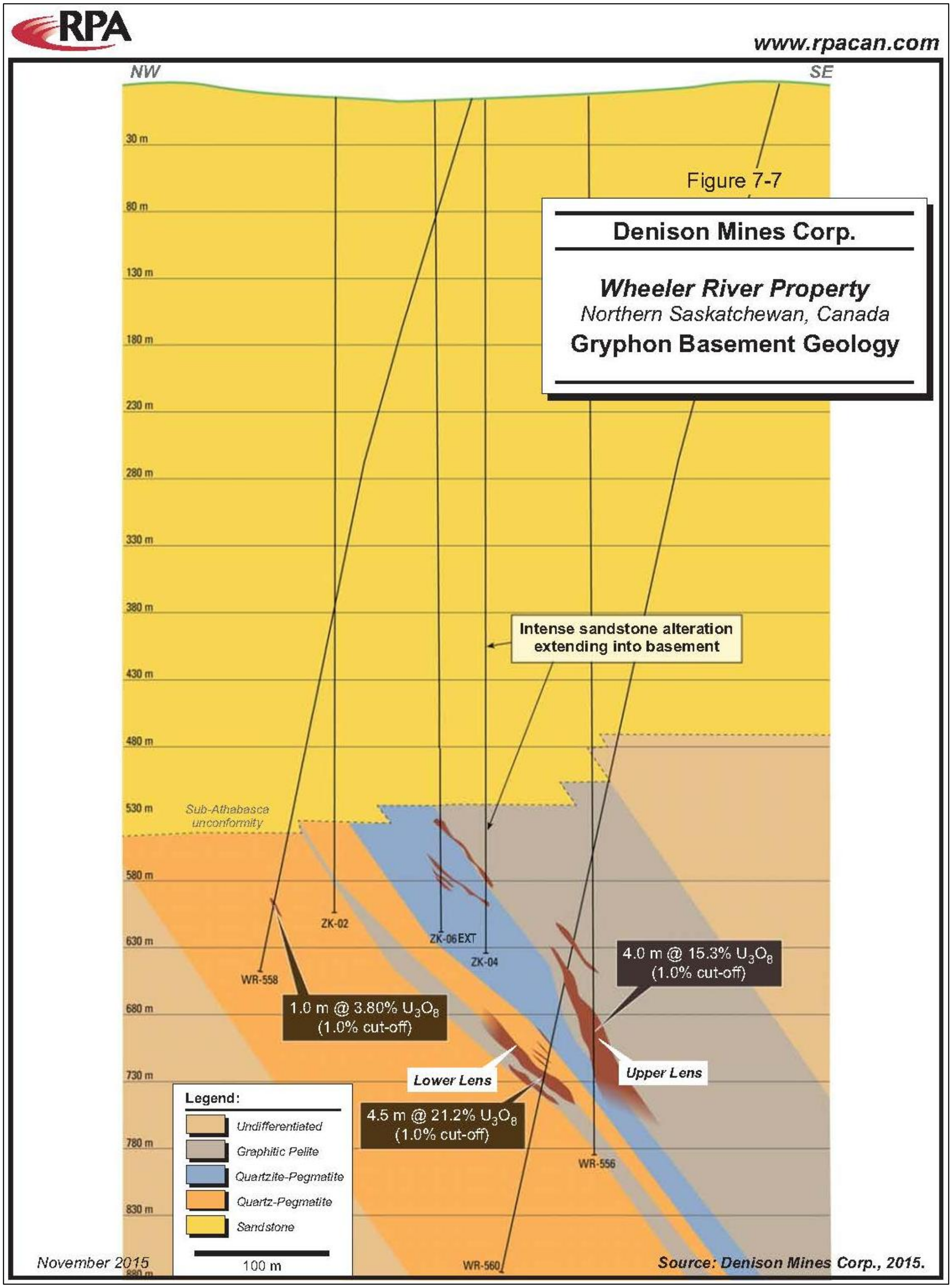



**Figure 7-7: Gryphon Simplified Basement Geology**

## 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 200 m above the unconformity and exhibit varying degrees of silicification and desilicification (which causes many technical drilling problems), as well as dravitization, chloritization, and illitization. In addition, hydrothermal hematite and druzy quartz are present in the sandstone and commonly in the basement rocks. Alteration is focussed 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.

### 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.
- From approximately 150 m to 440 m, there is no significant alteration. Diagenetic hematite banding is predominant.
- From approximately 440 m to 540 m, 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
  - 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 hanging wall to the Gryphon mineralization, although drill holes that intersected an up-faulted basement wedge exhibit moderate silicification with preserved diagenetic hematite.

Sandstone alteration in the footwall to the Gryphon mineralization consists of isolated alteration zones with strong bleaching, grey alteration, silicification, and vuggy quartz. The isolated zones of alteration are assumed to be related to the up-dip projection of the offsetting basement reverse faults to the southeast.

Basement alteration exhibits a zoned sequence around mineralization. Directly below the unconformity, the typical basement paleoweathering profile is preserved. Distal alteration includes chlorite and sericite. Proximal alteration signatures include weak bleaching, dravite and druzy quartz formation. Strong clay replacement, pervasive bleaching, and strong dravite are intimately associated with mineralization. Hematite is also commonly directly associated with mineralization. Clay alteration mineralogy is dominated by illite with subordinate kaolinite and chlorite.

## 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 kilometres of relatively incompetent graphitic pelite 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).

### 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 pelite unit along the east edge (hanging wall) 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 data currently available, 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 hanging wall sequence are always less than 10 m (Figure 7-6).

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 hanging wall 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 hanging wall 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. 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.

### 7.5.2 Gryphon Deposit

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 the less competent graphitic pelites and more competent quartz-pegmatites, pegmatites, and pelitic units. They are described as a combination of cataclasites and gouges, and intervals of blocky and friable core. The most significant structures occur at the contact of the upper graphite with the overlying pelite and at the base of the upper graphite in contact with the underlying quartz-pegmatite. These structures are termed the Offset Fault and Graphitic Fault (G-Fault), respectively. Mineralization generally occurs along the G-Fault and its associated subordinate parallel structures where a shallowing of stratigraphic foliation is observed. Structural data analysis has recorded several thrust faults with reverse sinistral movement. The shallowing of foliation in combination with reverse sinistral movement would have provided a zone of dilation, amenable to fluid movement and uranium precipitation.

# 7.6 Mineralization

## 7.6.1 Phoenix Deposit

Uranium mineralization at the Phoenix deposit occurs at the unconformity between Athabasca sandstones and basement rocks, with the most intense mineralization adjacent to the WS fault. A minor amount is basement fracture hosted mineralization extending below the north part of Zone A. The Phoenix deposit can be classified as an unconformity-associated uranium deposit.

Mineralization and alteration have been traced over a strike length of approximately one kilometre. Since discovery hole WR-249 was drilled in 2008, 253 drill holes have reached the target depth, delineating two distinct zones (A and B) of high-grade uranium mineralization.

Mineralization is in the form of the oxide uraninite/pitchblende ($UO_2$).

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, and 9,143 ppm Pb. 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

Mineralization at Gryphon occurs 720 m below surface and is centred approximately 220 m below the sub-Athabasca unconformity. It is within 80 m of the unconformity at its highest point and 370 m below the unconformity at its deepest point. The deposit consists of a set of parallel, stacked, elongate lenses that are broadly conformable with the basement geology and associated with a significant fault zone (G Fault) that separates a thin unit of quartzite (quartz-pegmatite) from an overlying graphitic pelite (upper graphite). The lenses dip moderately to the southeast and plunge moderately to the northeast. The deposit is approximately 450 m long in the plunge direction and 80 m wide across the plunge. Thickness is variable and is a function of the number of stacked lenses present, generally varying between 2 m and 20 m. To date, the majority of mineralization is hosted within two lenses associated with the upper and lower graphite units. Two predominant types of mineralization have been noted:

- Irregular Fracture Fill - Weak, dark black, low grade mineralization occurring as blebs and foliation-parallel fracture fill associated with breccias and centimetre-scale dravite veinlets in the Upper Graphite.
- Semi Massive - Black, high grade mineralization associated with hematite and secondary uranium minerals occurring as lenses and pods parallel to foliation. "Worm rock" textures are also observed.

Mineralization at Gryphon is dominated by uraninite/pitchblende, with very minor coffinite and trace carnotite, uranophane, and brannerite. 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 pyrs are noted comprising galena, chalcopyrite, and pyrite.

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, and 3,525 ppm Pb. These concentrations are lower than those recorded for the Phoenix deposit.

# 8 Deposit Types

*This entire report section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada," Roscoe Postle Associates, November 25, 2015.*

Both the Phoenix and Gryphon deposits are classified as Athabasca Basin unconformity-associated uranium deposits. Phoenix straddles the unconformity contact between the Athabasca Sandstone and underlying basement, while Gryphon is entirely hosted in the basement rocks. 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 metapelite 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 metres 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 (e.g., 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 (e.g., 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 metres 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 pelites 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 metres 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.

Figure 8-1 and Figure 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.

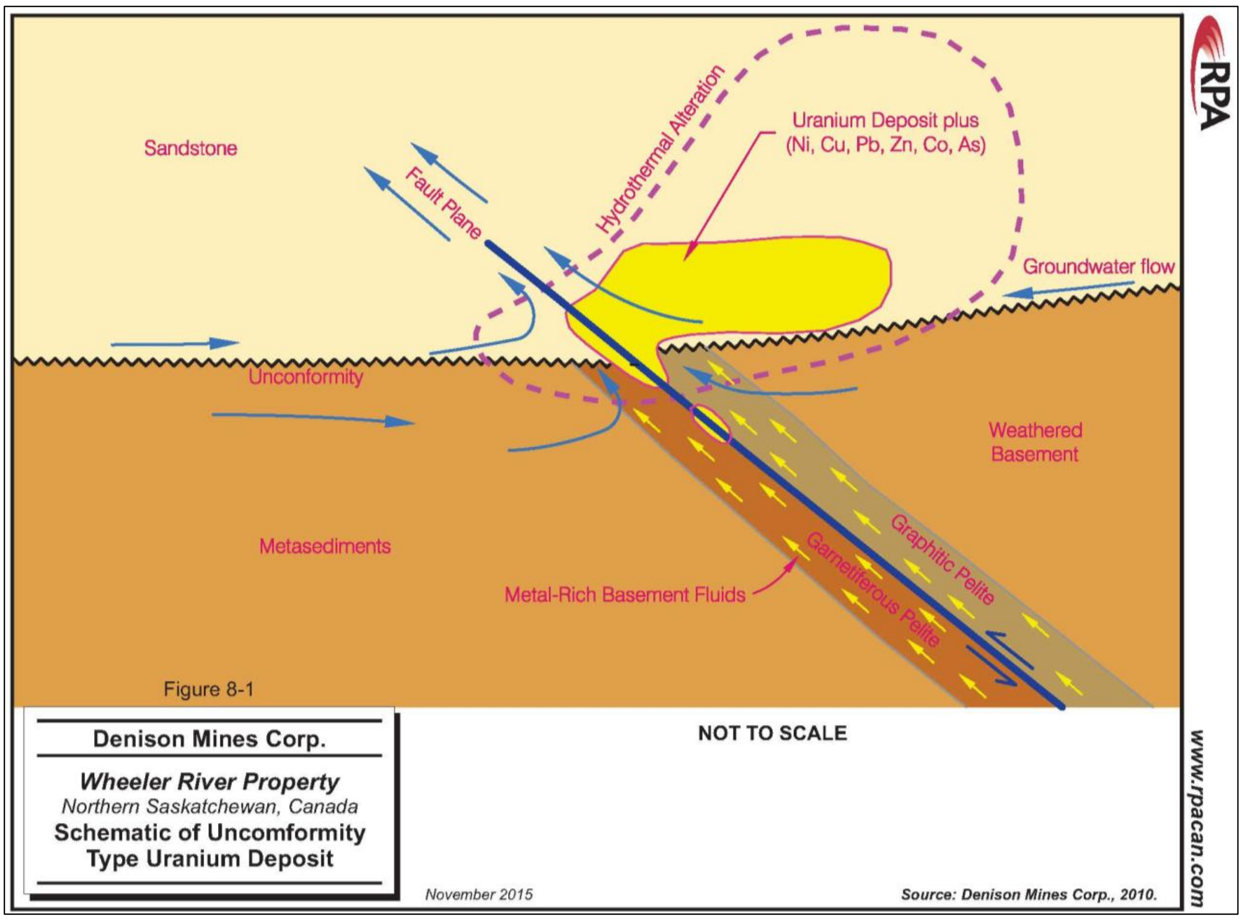



**Figure 8-1: Schematic of Unconformity Type Deposit**

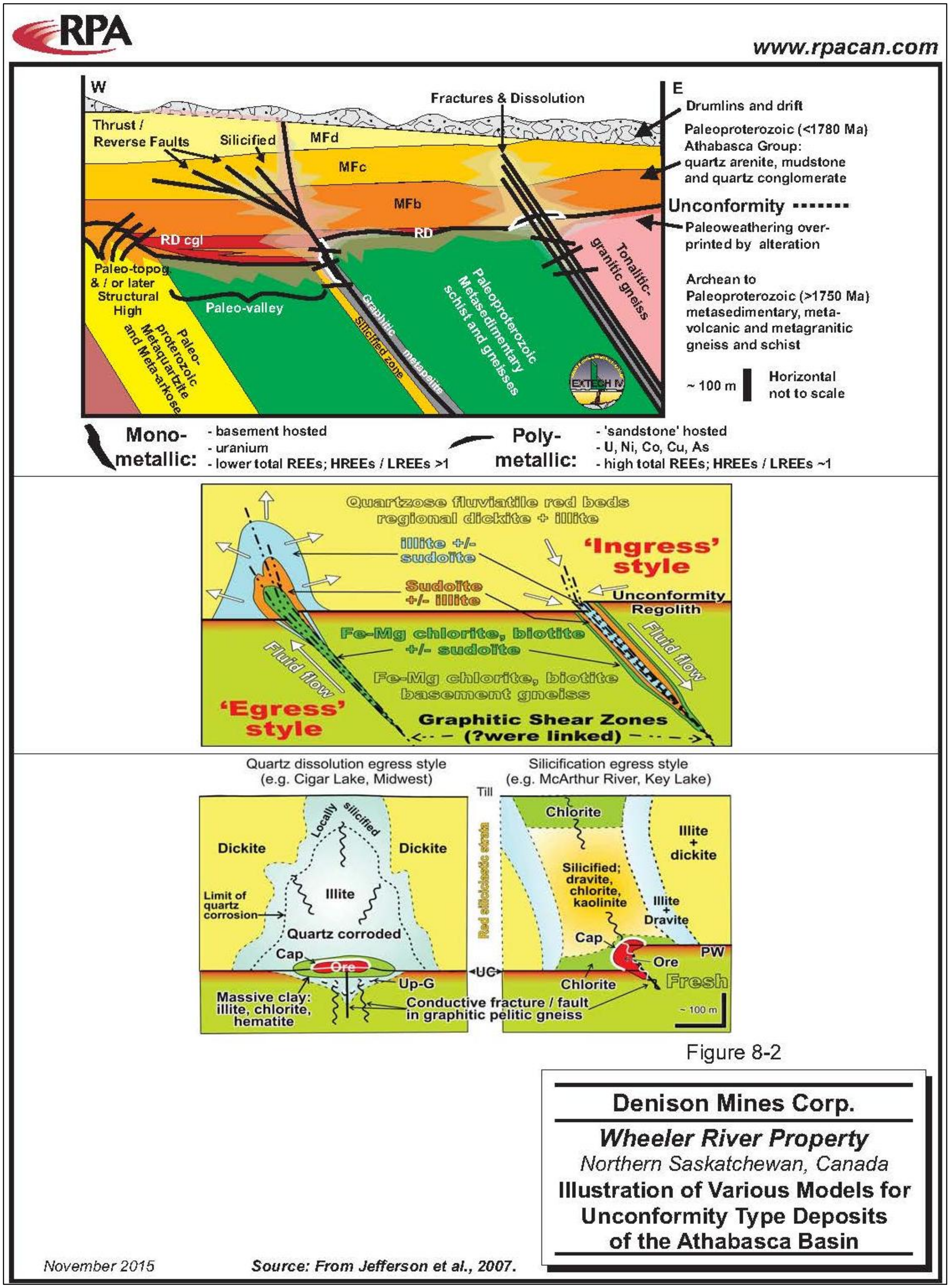



**Figure 8-2: Various Models for Unconformity Type Deposits of the Athabasca Basin**

# 9 Exploration

*This entire report section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada," Roscoe Postle Associates, November 25, 2015.*

Since discovery of the McArthur River deposit in 1988, the McArthur River exploration model (McGill et al., 1993) has emphasized a different association between uranium mineralization and rock type compared to the earlier Key Lake exploration model. At McArthur River, one of the most significant rock types in the basement succession is a massive, homogenous, and competent quartzite. Mechanically, particularly compared to the adjacent layered members of the basement stratigraphy, the quartzite is extremely competent, and thus exerts an important control both in basement and post-Athabasca sandstone structural evolution. Both the footwall and hanging wall contacts of the quartzite unit, particularly where these contacts involve highly incompetent rocks such as graphitic pelite, are sites of major thrust and strike-slip faults.

Although these faults are loci for mineralization; the poor conductivity, low magnetic susceptibilities, and low density values associated with the quartzite limits the effectiveness of airborne and ground geophysical methods in mapping these basement units especially when they are covered by hundreds of metres of sandstone. Another noteworthy characteristic of McArthur River type mineralization is the widespread presence of hydrothermal dravite, indicating boron addition into the overlying Athabasca sandstone. Thus, borehole geochemistry and drilling are the primary exploration methods.

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 Drilling.

## 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 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 Wheeler River joint venture 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 Wheeler River Joint Venture 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).

## 9.2 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

*This entire report section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada,"", Roscoe Postle Associates, November 25, 2015.*

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 641 diamond drill holes and 84 reverse circulation (RC) drill holes totalling 302,127 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 |
| **TOTAL** | | **641** | **84** | **302,127** |

## 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 kilometres northeast of the previous work in the WR-204 area and, more significantly, was drilled on the hanging wall 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 hanging wall to the quartzite unit. This new discovery was termed Phoenix.

During 2009, three drill programs consisting of a total of 43 diamond drill holes (19,006 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 pelite 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 kilometre.

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, 59 totalling 27,853.25 m were completed to the desired depth and three were lost or abandoned due to poor ground conditions or excessive deviation. The three lost holes were re-drilled and successfully 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, 77 were successfully completed to design depth.

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

In 2013, 30 diamond drill holes totaling 13,797 m were carried out on mineral dispositions across the property of which 14 were completed as infill delineation drilling on Phoenix Zone A.

In 2014, an additional 11 diamond drill holes were completed on Phoenix Zone A to extend higher grade portions of the deposit.

Since 2008, 263 drill holes totalling 123,749 m of drilling have delineated the Phoenix deposit (Figure 10-1, 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 | 71 | 34,436 |
| | 2012 | Denison | 51 | 23,073 |
| | 2013 | Denison | 25 | 12,083 |
| | 2014 | Denison | 10 | 4,298 |
| | 2015 | Denison | 6 | 2,862 |
| **Phoenix Total** | | | **263** | **123,749** |
| | | **Target** | **# Holes** | **Total Drilled (m)** |
| | | Zone A | 137 | 62,678 |
| | | Zone B | 55 | 25,347 |
| | | Zone C | 24 | 10,438 |
| | | Zone D | 27 | 15,214 |
| | | North Target | 18 | 9,482 |
| | | East Target | 2 | 591 |

To date, the Phoenix deposit area has been systematically drill tested over approximately one kilometre 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 were transferred into a Fusion database. All drill hole data were 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.

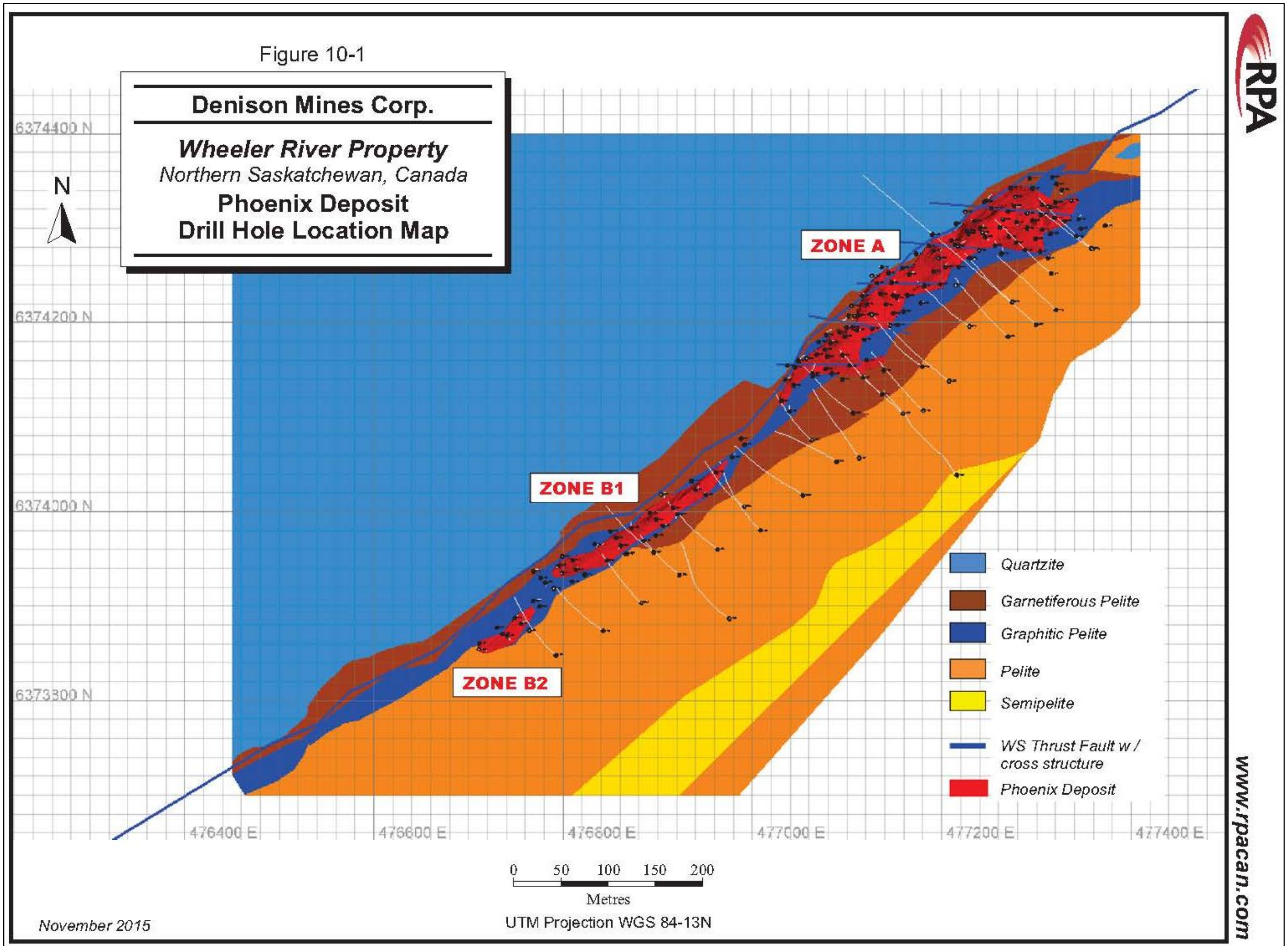



**Figure 10-1: Phoenix Deposit Drill Hole Location Map**

## 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 this 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:

1. The unconformity down-dip of a sandstone structure intersected in ZK-06
2. 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 pelite 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 was termed the Upper Lens.

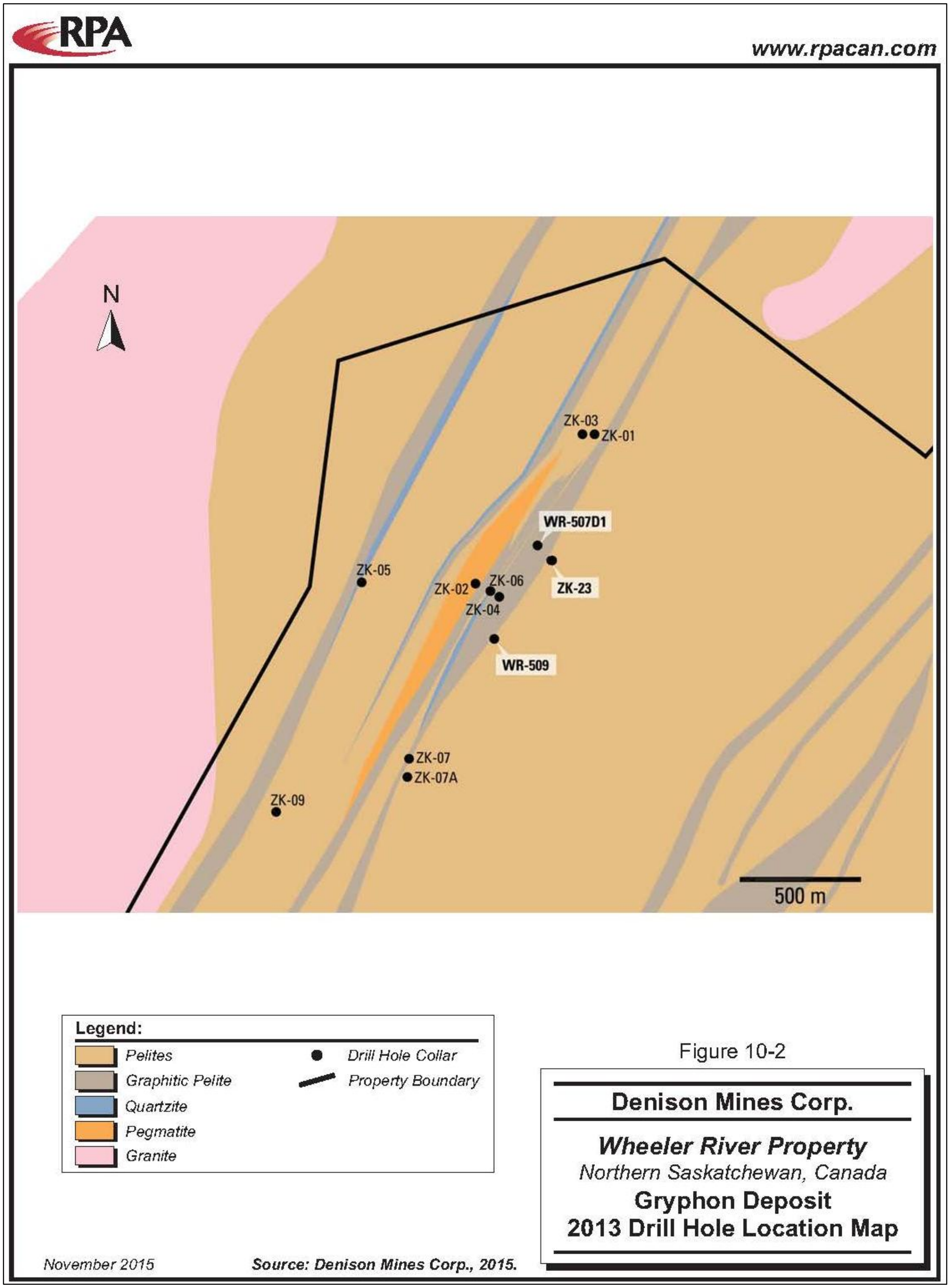



**Figure 10-2: Gryphon Deposit 2013 Drill Hole Location Map**

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. The mineralization graded 7.3% $U_3O_8$ over 0.5 m from 611.7 m and is considered peripheral mineralization to the Gryphon Upper Lens. 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 was termed the Lower Lens. 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 both the Upper Lens and Lower Lens. The Upper Lens has 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 winter and summer drilling campaigns with an additional 37 holes, totalling 21,591 m. As of September 17, 2015, the effective date of the current mineral resource estimate, Denison and predecessor companies have drilled a total of 69 holes totalling 44,083 m over the Gryphon deposit. Table 10-3 lists the holes by drilling program.

**Table 10-3: Gryphon Drilling Statistics**

| Deposit | Year | Company | # Holes | Total Drilled (m) |
|---|---|---|---|---|
| Gryphon | 1988 | SMDC | 3 | 1,848 |
| | 2001 | Cameco | 1 | 584 |
| | 2013 | Denison | 3 | 1,515 |
| | 2014 | Denison | 25 | 18,546 |
| | 2015 | Denison | 37 | 21,591 |
| **Gryphon Total** | | | **69** | **44,083** |

Diamond drilling at Gryphon was primarily done with NQ sized core (47.6 mm diameter) with 68 of 76 holes angled between 67° to 79° to the northwest with the remaining holes drilled vertically.

Highlights from the Gryphon drilling program are listed in 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 |

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

Since the September 25, 2015 Mineral Resource estimate, Denison has disclosed significant results from two drill holes on the Wheeler River property in the vicinity of the Gryphon deposit. Drill hole WR-633D1 intersected 5.7% $eU_3O_8$ over 1.0 m and 6.3% $eU_3O_8$ over 1.7 m (Denison News Release February 9, 2016). Drill hole WR-641 intersected 6.7% $eU_3O_8$ over 5.3 m (Denison News Release March 10, 2016). Radiometric uranium assays ($eU_3O_8$) are based on total gamma down-hole probe results. Denison expects true thicknesses to be approximately 75% of the intersection lengths. WR-

633D1 is located approximately 100 m north of the Gryphon deposit and WR-641 is approximately 160 m northwest of the Gryphon deposit.

## 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. In general, most of the drilling was completed on northwest-southeast oriented profiles spaced approximately 50 m apart.

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 metres 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. For the Gryphon mineral resource estimate, 100% of the $U_3O_8$ grade data are obtained from chemical assay of the rock.

### 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.

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 214Bi (bismuth 214), and in the thorium series from decay of 208Tl (thallium 208).

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, or 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 e $U_3O_8$ 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 Saskatchewan Research Council (SRC) uranium calibration pits (sodium iodide crystal) and empirical values developed in-house (Sweet and Petrie, 2010) for the triple-gamma probe (Figure 10-3).

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 metres 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.

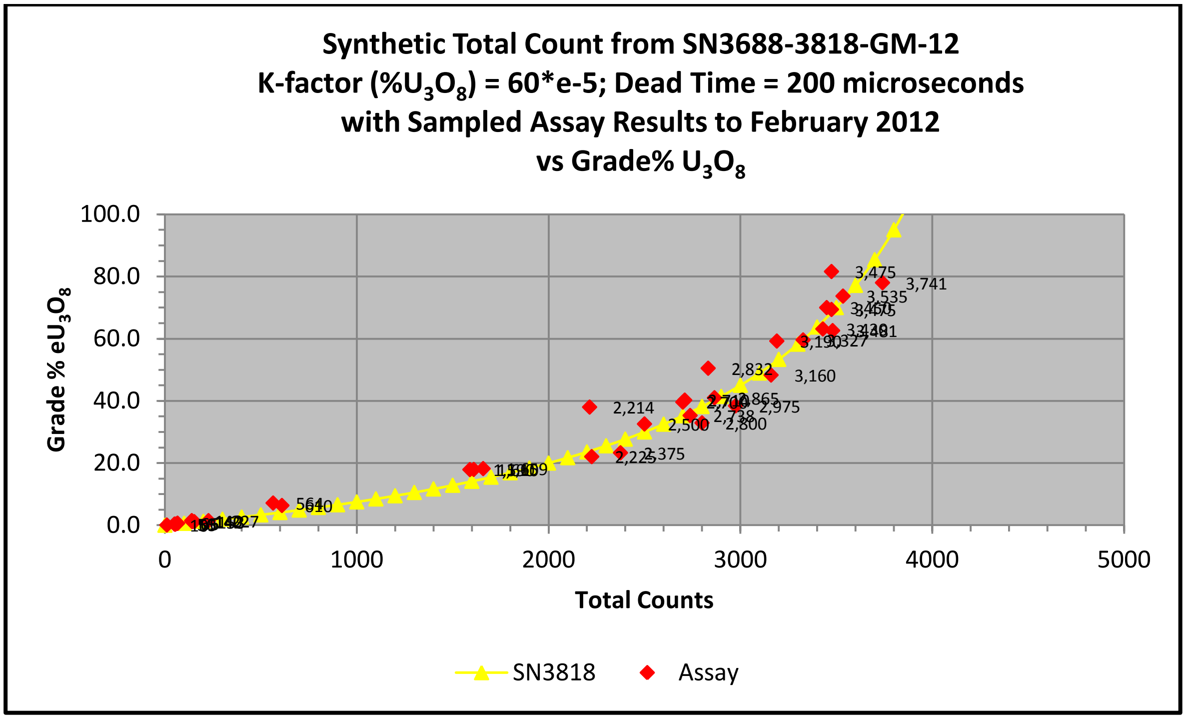



**Figure 10-3: Calibration Curve for Geiger-Meuller SN 3818 Probe**

## 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 metres 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 three metre 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.

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) (Section 11).

#### 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 centimetres 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 centimetre 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 excellent. Of the 69 drill holes drilled at Gryphon, 19 drill holes contained only radiometric data and were not sampled for assay, 36 drill holes contained both radiometric and assays, and 14 drill holes did not have any grade data. In total there were 55 drill holes used to interpret the mineralized domains, but only the 36 holes containing assays were used for mineral resource estimate. There are 1,019 $U_3O_8$ assay records totalling 510 m in the Gryphon deposit database

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

*This entire report section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada," Roscoe Postle Associates, November 25, 2015.*

As described in Section 10 Drilling, 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 Saskatchewan Research Council 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) or AusSpec International Ltd. (AusSpec), respectively, for interpretation. 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 is 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" 2000 – 2,999 cps
- "3 Dots" 3000 – 3,999 cps
- "4 Dots" 4000 – 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 HNO3 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.

In 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/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 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.

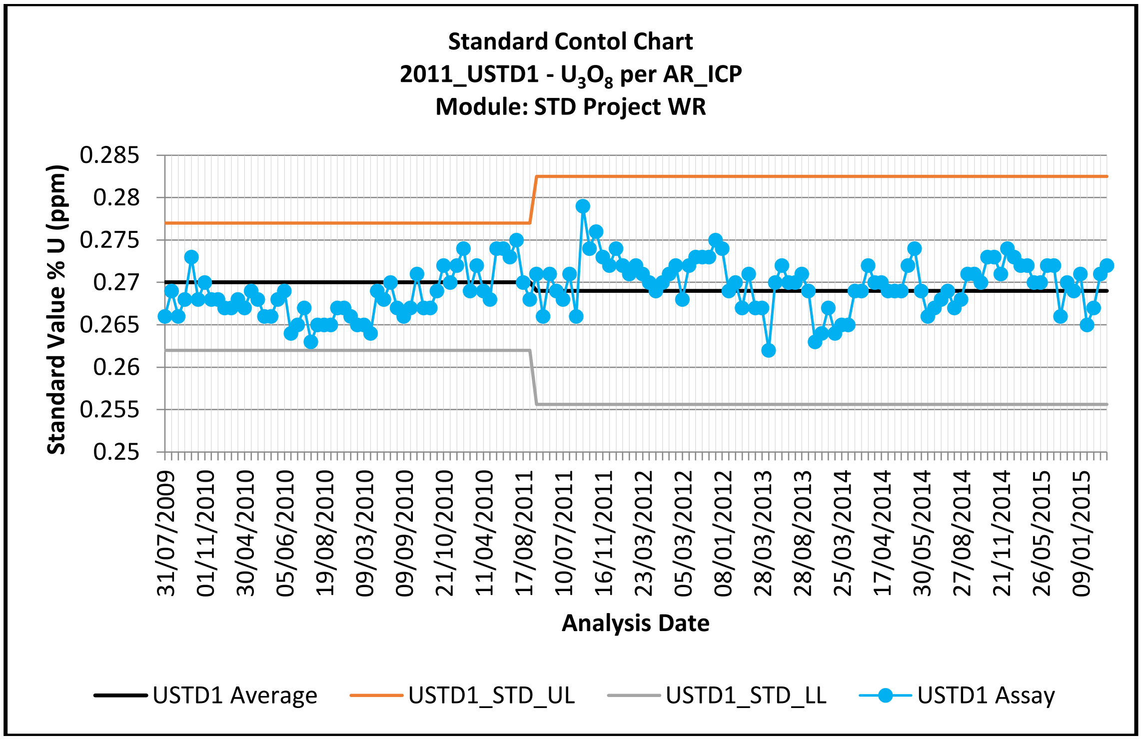



**Figure 11-1: USTD1 Analyses**

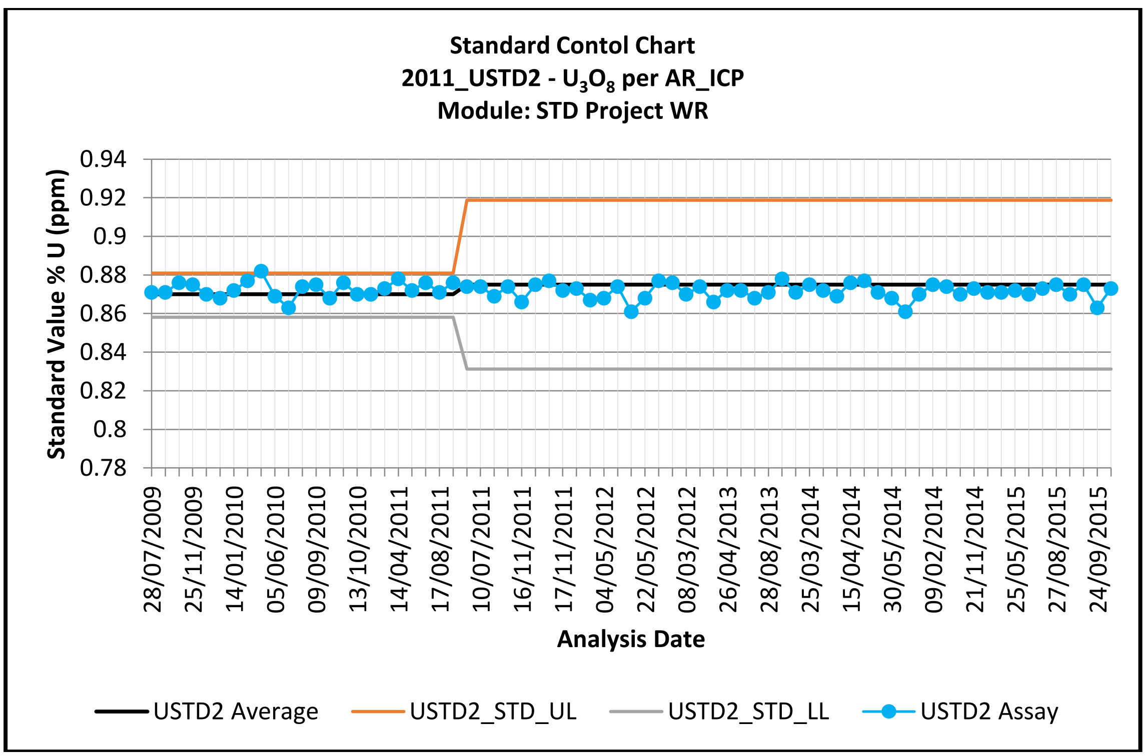



**Figure 11-2: USTD2 Analyses**

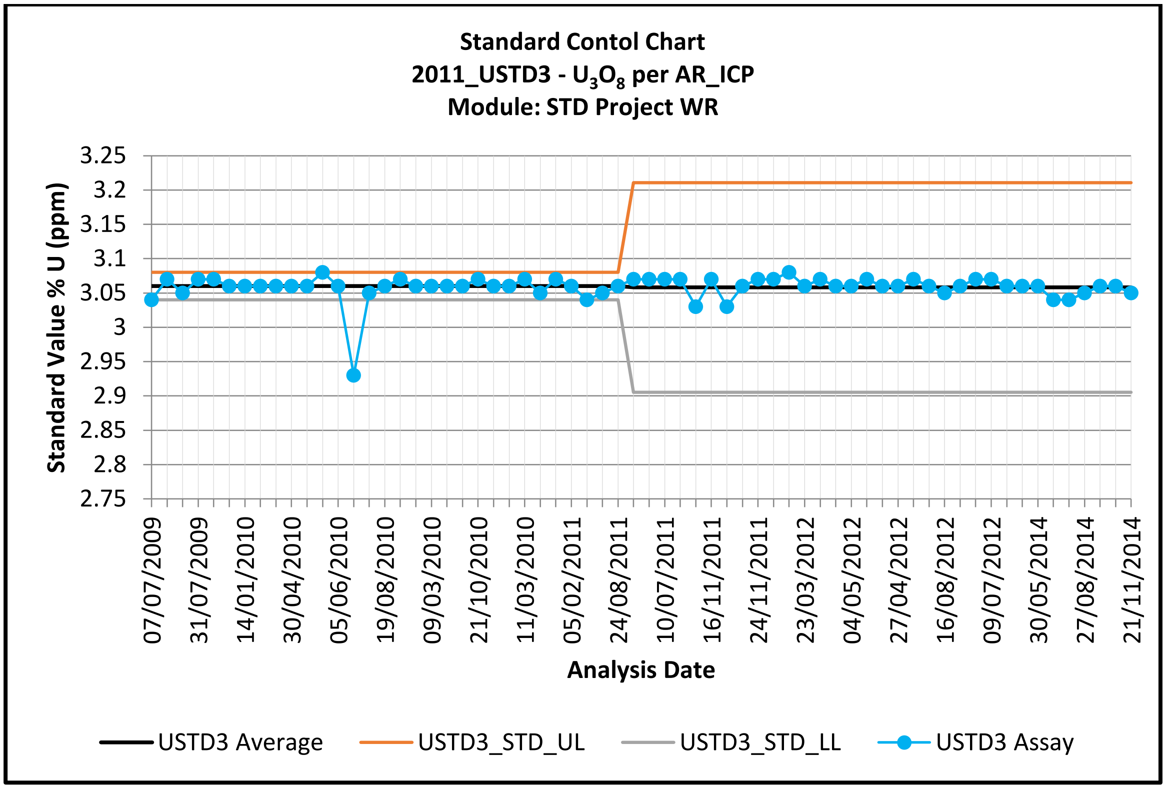



**Figure 11-3: USTD3 Analyses**

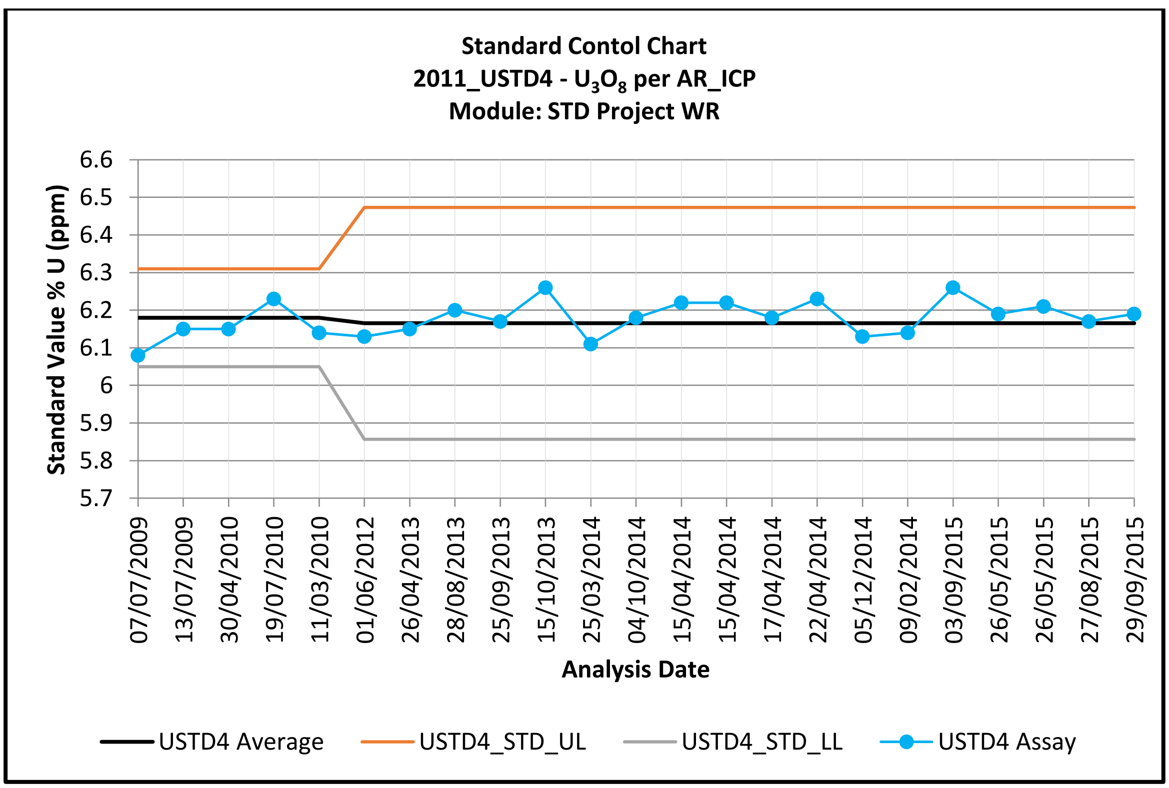



**Figure 11-4: USTD4 Analyses**

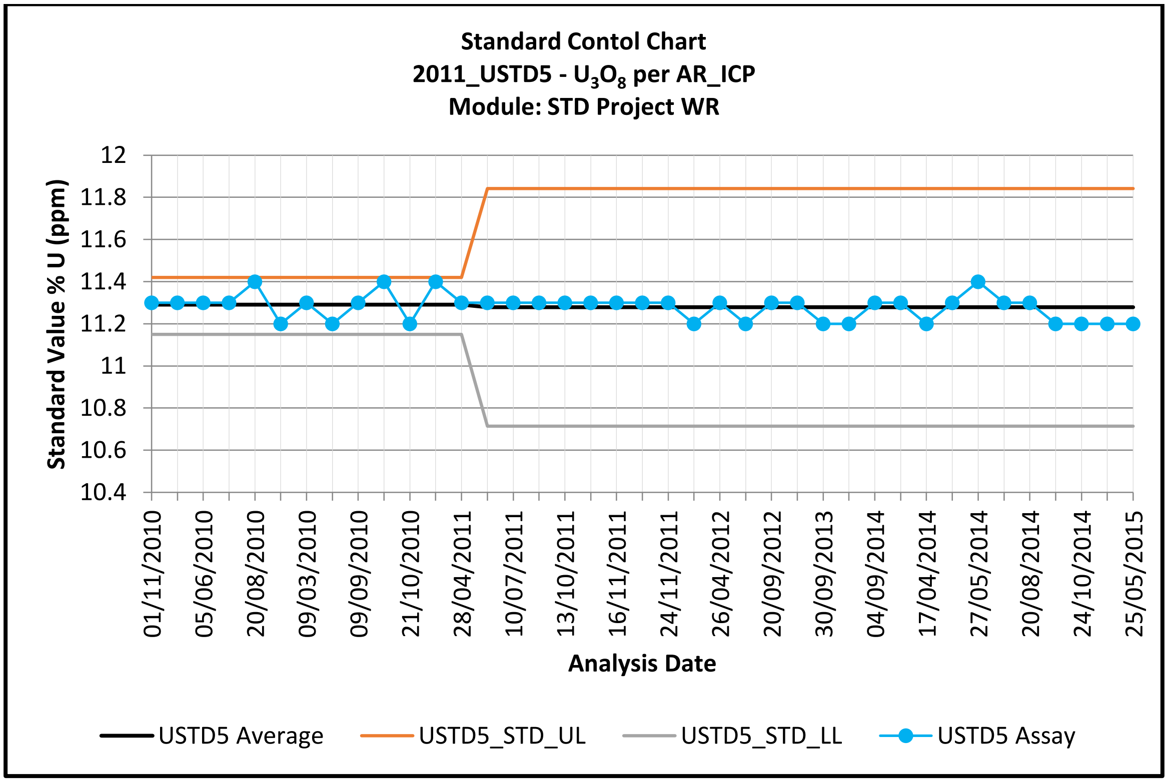



**Figure 11-5: USTD5 Analyses**

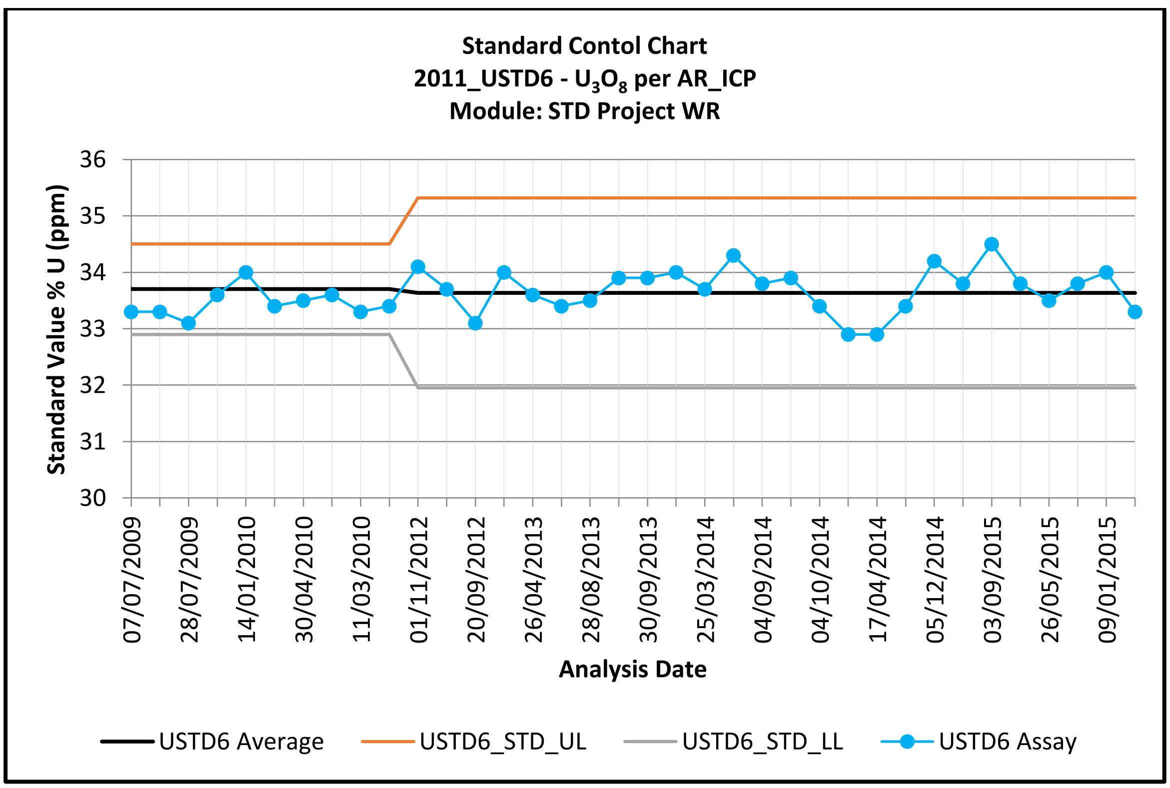



**Figure 11-6: USTD6 Analyses**

## 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 0.024% $U_3O_8$.

## 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 98%.

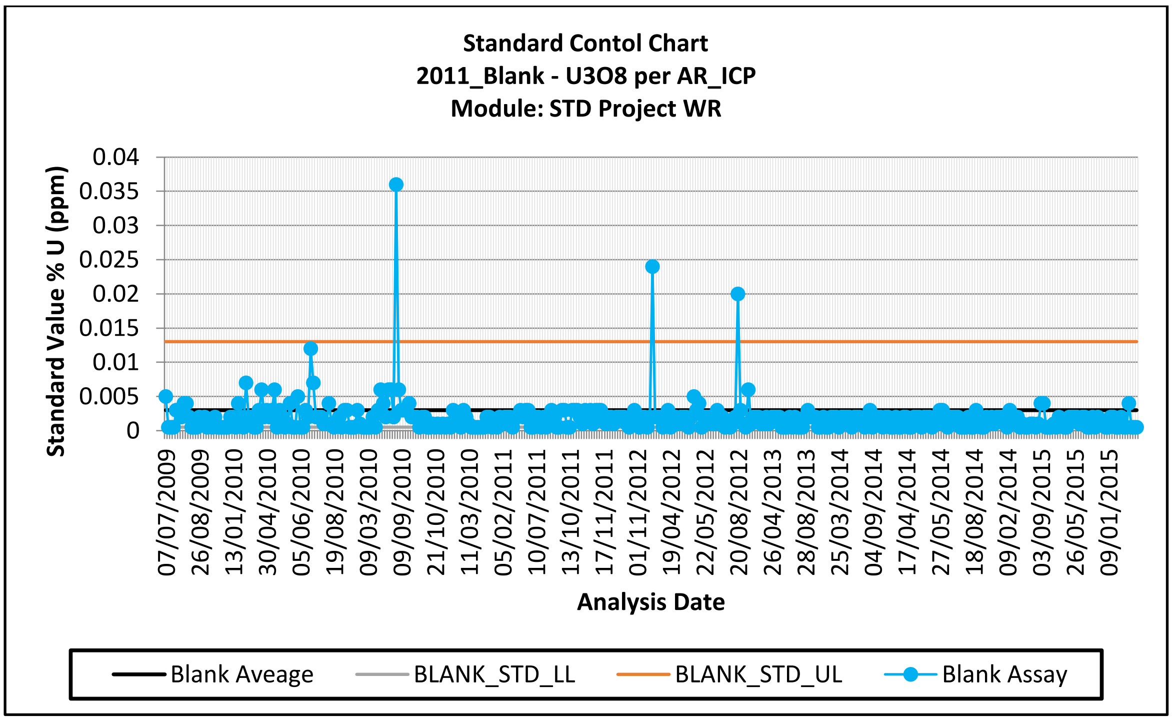



**Figure 11-7: Blank Sample Analyses Results**

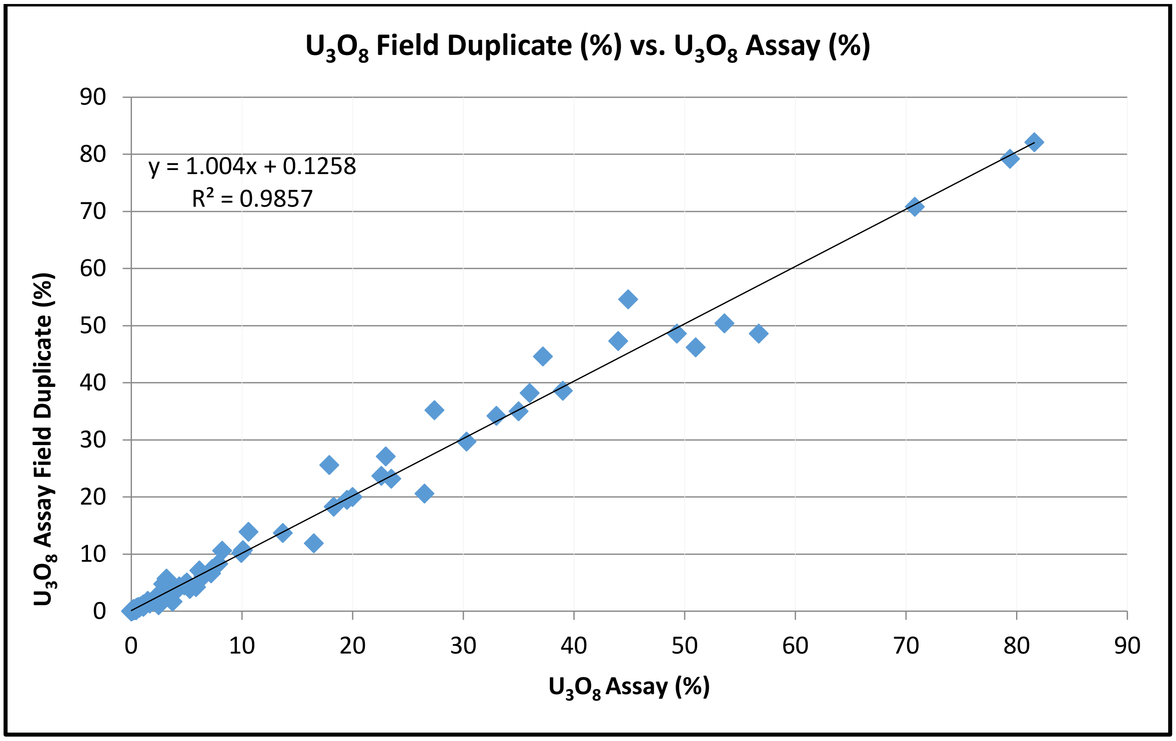



**Figure 11-8: Field Duplicate Analyses**

## 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 will be 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).

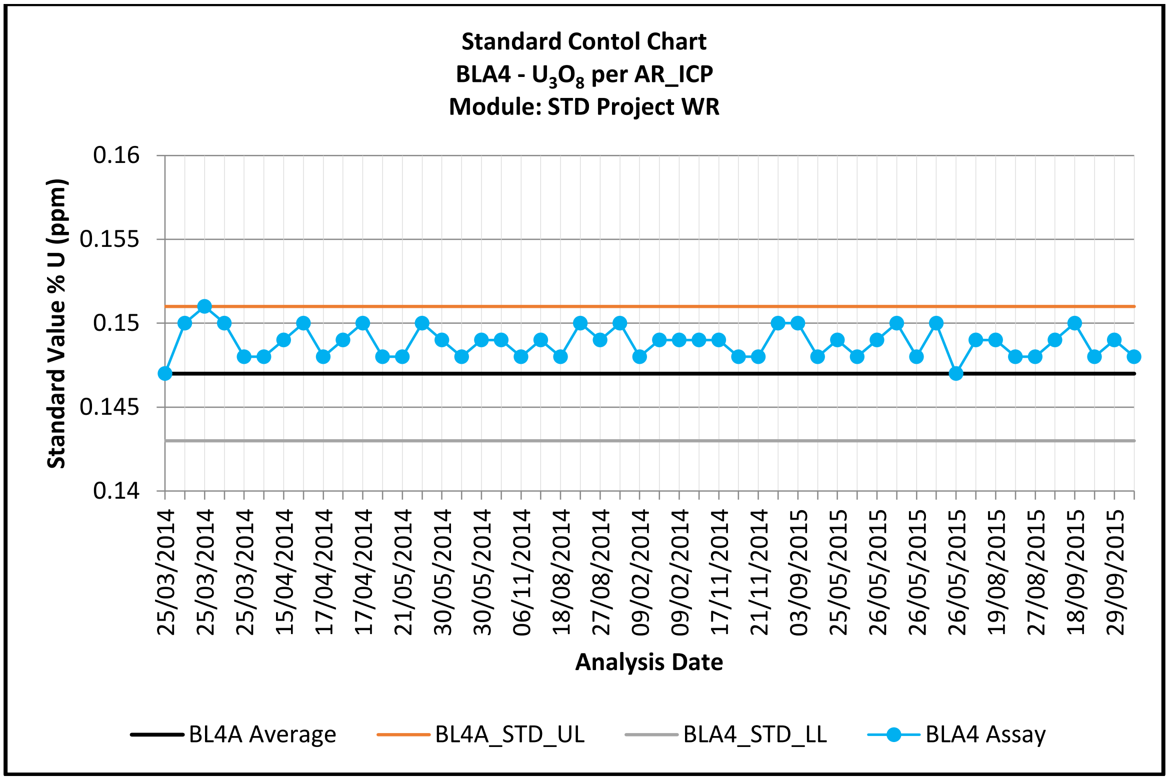



**Figure 11-9: BLA4 Analyses**

**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 the SRC's Delayed Neutron Counting (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 85 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.

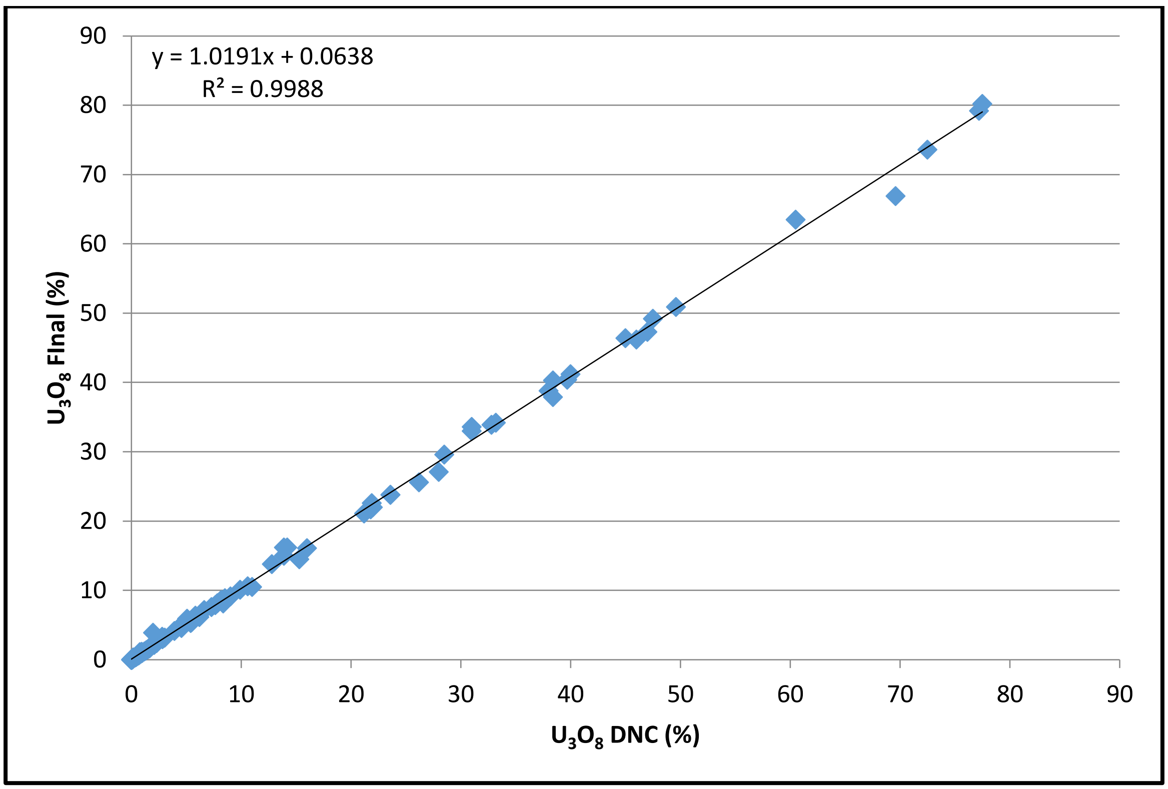



**Figure 11-10: $U_3O_8$ DNC Versus ICP-OES Assay Values**

### 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

*This entire report section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada," Roscoe Postle Associates, November 25, 2015.*

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 two site visits.

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, Sample Preparation, Analyses and Security. 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 Gryphon mineral resource estimate. 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 3,233 laboratory records. RPA verified approximately 1,010 records representing 30% of the data for uranium values against 39 different laboratory certificates. No discrepancies were found.

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.

## 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.
- 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.

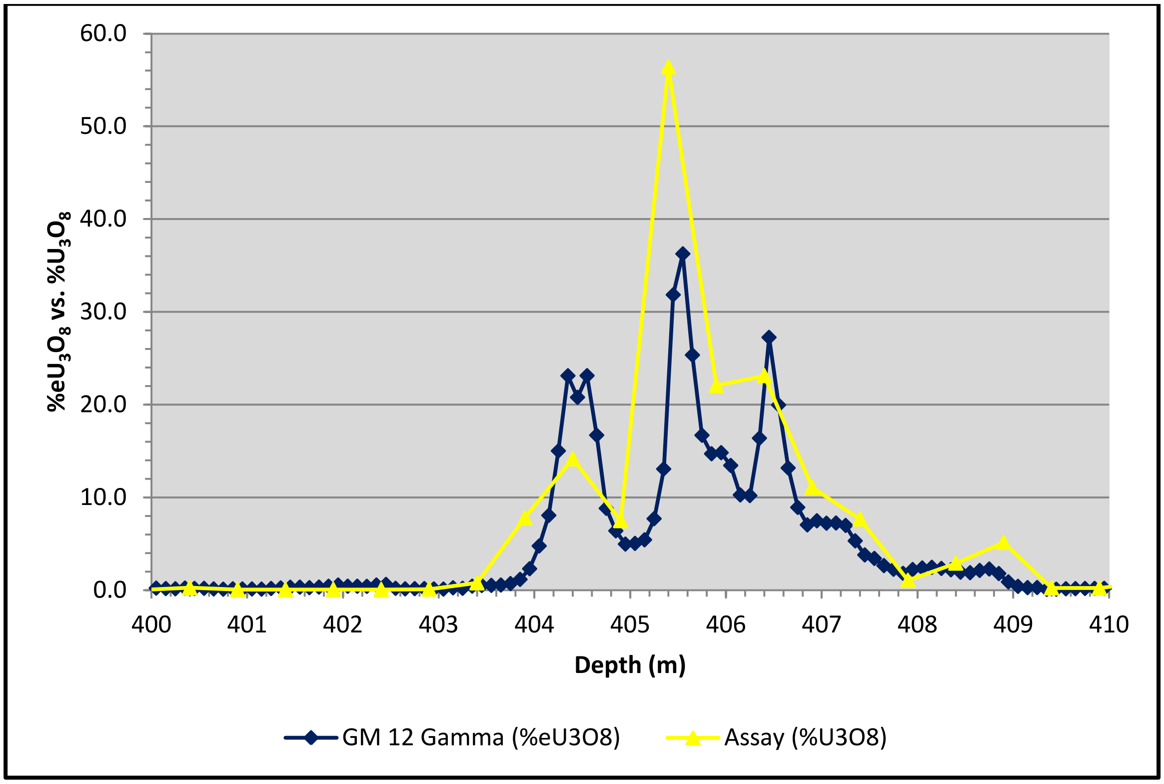



**Figure 12-1: WR-318 Radiometric versus Assay % $U_3O_8$ Values**

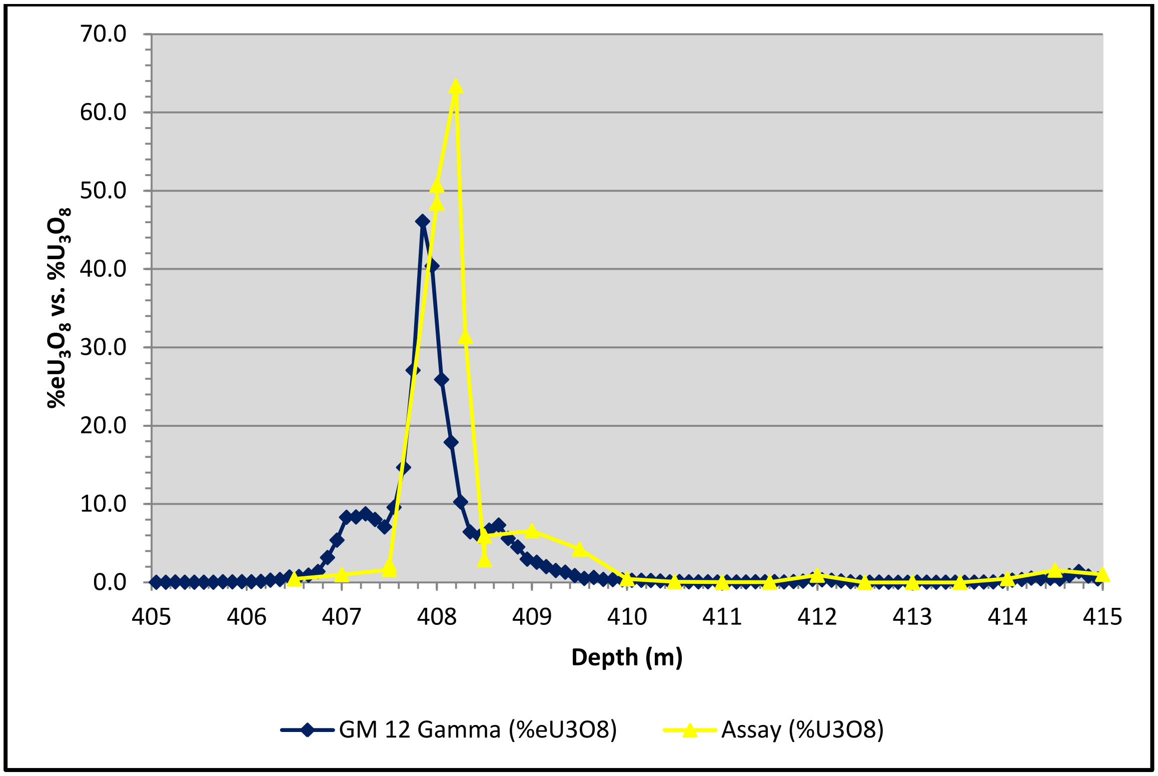



**Figure 12-2: WR-334 Radiometric versus Assay % $U_3O_8$ Values**

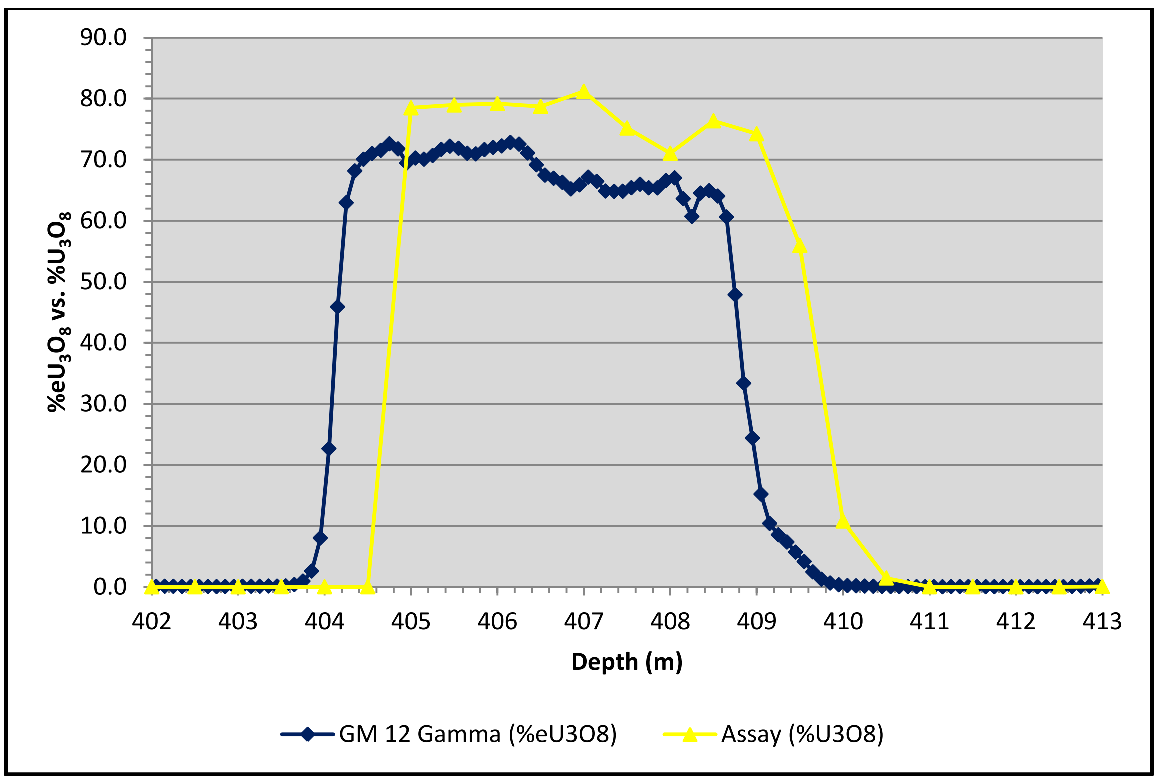



**Figure 12-3: WR-273 Radiometric versus Assay % $U_3O_8$ Values**

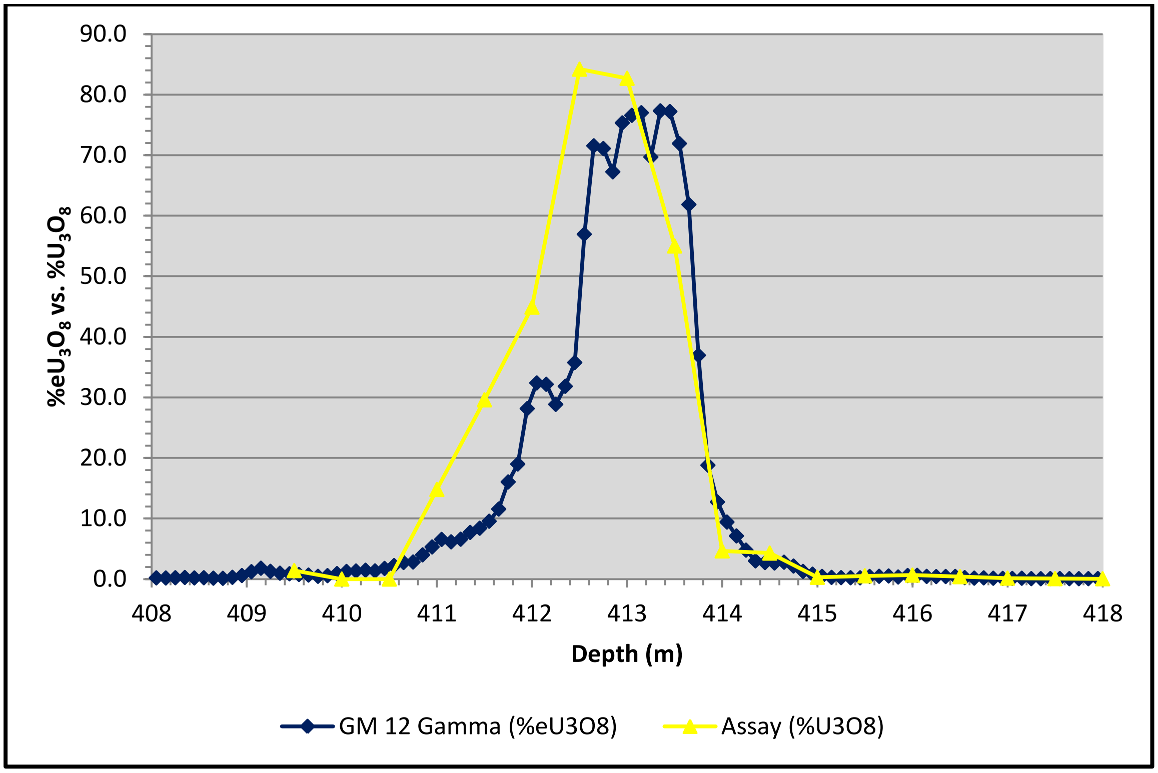



**Figure 12-4: WR-435 Radiometric versus Assay % $U_3O_8$ Values**

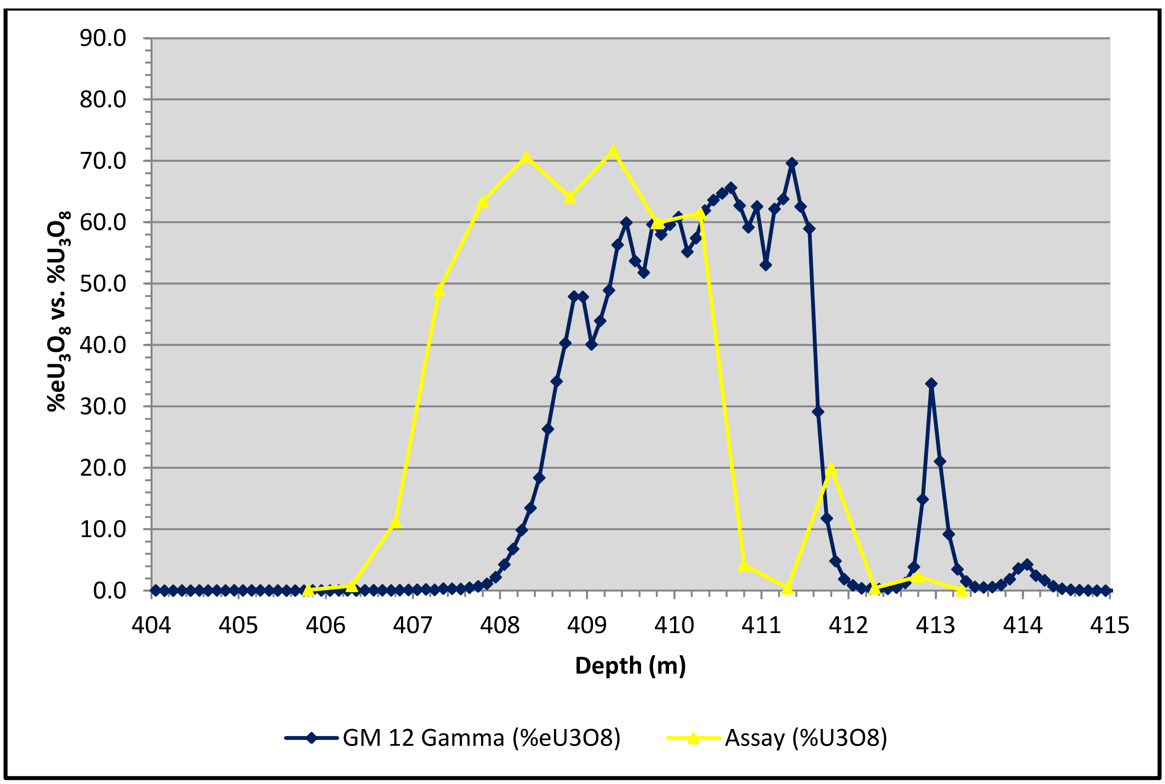



**Figure 12-5: WR-548 Radiometric versus Assay % $U_3O_8$ Values**

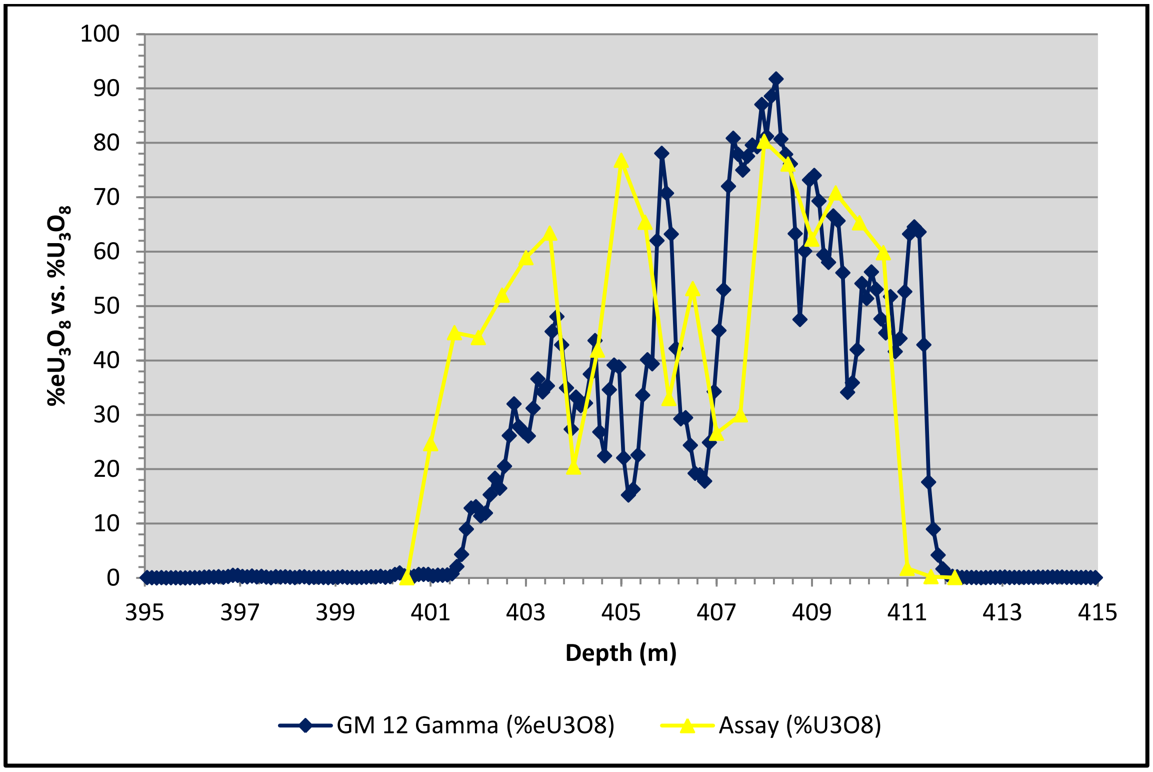



**Figure 12-6: WR-525 Radiometric versus Assay % $U_3O_8$ Values**

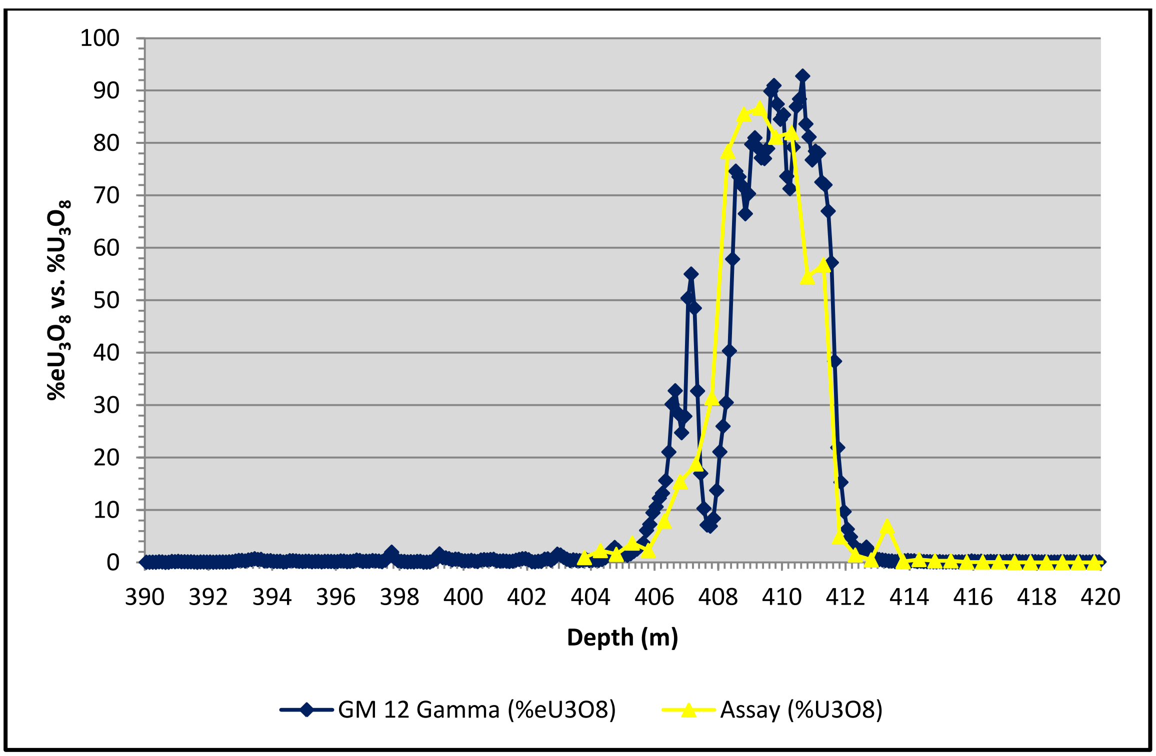



**Figure 12-7: WR-401 Radiometric versus Assay % $U_3O_8$ Values**

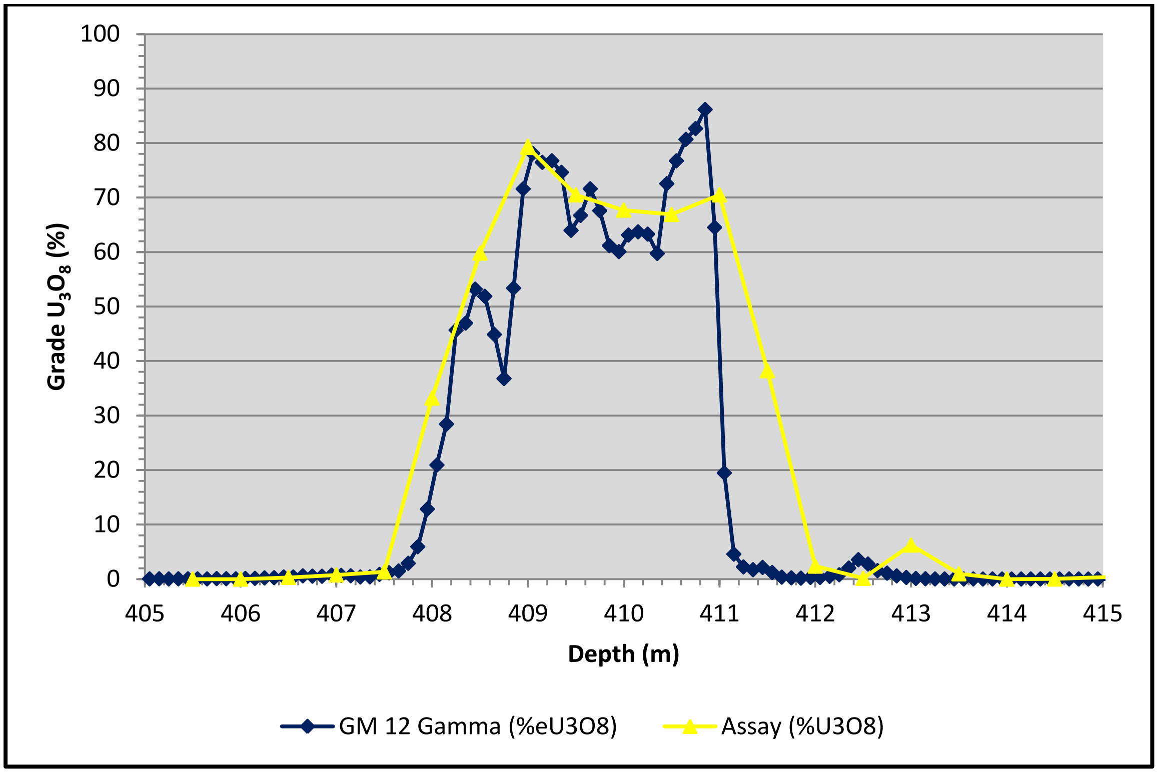



**Figure 12-8: WR-306 Radiometric versus Assay % $U_3O_8$ Values**

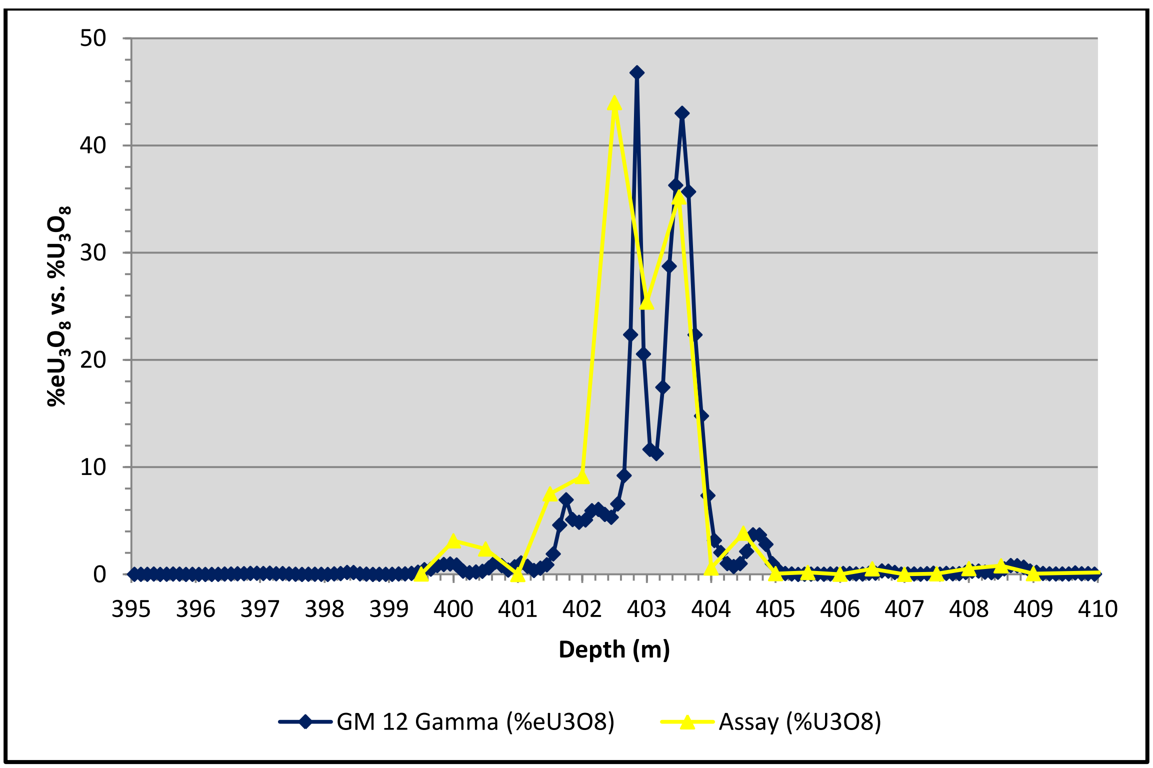



**Figure 12-9: WR-539 Radiometric versus Assay % $U_3O_8$ Values**

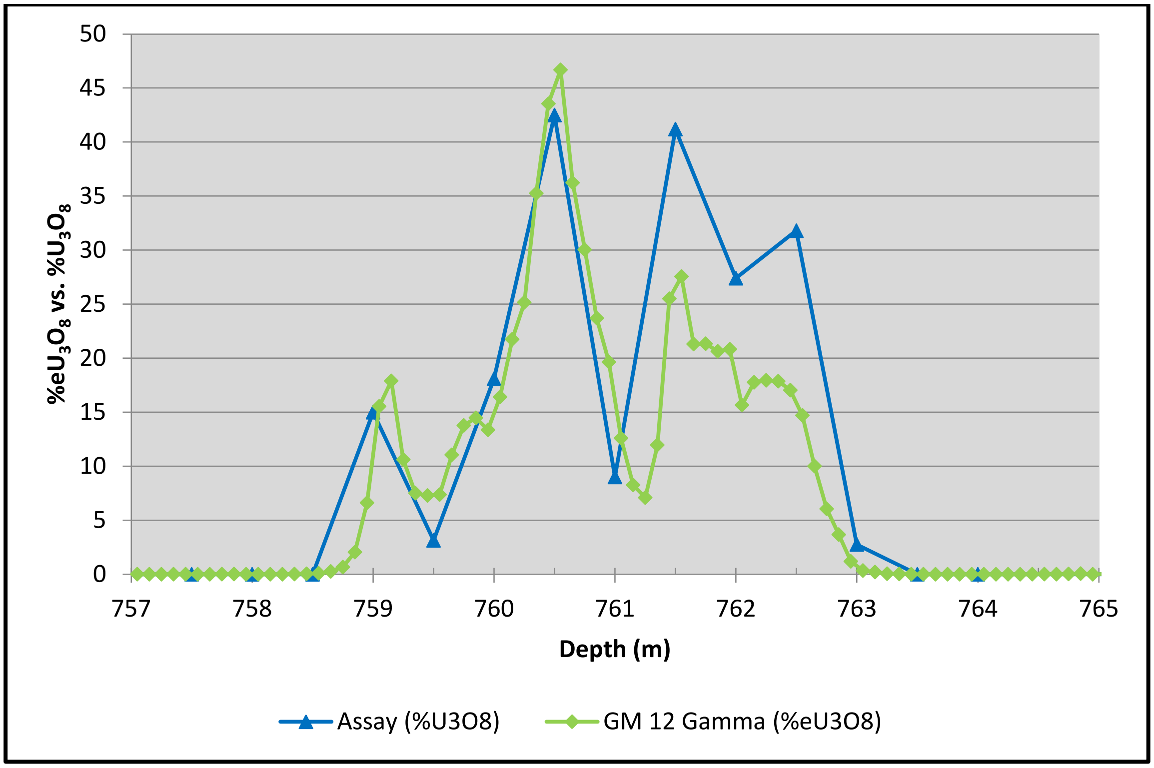



**Figure 12-10: WR-560 Radiometric versus Assay % $U_3O_8$ Values**

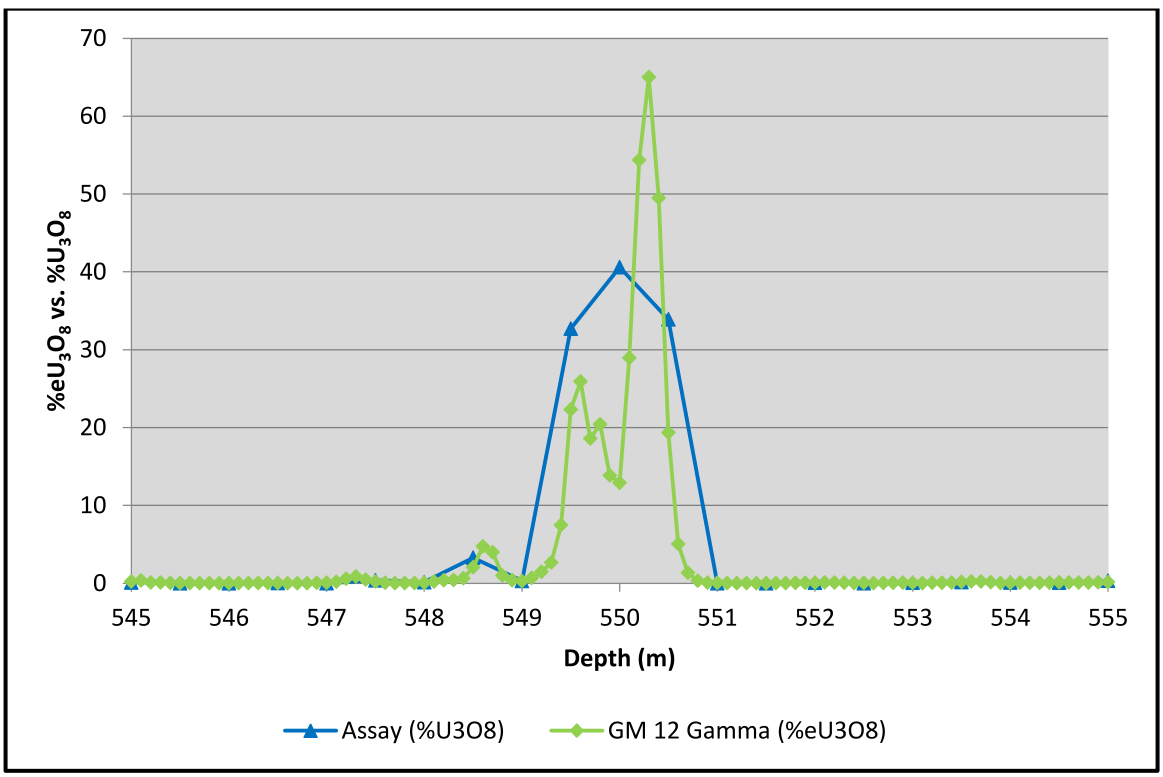



**Figure 12-11: WR-573D1 Radiometric versus Assay % $U_3O_8$ Values**

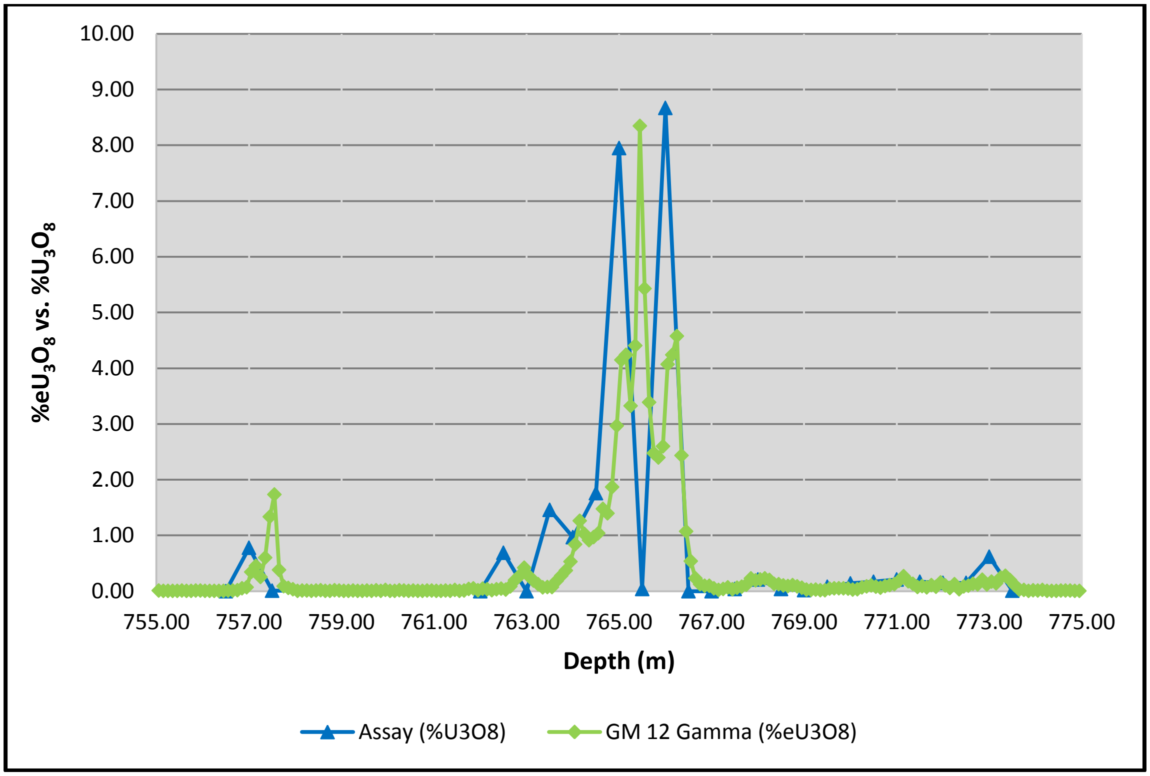



**Figure 12-12: WR-582 Radiometric versus Assay % $U_3O_8$ Values**

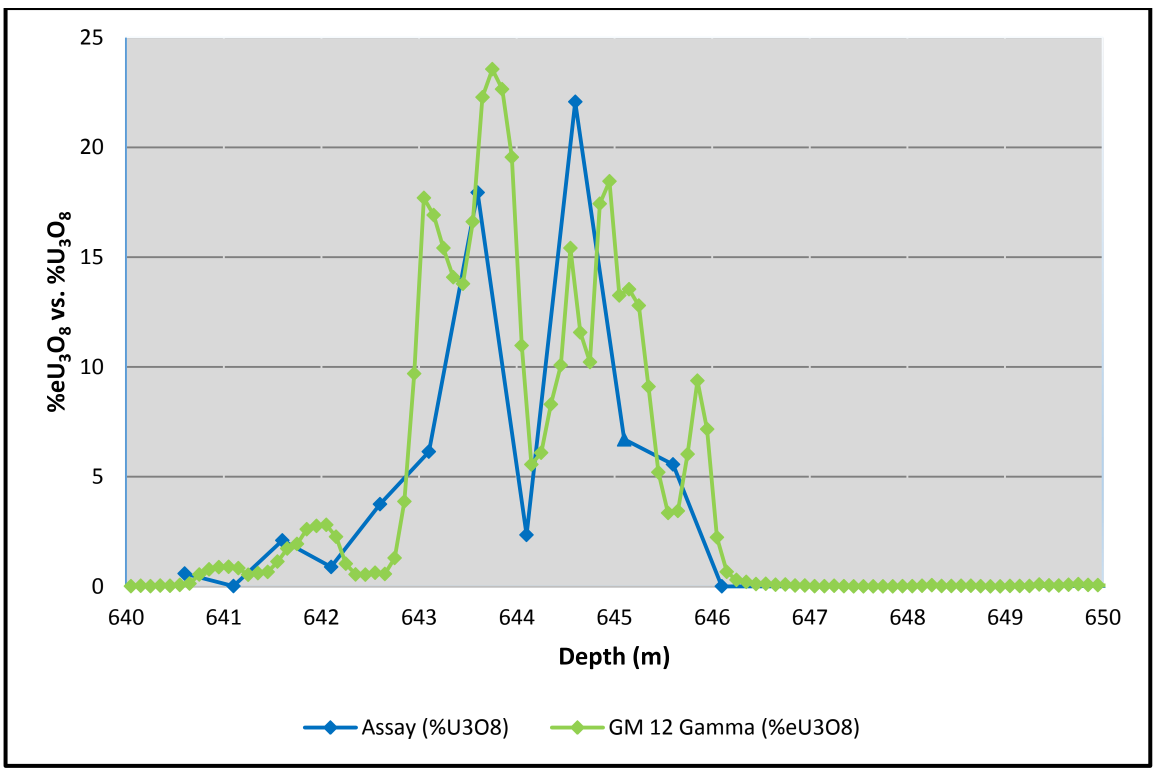



**Figure 12-13: WR-584B Radiometric versus Assay % $U_3O_8$ Values**

# 13 Mineral Processing and Metallurgical Testing

*This entire report section has been reproduced from "Denison Mines Limited, Wheeler River Preliminary Economic Assessment, Process Aspects,", Amec Foster Wheeler Americas Limited, January 21, 2016, an internal company supporting document commissioned by Denison.*

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 processing at a regional acid leach mill. The results are used to support process design criteria suitable for the Wheeler River site and the McClean Lake mill.

## 13.1 Phoenix Deposit Metallurgical Testing

In October 2014, the Saskatchewan Research Council (SRC) completed a preliminary testing program on the Phoenix uranium deposit for Denison Mines Corp., under guidance from Amec Foster Wheeler. The objectives of the tests were to determine the preliminary leaching process, leach residue settling, solvent extraction (SX) efficiency and raffinate composition, and purity of yellow cake. Mineralogy analysis using QEMSCAN was also performed. The overall test conditions emulated a regional acid leach mill flowsheet.

### 13.1.1 Sample Preparation

The SRC mineral processing group received 28 individual 0.5 m interval drill core assay coarse reject samples, from 12 different drill holes distributed across Zone A of the Phoenix deposit. A detailed list of the samples is shown in Table 13-1. The total weight of sample was 17.5 kg. The samples were combined and homogenized.

The homogenized sample major and minor components are shown in Table 13-2. The sample contained 19.7% $U_3O_8$. As indicated in Section 7.6, typical Phoenix deposit mineralization has arsenic concentration of approximately 300 ppm, similar to other basement-hosted deposits in the Athabasca Basin.

**Table 13-1: Feed Sample Preparation**

| Hole # | Year | % $U_3O_8$ |
|---|---|---|
| WR-268 | 2009 | 24.2 |
| WR-273 | 2009 | 71.9 |
| WR-273 | 2009 | 71.7 |
| WR-273 | 2009 | 57.7 |
| WR-299 | 2010 | 3.31 |
| WR-299 | 2010 | 2.01 |
| WR-302 | 2010 | 0.791 |
| WR-302 | 2010 | 1.3 |
| WR-302 | 2010 | 1.56 |
| WR-302 | 2010 | 0.534 |
| WR-306 | 2010 | 33.2 |
| WR-306 | 2010 | 38.8 |
| WR-318 | 2010 | 8.38 |
| WR-318 | 2010 | 7.37 |
| WR-318 | 2010 | 7.48 |
| WR-342 | 2010 | 6.85 |
| WR-342 | 2010 | 1.59 |
| WR-376 | 2011 | 3.95 |
| WR-376 | 2011 | 4.2 |
| WR-405 | 2011 | 9.65 |
| WR-405 | 2011 | 16.7 |
| WR-409 | 2011 | 1.16 |
| WR-409 | 2011 | 1.3 |
| WR-419 | 2011 | 7.66 |
| WR-419 | 2011 | 16.3 |
| WR-535 | 2013 | 58.6 |
| WR-535 | 2013 | 13.2 |
| WR-535 | 2013 | 37.9 |

**Table 13-2: Phoenix Deposit Composite Test Sample Assay**

| Major components ( % ) | | | | | | |
|---|---|---|---|---|---|---|
| $Al_2O_3$ | CaO | $Fe_2O_3$ | $K_2O$ | MgO | $TiO_2$ | $U_3O_8$ |
| 16.5 | 1.4 | 12.5 | 1.9 | 3.7 | 0.74 | 19.7 |
| **Minor components ( ppm )** | | | | | | |
| Mo | Ni | Pb | Co | Cu | V | Zn |
| 557 | 630 | 18500 | 251 | 7870 | 1630 | 4900 |

The homogenized sample was crushed to -2.0 mm and classified into three size range fractions for QEMSCAN analysis. The major uranium bearing mineral is uraninite. The grain size analysis for uraninite indicates over 99% of the uraninite grains are less than 360 μm diameter.

The liberation of uraninite in each size fraction was determined. Exposed uraninite is present in large quantities in all size fractions. Uraninite locked in other minerals is ≤5% of the total uraninite in all size fractions. These liberation characteristics suggest that uraninite should be readily accessible by lixiviant.

Most of the uraninite is associated with clay minerals (predominantly illite). Significant uraninite is associated with sulphides and iron oxides. Lesser amounts of uraninite are associated with carbonates and quartz. The uraninite mineral association indicates that acid leaching should be effective for leaching this material.

### 13.1.2 Leaching Tests

Three preliminary leaching tests were performed with the homogenized sample, wet ground to $P_{100}$=300 µm. Other standard test conditions include ORP >450 mV, 50% pulp density, 1 atm pressure, and 50°C for a minimum of eight hours.

In the first two tests, the initial free acid was 80 g/L, decreasing to 23 g/L by the end of eight hours. Test 1 used sodium chlorate as the oxidant, whereas Test 2 used hydrogen peroxide. The extraction rates were viewed as unacceptably slow, with 95.7% and 92.6% recovery respectively after eight hours.

For Test 3, initial free acid was adjusted to 100 g/L, decreasing to 31 g/L at the end of 12 hours. One g/L of ferric ion ($Fe^{3+}$) in the form of ferric sulphate was also added. Consequently, recovery increased to 98.1%, 98.9% and 99.4% after 4, 8 and 12 hours respectively.

Leaching Test 4 used a finer feed grind to $P_{100}$=212 µm, with other test conditions the same as Test 3. Recovery was 99.3% after 12 hours.

Leaching Test 5 used optimized test conditions based on the first four tests, with feed grind size $P_{100}$=300 µm. Initial free acid was adjusted to 100 g/L, decreasing to 25 g/L at the end of 12 hours. Hydrogen peroxide was used as oxidant, with 1 g/L of ferric ion as ferric sulphate added. Other test conditions were the same as above. Recovery was improved to 98.6%, 99.4% and 99.5% after 4, 8, and 12 hours, respectively.

### 13.1.3 Settling Tests

The settling test on each leach residue slurry was performed immediately after the leaching was completed. The leach residue slurry was diluted from 50% solids to 6% solids with acidic water at pH=2, to simulate the high degree of CCD washing expected for full scale operation. Dilution of the pregnant leaching solution feeding SX to approximately 10 g/L $U_3O_8$ is anticipated.

Magnafloc 351 flocculant was dosed at 180 g/t solids to obtain a satisfactory initial settling rate. Settling effectively stopped after five hours. The settled density was 27% solids.

### 13.1.4 Solvent Extraction Tests

The pregnant leaching solution was the blended supernatant from the settling tests. Four stages of solvent extraction (SX) were performed at the organic/aqueous ratio of 1/1. The organic solution was made with 6 volume % Armeen 380, 3 volume % isodecanol and 91 volume % diluent.

The $U_3O_8$ was almost completely extracted with 99.99% transferred to the organic phase. Most of the impurity metals (Fe, Ca, Na, Mn, V, Zr, Cu, Co, Ni, As, and Zn) were left in the raffinate.

Solvent extraction is effective to selectively extract and purify uranium. No abnormal challenges are expected for effluent treatment based on the raffinate composition.

Stripping of the loaded organic was performed using a 400 g/L $H_2SO_4$ solution. The pregnant strip solution was used as feed to the yellowcake production test.

### 13.1.5 Yellowcake Precipitation and $U_3O_8$ Production

The pregnant strip solution was diluted and neutralized to pH 2.8 using 10% lime slurry. The gypsum produced was removed by filtration. Uranium was then precipitated from the filtrate as uranyl peroxide by the addition of hydrogen peroxide, using magnesium oxide to maintain pH 3.8 to 4.0. The resulting yellowcake sample was analyzed for major elements, as shown in Table13-3 below. A high purity yellowcake product was produced, meeting all specifications on ASTM C967-13, “Standard Specifications for Uranium Ore Concentrate.”

**Table13-3: Phoenix Zone $U_3O_8$ Product Assay**

| Specifications | ASTM C967-13 (Mass%, Uranium Basis) | | Denison Mines, Phoenix Deposit (Mass%, Uranium Basis) |
|---|---|---|---|
| **Component** | **Limit without Penalty** | **Limit without Rejection** | **$U_3O_8$ Product** |
| Uranium (U) | N/A | 65% min. | 72.4% |
| Calcium (Ca) | 0.05% | 1.00% | <0.01% |
| Chromium (Cr) | N/A | N/A | <0.0001% |
| Iron (Fe) | 0.15% | 1.00% | 0.19% |
| Lead (Pb) | N/A | N/A | <0.0001% |
| Magnesium (Mg) | 0.02% | 0.50% | <0.006% |
| Molybdenum (Mo) | 0.10% | 0.30% | <0.0001% |
| Phosphorus ($PO_4$) | 0.10% | 0.70% | <0.01% |
| Potassium (K) | 0.20% | 3.00% | <0.01% |
| Silver (Ag) | N/A | N/A | <0.0002% |
| Sodium (Na) | 1.00% | 7.50% | <0.01% |
| Thorium | 0.10% | 2.50% | <0.0001% |
| Titanium | 0.01% | 0.05% | <0.01% |
| Vanadium (V) | 0.06% | 0.30% | 0.038% |
| Zirconium (Zr) | 0.01% | 0.10% | <0.0001% |

### 13.1.6 Phoenix Deposit Process Design Criteria

The leach feed grade of the Phoenix testwork sample was 19.7%, whereas the run-of-mine grade is expected to be 12.1%. While it is normal to see a difference such as this between core sampling and estimated mine production, it means that leach recoveries representative of the run-of-mine grade need to be adjusted compared to the test sample results cited above.

The test results show 99.4% uranium leach recovery (0.6% loss) with a minimum retention time of 12 hours. For the 19.7% $U_3O_8$ sample feed grade, this corresponds to approximately 0.12% $U_3O_8$ in leach residue. To adjust to an expected 12.1% run-of-mine grade, this same 0.12% residue assay was used to calculate a loss of 1.0%.

Circuit design uranium recovery losses for Phoenix deposit milling are:

- Leaching – 1.0%
- Solid/liquid separation (CCD) soluble loss – 0.7%
- Downstream (SX, precipitation, other) – 0.2%.

Total losses are anticipated to be 1.9%, yielding an overall uranium mill recovery of 98.1%.

For reference, the 2012 Cigar Lake NI 43-101 technical report’s mine production schedule was based on the McClean Lake mill having an overall uranium recovery of 98.5% with an average 18.3% U3O8 feed grade. With a potential increase in leaching residence time beyond the 12 hours tested to date, there is an opportunity to reduce Phoenix leach recovery loss.

## 13.2 Gryphon Deposit Metallurgical Testing

In October 2015, the Saskatchewan Research Council (SRC) completed a preliminary testing program on the Gryphon uranium deposit for Denison Mines Corp., under guidance from Amec Foster Wheeler. The objectives of the tests were to determine the preliminary leaching process, leach residue settling, solvent extraction (SX) efficiency and raffinate composition, and purity of yellow cake. Mineralogy analysis using QEMSCAN was also performed. The overall test conditions emulated the McClean Lake mill flowsheet.

### 13.2.1 Sample Preparation

The SRC mineral processing group received 26 individual 0.5 m interval drill core assay coarse reject samples, from 10 different drill holes distributed across the deposit. A detailed list of the samples is shown in Table 13-4. The total weight of sample was approximately 22.8 kg. The samples were combined, wet ground to $P_{100}$=300 µm and homogenized. The composite was split to prepare the other two grind size samples ($P_{100}$=106 µm and $P_{100}$=212 µm), the QEMSCAN sample and the sample for assay.

**Table 13-4: Feed Sample Preparation**

| Hole # | Year | % $U_3O_8$ |
|---|---|---|
| WR-584B | 2015 | 19.20 |
| WR-584B | 2015 | 0.63 |
| WR-574 | 2014 | 5.92 |
| WR-574 | 2014 | 0.39 |
| WR-574 | 2014 | 0.17 |
| WR-569A | 2014 | 0.25 |
| WR-569A | 2014 | 0.28 |
| WR-569A | 2014 | 0.19 |
| WR-560 | 2014 | 18.10 |
| WR-560 | 2014 | 0.15 |
| WR-560 | 2014 | 0.16 |
| WR-556 | 2014 | 5.66 |
| WR-556 | 2014 | 0.15 |
| WR-556 | 2014 | 0.15 |
| WR-564 | 2014 | 14.00 |
| WR-564 | 2014 | 0.16 |
| WR-564 | 2014 | 0.23 |
| WR-571 | 2014 | 4.95 |
| WR-571 | 2014 | 0.55 |
| WR-571 | 2014 | 0.13 |
| WR-583 | 2015 | 0.53 |
| WR-583 | 2015 | 0.15 |
| WR-573D1 | 2014 | 3.29 |
| WR-573D1 | 2014 | 0.14 |
| WR-572 | 2014 | 2.31 |
| WR-572 | 2014 | 0.12 |

The major components as well as some minor components are shown in Table 13-5. The sample contains 3.36% $U_3O_8$. As indicated in Section 7.6, typical Gryphon deposit mineralization has arsenic concentration of approximately 30 ppm, similar to other basement-hosted deposits in the Athabasca Basin.

**Table 13-5: Gryphon Deposit Composite Test Sample Assay**

| Major Components ( % ) | | | | | | |
|---|---|---|---|---|---|---|
| $Al_2O_3$ | CaO | $Fe_2O_3$ | $K_2O$ | MgO | $TiO_2$ | $U_3O_8$ |
| 25.0 | 0.32 | 1.15 | 2.97 | 3.55 | 1.00 | 3.36 |
| **Minor components ( ppm )** | | | | | | |
| Mo | Ni | Pb | Co | Cu | V | Zn |
| 1630 | 231 | 1790 | 81 | 77 | 1240 | 3 |

The modal mineralogy analysis of the sample is shown in Table 13-6. The major uranium bearing mineral is uraninite with brannerite a minor uranium bearing mineral. Uraninite was observed as both large grains and fine grained disseminated uraninite in illite. The mineralogy is typical for a clean (that is, low nickel and arsenic) Athabasca Basin deposit.

Mineral associations were determined by QEMSCAN analysis of the 2D surface area of grains identified. Most of the uraninite is associated with phyllosilicates/clays minerals, followed by uraninite-illite intergrowth, quartz, tourmaline (dravite) and complex intergrowths. The mineral associations of uraninite indicate that a grinding size of $P_{100}$=300 µm would be effective for leach extraction of uranium.

**Table 13-6: Modal Mineralogy of Gryphon Deposit Sample**

| Minerals | Average (%) |
|---|---|
| Illite | 42.40 |
| Tourmaline (dravite) | 19.93 |
| Quartz | 18.45 |
| Kaolinite | 10.13 |
| Uraninite | 4.14 |
| Fine grained disseminated uraninite in illite | 1.38 |
| Rutile/anatase | 1.83 |
| Galena | 0.71 |
| Fe oxides/hydroxides | 0.54 |
| Pyrite | 0.19 |
| Calcite | 0.10 |
| Brannerite | 0.09 |
| Zircon | 0.07 |
| Apatite | 0.05 |
| Ca-sulphate | 0.04 |
| Chalcopyrite | 0.01 |

### 13.2.2 Leaching Tests

Two sets of leach tests were performed: 1) Vary the grind size of samples to $P_{100}$=106, 212, and 300 µm, and 2) evaluate the effect of final free acid levels in conjunction with ferric sulphate addition.

The first three leaching tests were performed on the samples ground to $P_{100}$=300 (DT-1), 212 (DT-2), and 106 µm (DT-3) in a 1 L glass reactor. Hydrogen peroxide was used as the oxidant. Other testing conditions include ORP≥450 mV, 50% pulp density, 1 atm pressure, and 50°C. The tests were performed for 12 hours. Final free acid ranged from 35.8 to 38.1 g/L.

The last two leaching tests were performed with the targeted final free acid of 25 (DT-4) and 15 g/L (DT-5), respectively. The grind size was $P_{100}$=300 µm, the same as that used in the leaching test DT-1. Other conditions were the same as the first three tests, except for the addition of 1 g/L $Fe^{3+}$ in the form of ferric sulphate for both test DT-4 and DT-5.

The results of the five leaching tests are shown in Figure 13-1. Without $Fe^{3+}$ addition, 95.4% to 98.8% of uranium can be extracted in eight hours and 98.6% to 99.2% in 12 hours, depending on leaching conditions. The average acid consumption was 11.3 kg/t ore. With $Fe^{3+}$ addition of 1g/L, leaching kinetics were enhanced, with 98.4% and 98.8% uranium extraction in 4 and 8 hours, respectively.

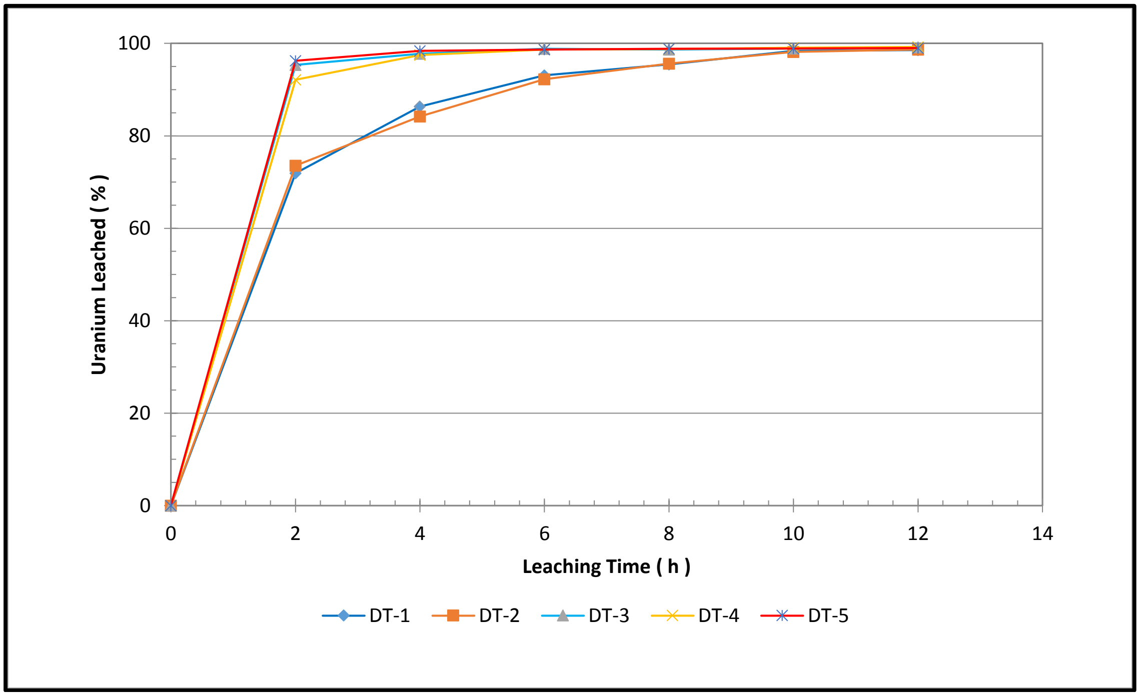



**Figure 13-1: Gryphon Zone Leaching Kinetics**

### 13.2.3 Settling Tests

The settling test of each leaching slurry residue was performed immediately after the leaching was completed. The leaching slurry was diluted from 50% solids to 25% solids with acidic deionized water at pH=2, to simulate the operation of CCD washing.

Settling was assisted with the addition of Magnafloc 351, a non-ionic polyacrylamide flocculant. The dosage of Magnafloc 351 was 180 g/t.

Table 13-7 shows the slurry density at the end of settling, which was terminated at 90 to 92 hours for test DT-1, DT-2 and DT-3, and 12 hours for test DT-4 and DT-5.

**Table 13-7: Gryphon Residue Settling**

| Test | Slurry Density (%) |
|---|---|
| DT-1 | 38.2 |
| DT-2 | 38.7 |
| DT-3 | 37.1 |
| DT-4 | 44.8 |
| DT-5 | 43.8 |
| **Average** | **40.5** |

Densities of settled leach residue ranged from 37.1% to 44.8% solids, with highest density achieved under conditions of grinding size of $P_{100}$=300 µm and low residual acid concentration in leaching.

### 13.2.4 Solvent Extraction Tests

The pregnant leaching solution was the blended supernatant from test DT-1 to DT-5 after settling tests. Four stages of solvent extraction (SX) were performed at the organic/aqueous ratio of 1/1. The organic solution was 6 volume % Armeen 380, 3 volume % isodecanol and 91 volume % diluent (CALUMET 400-500).

The $U_3O_8$ was almost completely extracted with 99.999% transferred to the organic phase. Most of the impurity metals (Fe, Ca, Na, Mn, V, Zr, Cu, Co, Ni, As, and Zn) were left in the raffinate. Solvent extraction is effective to selectively extract and purify uranium. No abnormal challenges are expected for effluent treatment based on the raffinate composition.

### 13.2.5 Yellowcake Precipitation and $U_3O_8$ Production

The pH of the pregnant stripping solution was adjusted using ammonium hydroxide solution to pH 7.0 to 7.5 to precipitate the yellow cake as ammonium diuranate (ADU). The produced yellow cake was rinsed with deionized water and was calcined at 700°C for two hours to produce the $U_3O_8$ sample. The assay of the $U_3O_8$ sample is shown in Table 13-8. A high purity $U_3O_8$ product was produced, meeting all specifications on ASTM C967-13, “Standard Specifications for Uranium Ore Concentrate.”

**Table 13-8: Gryphon Zone $U_3O_8$ Product Assay**

| Specifications | ASTM C967-13 (Mass%, Uranium Basis) | | Denison Mines, Gryphon Deposit (Mass%, Uranium Basis) |
|---|---|---|---|
| **Component** | **Limit without Penalty** | **Limit without Rejection** | **$U_3O_8$ Product** |
| Uranium (U) | N/A | 65% min. | 84.5% |
| Calcium (Ca) | 0.0005 | 0.0100 | 0.01% |
| Chromium (Cr) | N/A | N/A | 0.006% |
| Iron (Fe) | 0.0015 | 0.0100 | <0.008% |
| Lead (Pb) | N/A | N/A | 0.0006% |
| Magnesium (Mg) | 0.0002 | 0.0050 | <0.007% |
| Molybdenum (Mo) | 0.0010 | 0.0030 | 0.04% |
| Phosphorus (PO4) | 0.0010 | 0.0070 | 0.02% |
| Potassium (K) | 0.0020 | 0.0300 | <0.01% |
| Silver (Ag) | N/A | N/A | 0.00006% |
| Sodium (Na) | 0.0100 | 0.0750 | <0.009% |
| Thorium | 0.0010 | 0.0250 | 0.008% |
| Titanium | 0.0001 | 0.0005 | <0.007% |
| Vanadium (V) | 0.0006 | 0.0030 | <0.0001% |
| Zirconium (Zr) | 0.0001 | 0.0010 | <0.0001% |

## 13.2.6 Gryphon Deposit Process Design Criteria

The leach feed grade of the Gryphon testwork sample was 3.36%, whereas the run-of-mine grade is expected to be 1.90%. While it is normal to see a difference such as this between core sampling and estimated mine production, it means that leach recoveries representative of the run-of-mine grade need to be adjusted compared to the test sample results cited above.

The test results show 98.8% uranium leach recovery (1.2% loss) with a minimum retention time of eight hours. For the 3.36% $U_3O_8$ sample feed grade, this corresponds to approximately 0.04% $U_3O_8$ in leach residue. To adjust to an expected 1.90% run-of-mine grade, this same 0.04% residue assay was used to calculate a uranium loss of 2.1%.

Circuit design recovery losses for Gryphon deposit milling are:

- Leaching – 2.1%
- Solid/liquid separation (CCD) soluble loss – 0.7%
- Downstream (SX, precipitation, other) – 0.2%

Total losses are anticipated to be 3.0%, yielding an overall mill recovery of 97.0%. For reference, the 2012 Cigar Lake NI 43-101 technical report notes that historically the Rabbit Lake mill treating Eagle Point mine ore achieved a uranium recovery of approximately 97.0%. The historical feed grade from Eagle Point has been similar to that expected from Gryphon deposit.

The mill recovery for Gryphon is lower than for Phoenix, predominantly due to the much lower feed grade of Gryphon.

# 14 Mineral Resource Estimates

*This entire report section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada," Roscoe Postle Associates, November 25, 2015.*

RPA has estimated mineral resources for the Phoenix and Gryphon deposits based on results of several surface diamond drilling campaigns from 2008 to 2015. 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 3 km northwest of the Phoenix deposit.

Table 14-1 summarizes the mineral resource estimate, of which Denison's share is 60%. The effective date of the mineral resource estimate is September 25, 2015. The mineral resource estimate for Phoenix was reported in a previous NI 43-101 technical report (RPA, 2014) dated June 17, 2014 with and 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 – September 25, 2015**

| Category | Deposit | Tonnes | Grade (% $U_3O_8$) | Million lbs $U_3O_8$ |
|---|---|---|---|---|
| Indicated | Phoenix - Zone A | 147,200 | 19.81 | 64.3 |
| Indicated | Phoenix - Zone B | 19,200 | 13.94 | 5.9 |
| **Total Indicated** | | **166,400** | **19.14** | **70.2** |
| Inferred | Phoenix - Zone B | 5,500 | 3.30 | 0.4 |
| Inferred | Phoenix - Zone A Basement | 3,100 | 10.24 | 0.7 |
| Inferred | Gryphon Deposit | 834,000 | 2.31 | 43.0 |
| **Total Inferred** | | **842,600** | **2.37** | **44.1** |

Notes:
1. CIM definitions were followed for classification of mineral resources.
2. Mineral resources for Phoenix are reported above a cut-off grade of 0.8% $U_3O_8$, which is based on internal Denison studies and a price of US$50 per lb $U_3O_8$.
3. Mineral resources for Gryphon are reported above a cut-off grade of 0.2% $U_3O_8$, which is based on RPA assumptions and a price of US$50 per lb $U_3O_8$.
4. High grade composites are subjected to a high grade search restriction without capping at Phoenix.
5. High grade mineralization was capped at 30% $U_3O_8$ with no search restrictions at Gryphon.
6. Bulk density is derived from grade using a formula based on 196 measurements at Phoenix and 65 measurements at Gryphon.
7. Numbers may not add due to rounding.

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 2008 to 2015, which comprise 433 diamond drill holes totalling 222,154 m, of which 196 drill holes totalling 89,835 m have delineated the Phoenix deposit and 55 holes totalling 40,041 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.

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, Data Verification, 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 | 69 | 253 |
| Survey | 857 | 2,879 |
| Stratigraphy | 934 | 2,632 |
| Assay Values | 1,019 | 2,111 |
| Radiometric Values (% $eU_3O_8$) | 118,048 | 166,287 |
| Block Model 1m Composites in Wireframes | 419 | 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.

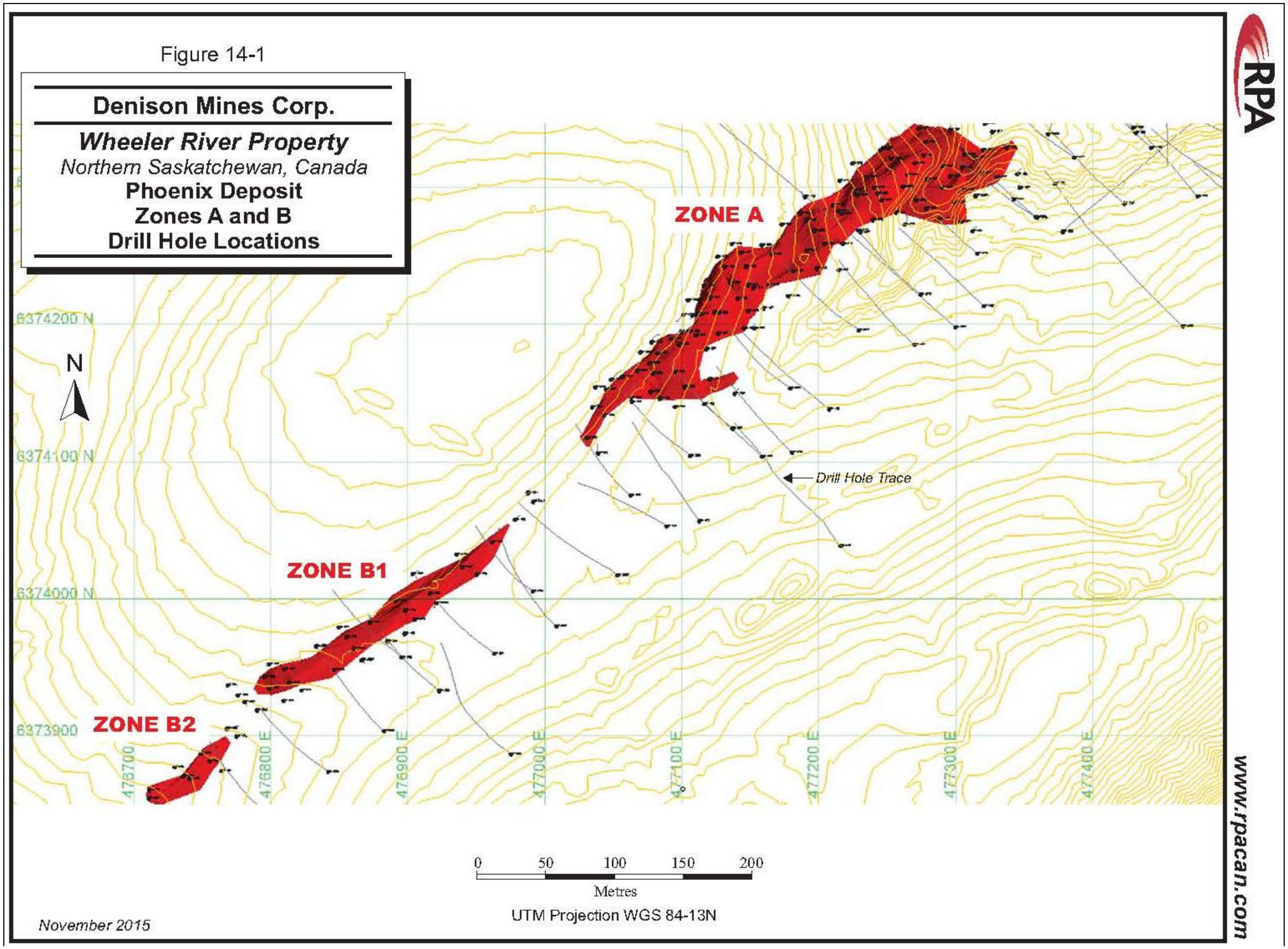



**Figure 14-1: Phoenix Deposit Zones A and B Drill Hole Locations**

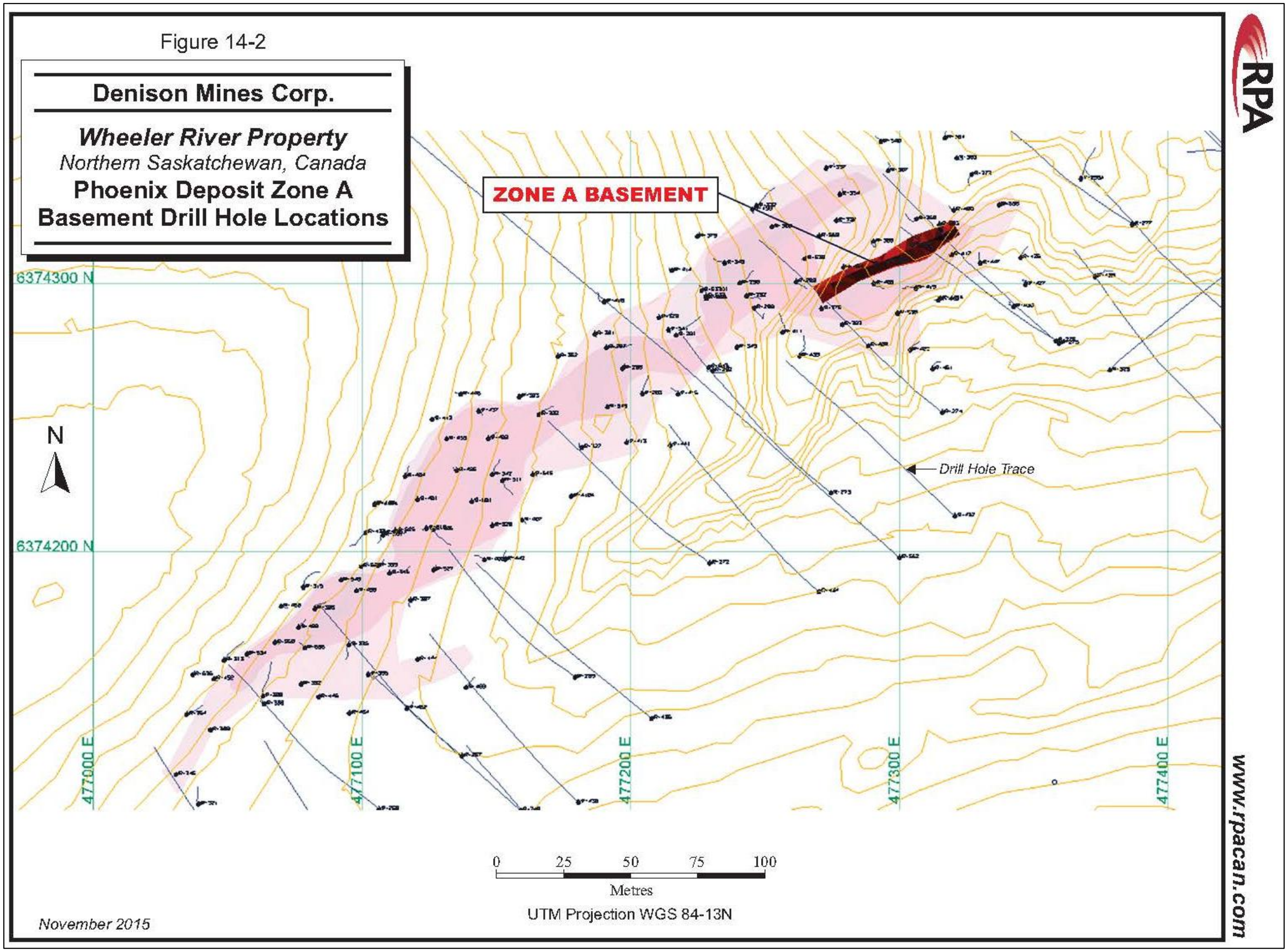



**Figure 14-2: Phoenix Deposit Zone A Basement Drill Hole Locations**

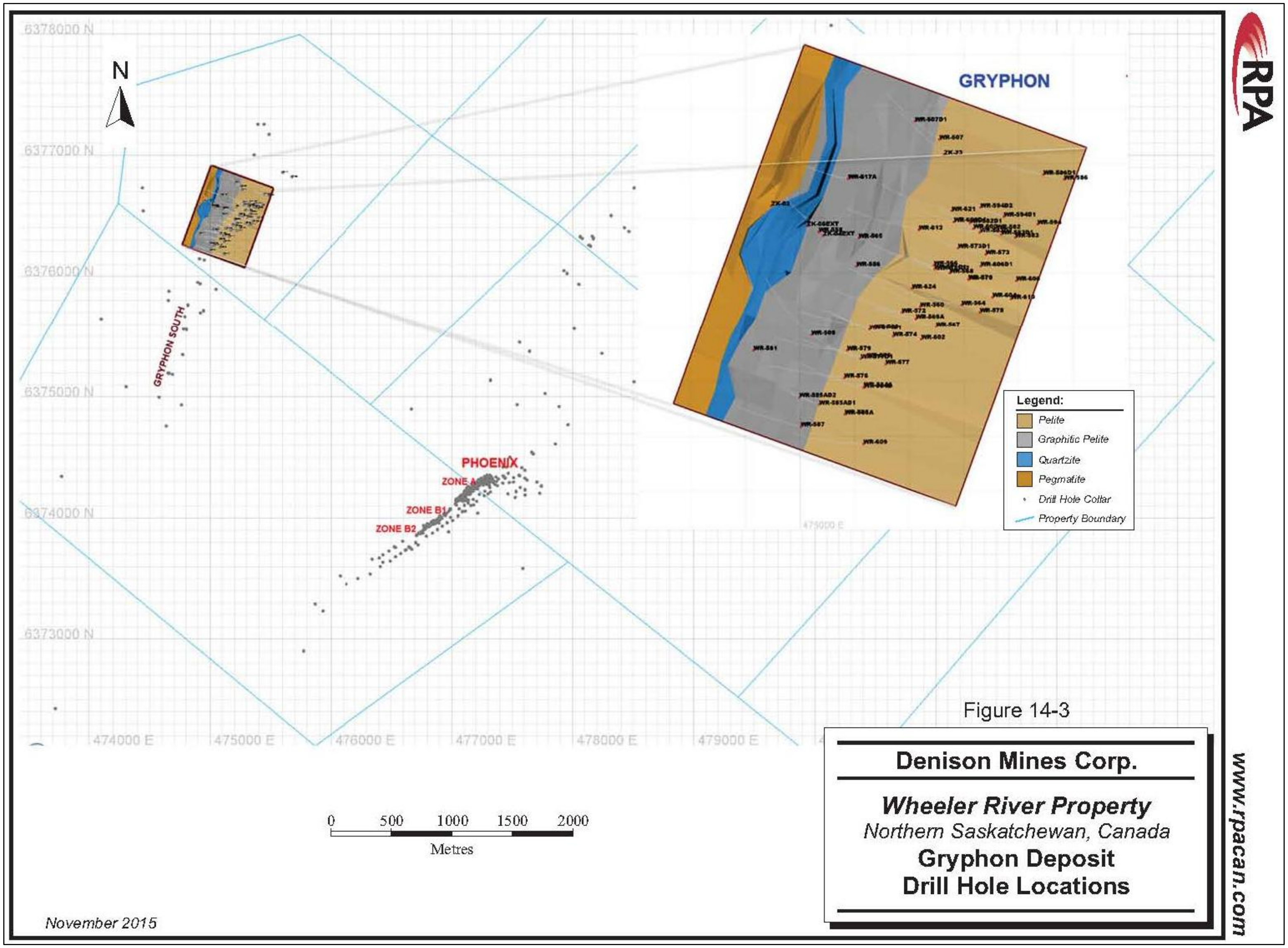



**Figure 14-3: Gryphon Deposit Drill Hole Locations**

# 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 pelite 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 pelite 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. Figure 14-4 to 14-6 are cross-sections of Zone A showing drill holes with 1-metre 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.

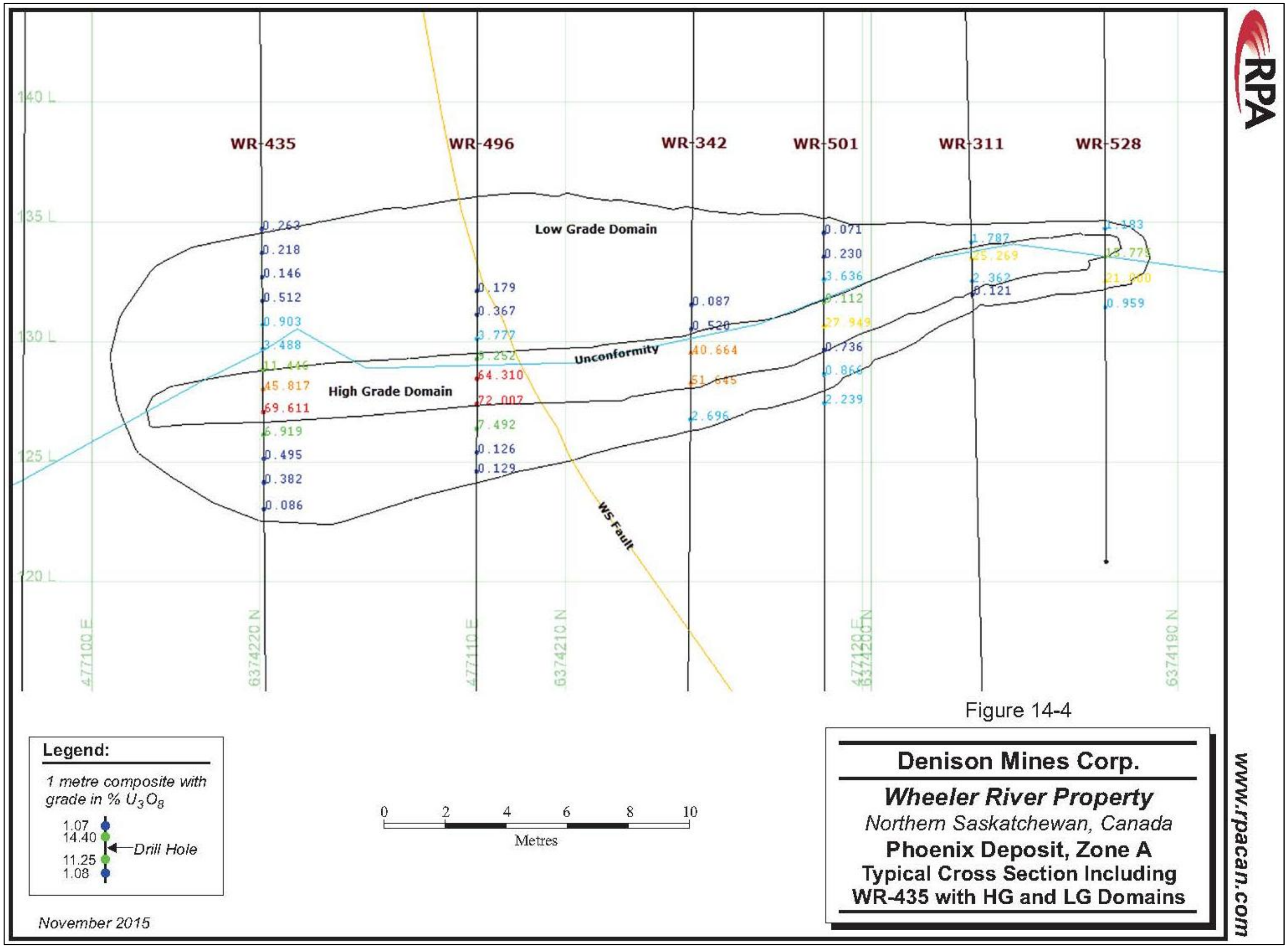



**Figure 14-4: Phoenix Deposit Zone A Typical Cross-Section with HG and LG Domains**

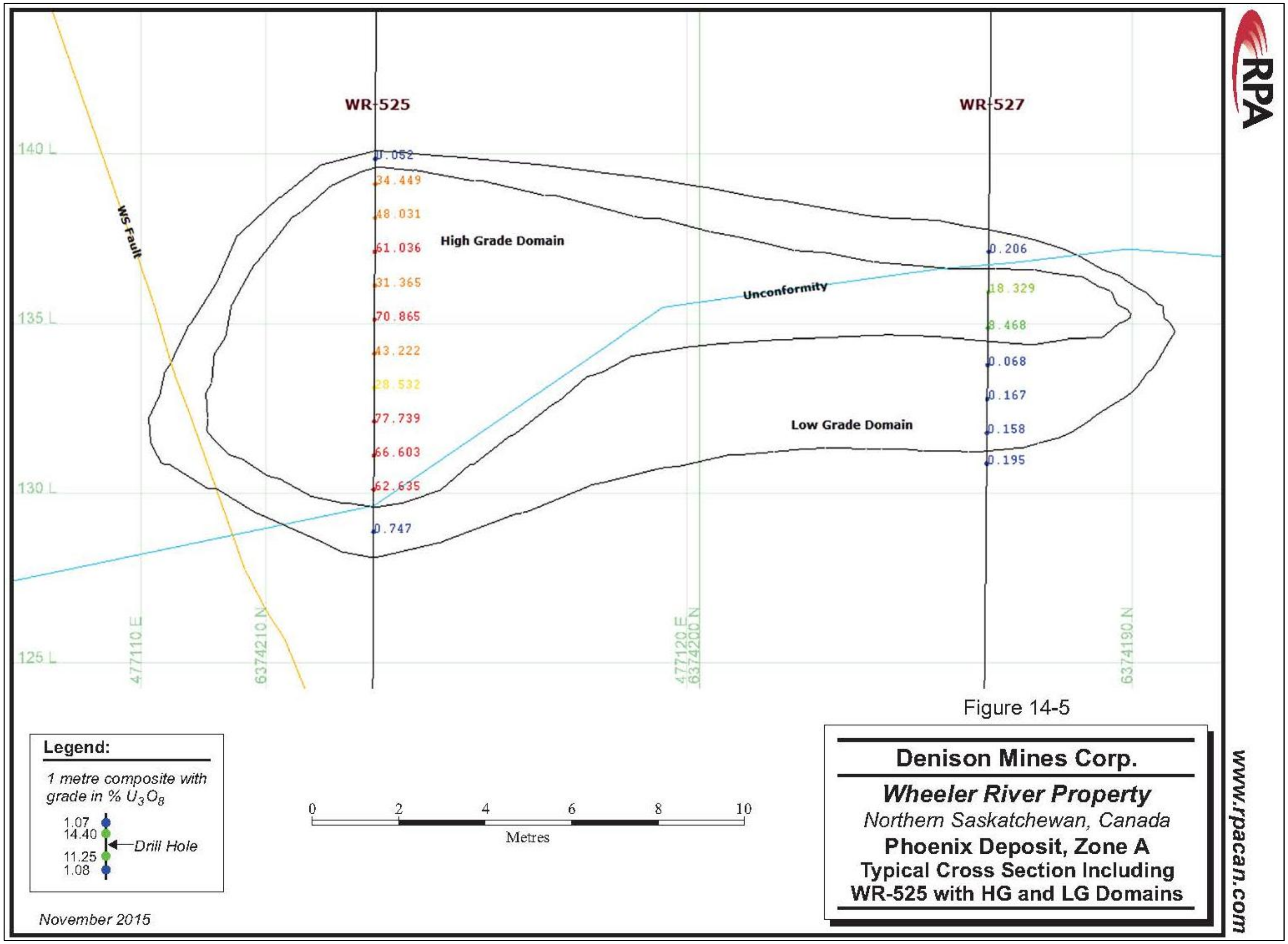



**Figure 14-5: Phoenix Deposit Zone A Typical Cross-Section Including WR-525 with HG and LG Domains**

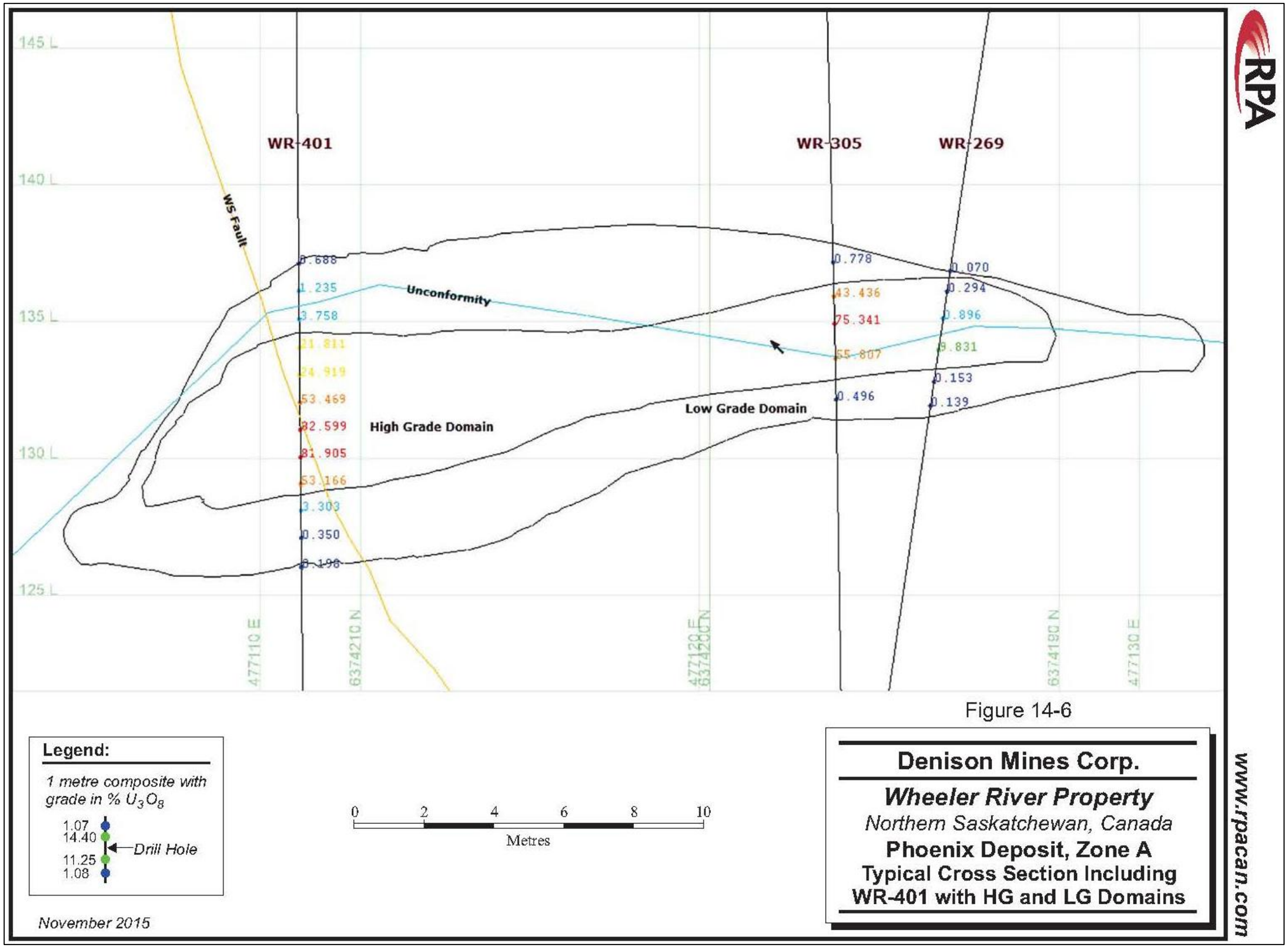



**Figure 14-6: Phoenix Deposit Zone A Typical Cross-Section Including WR-402 with HG and LG Domains**

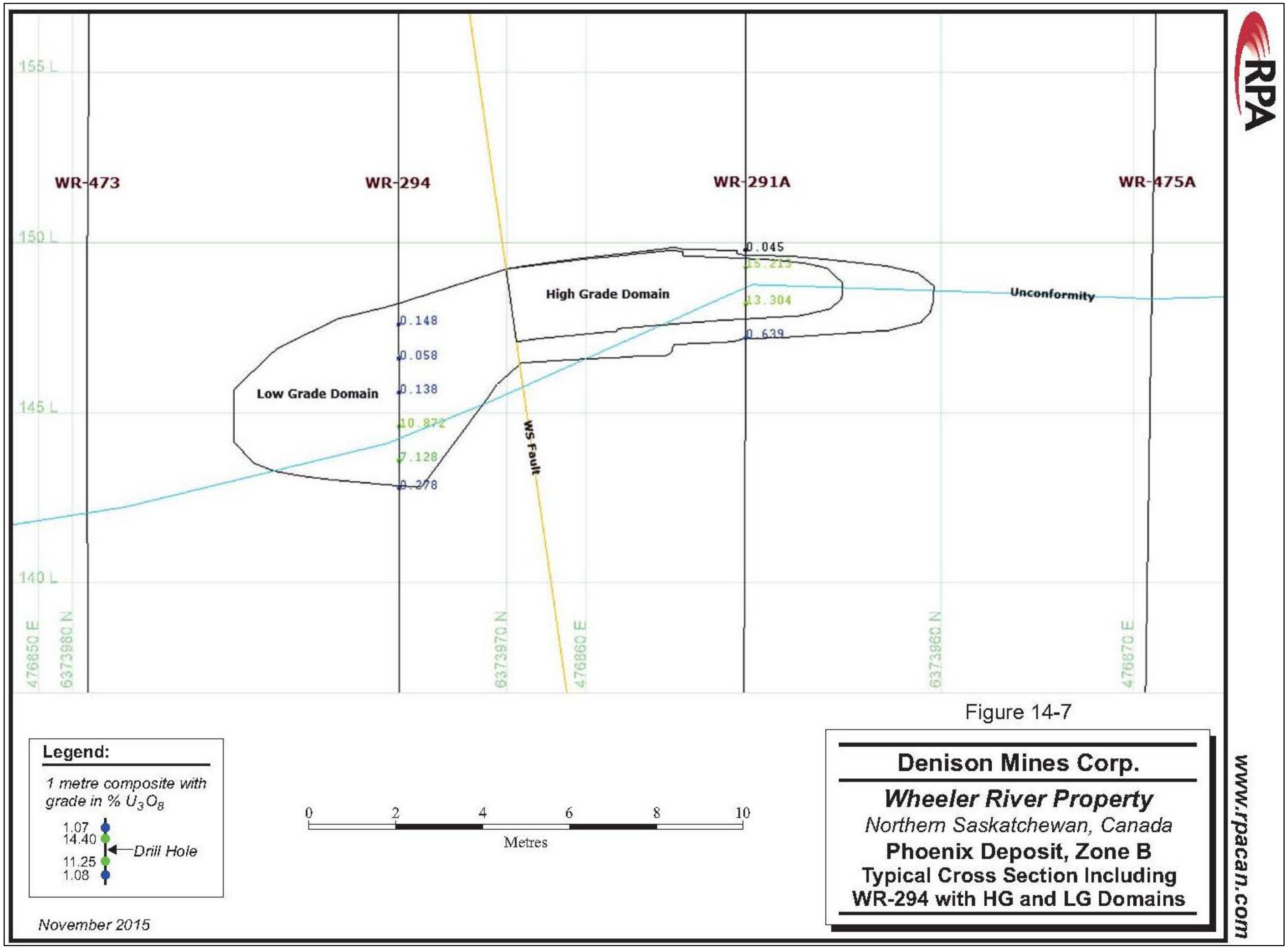



**Figure 14-7: Phoenix Deposit Zone B Typical Cross-section Including WR-294 with HG and LG Domains**

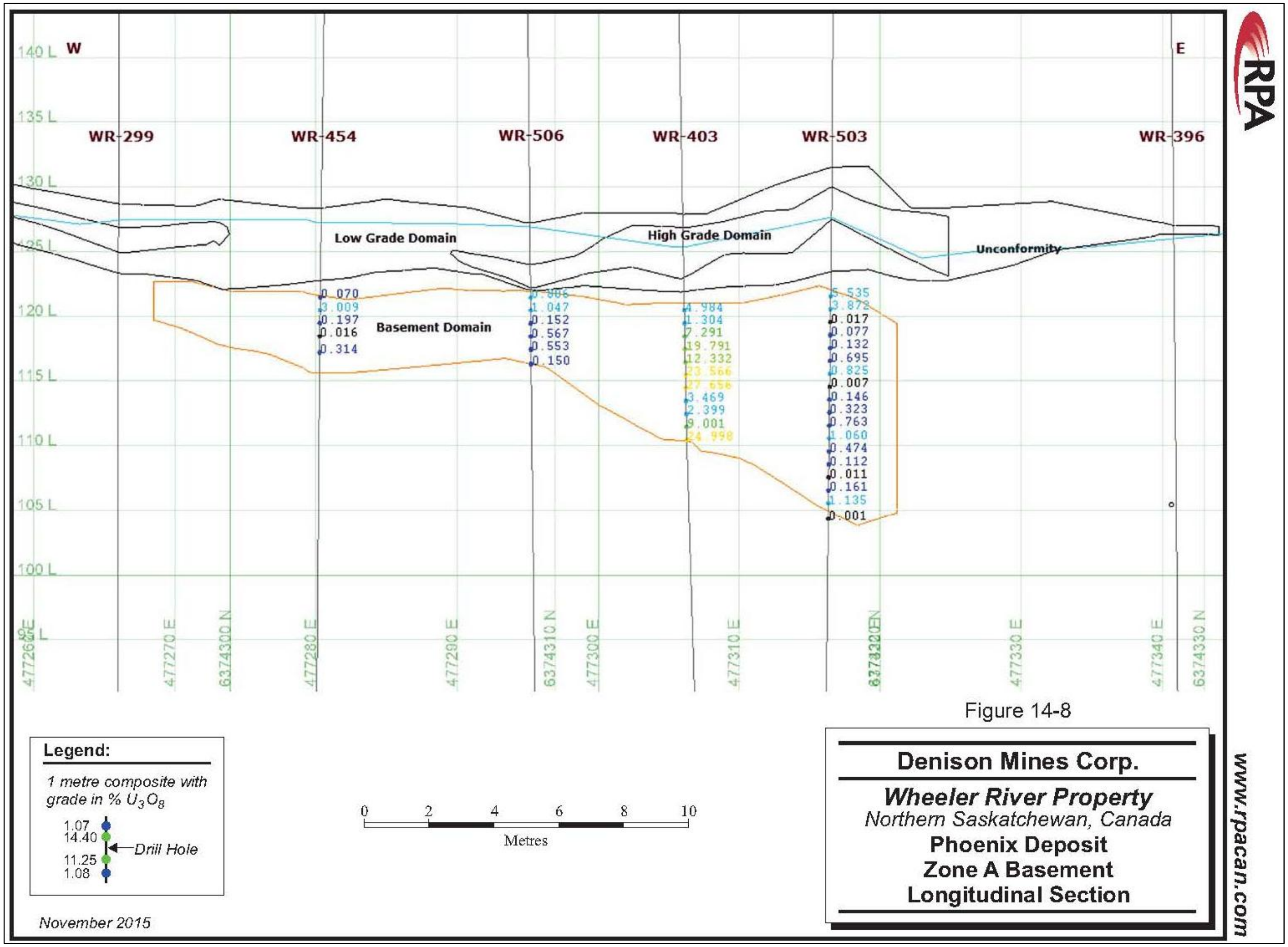



**Figure 14-8: Phoenix Deposit Zone A Basement Longitudinal Section**

### 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 55 holes. RPA built the wireframe models 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.

A threshold grade of 0.05% $U_3O_8$ and a minimum core length of 2 m was used as a guide, resulting in a series of eight 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 (Table 14-3 and Figure 14-9 and Figure 14-10). The mineralized wireframes were subsequently clipped to include only drill holes with intersections greater than 0.2% $U_3O_8$ over a minimum thickness of 2 m. 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 disclosures.

**Table 14-3: Summary of Gryphon Wireframe Models**

| Zone | Wireframe Name | Points | Triangles | Surface Area | Volume | Tonnage | Block Model Code |
|---|---|---|---|---|---|---|---|
| A1 | rpa_gryphon_min_a1_1.00t | 893 | 1,782 | 98,989 | 169,823 | 382,731 | 1 |
| A2 | rpa_gryphon_min_a2_1.00t | 810 | 1,616 | 62,098 | 72,158 | 162,622 | 2 |
| A3 | rpa_gryphon_min_a3_1.00t | 188 | 372 | 9,539 | 8,467 | 19,082 | 3 |
| B1 | rpa_gryphon_min_b1_1.00t | 525 | 1,046 | 46,301 | 63,537 | 143,192 | 4 |
| B2 | rpa_gryphon_min_b3_1.00t | 367 | 730 | 30,191 | 36,606 | 82,499 | 5 |
| B3 | rpa_gryphon_min_b2_1.00t | 205 | 406 | 8,855 | 11,467 | 25,844 | 6 |
| C1 | rpa_gryphon_min_c1_1.00t | 402 | 800 | 28,343 | 30,812 | 69,440 | 8 |
| C2 | rpa_gryphon_min_c2_1.00t | 511 | 1,018 | 13,895 | 12,053 | 27,163 | 9 |
| D1 | rpa_gryphon_min_d1.00t | 65 | 126 | 6,779 | 6,330 | 14,267 | 10 |
| D2 | rpa_gryphon_min_d2.00t | 46 | 88 | 3,121 | 2,912 | 6,563 | 11 |
| D3 | rpa_gryphon_min_d3.00t | 14 | 24 | 1,215 | 1,046 | 2,357 | 12 |
| D4 | rpa_gryphon_min_d4.00t | 24 | 44 | 800 | 637 | 1,435 | 13 |
| **Total** | | **4,050** | **8,052** | **310,127** | **415,848** | **937,196** | |

Notes:

- A-Series (A1, A2, and A3): represent the mineralized zones on the hanging wall (Upper Zone) of the quartz-pegmatite assemblage (wireframes 1, 2 and 3)
- B-Series (B1, B2, and B3): represent the mineralized zones within the quartz-pegmatite assemblage (wireframes 4, 5, and 6)
- C-Series (C1 and C2): represent the mineralized zones along the foot wall (Lower Zone) of the quartz-pegmatite assemblage (wireframes 8 and 9)
- D-Series (D1, D2, D3, and D4): represent four low grade mineralized zones (wireframes 10, 11, 12 and 13), which do not have enough drilling to be included in the resource estimate

Mineral resources were estimated for the A, B, and C Series lenses, and not for the D Series, which were considered to have insufficient drilling. The A1 and C1 domains collectively make up nearly 69% 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 2015 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.
- 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

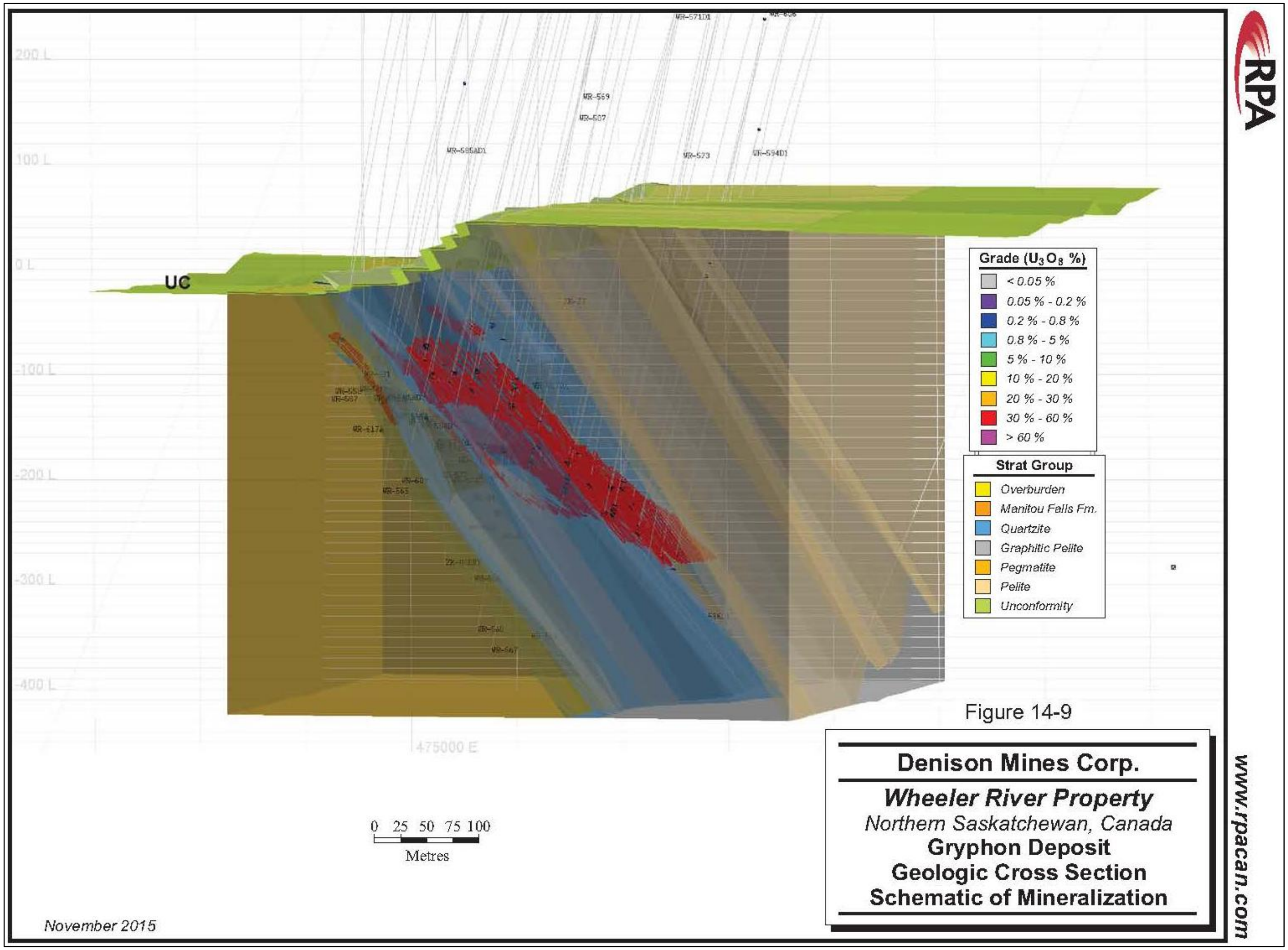



**Figure 14-9: Gryphon Deposit Geologic Cross-section Schematic of Mineralization**

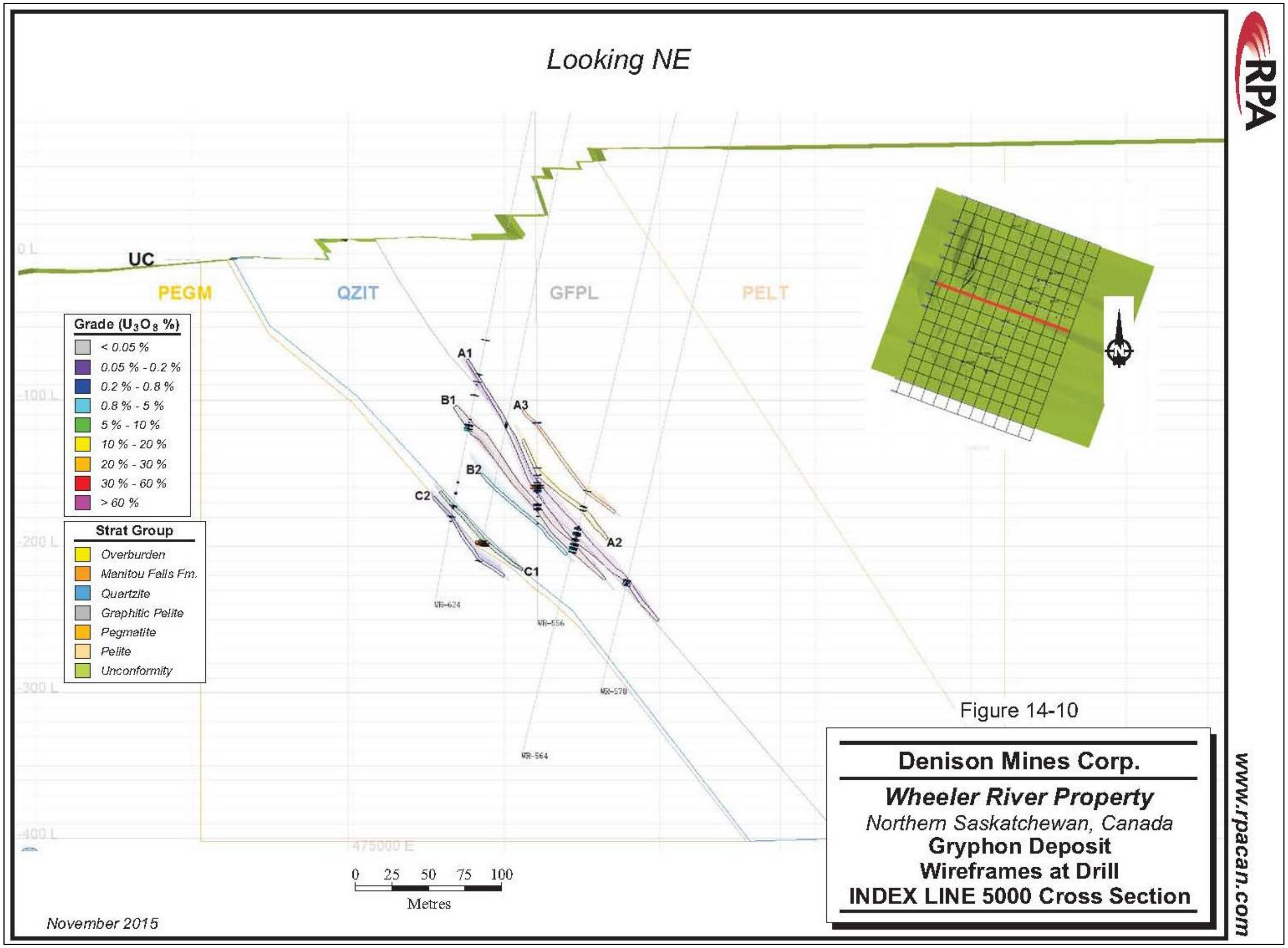



**Figure 14-10: Gryphon Deposit Wireframes at Drill Index Line 5000 Cross-section (Looking NE)**

## 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.

$$y = 0.0008x2 – 0.0077x + 2.3361$$

where *y* is dry bulk density (g/c $m^3$) 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.

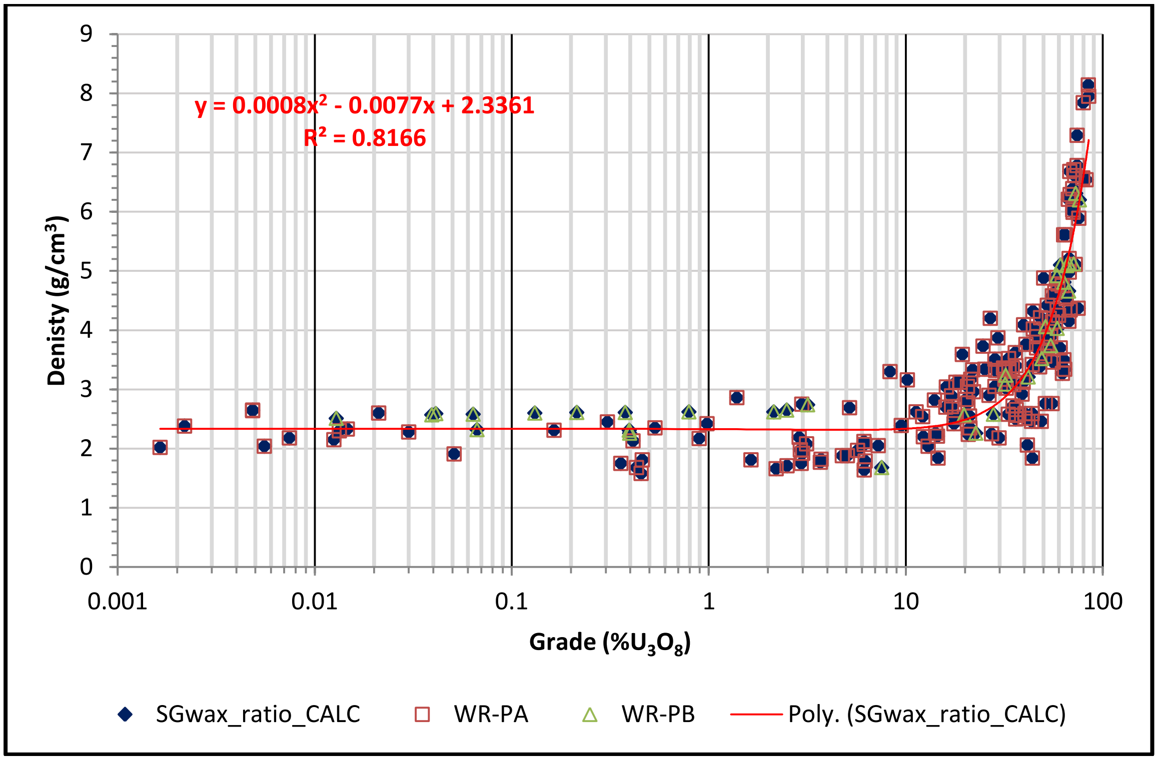



**Figure 14-11: Logarithmic Plot of Dry Bulk Density versus Uranium Grade – Phoenix Deposit**

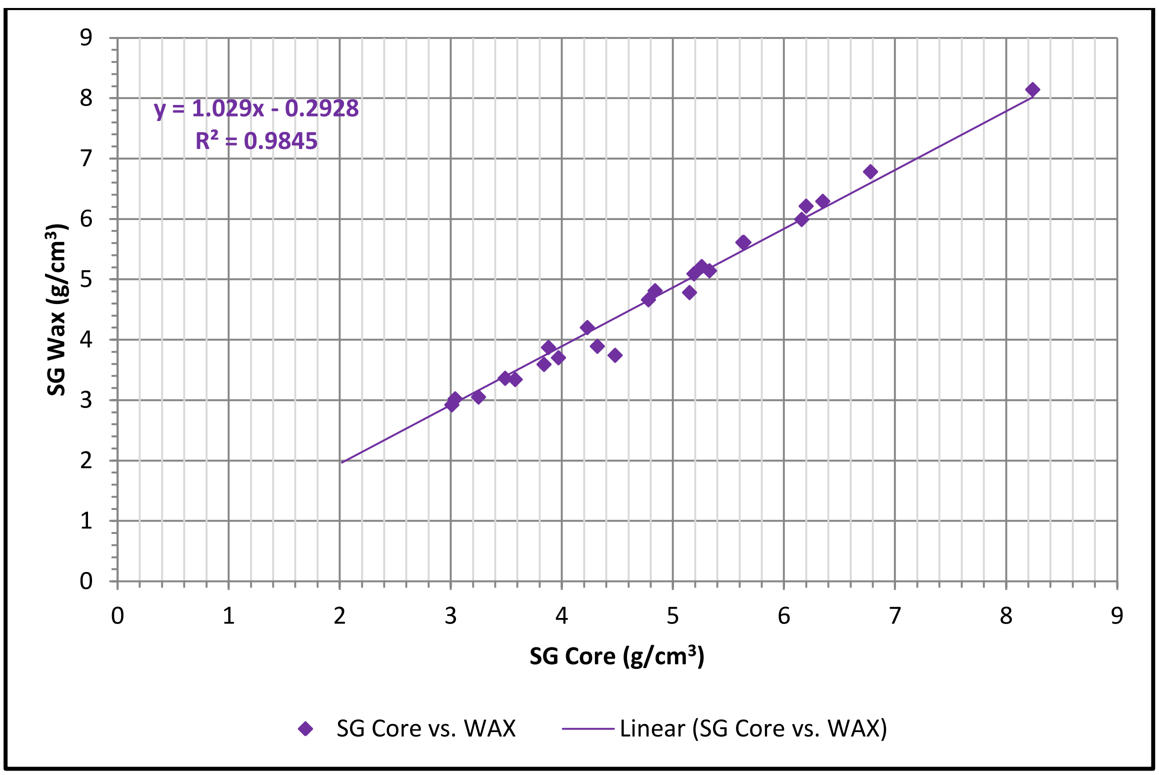



**Figure 14-12: Dry Bulk Density Wax versus Dry/Wet Methods – Phoenix Deposit**

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/c $m^3$. 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 65 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.

$$y = 4E\text{-}05x2 + 0.0166x + 2.2537$$

where *y* is dry bulk density (g/c $m^3$) 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.

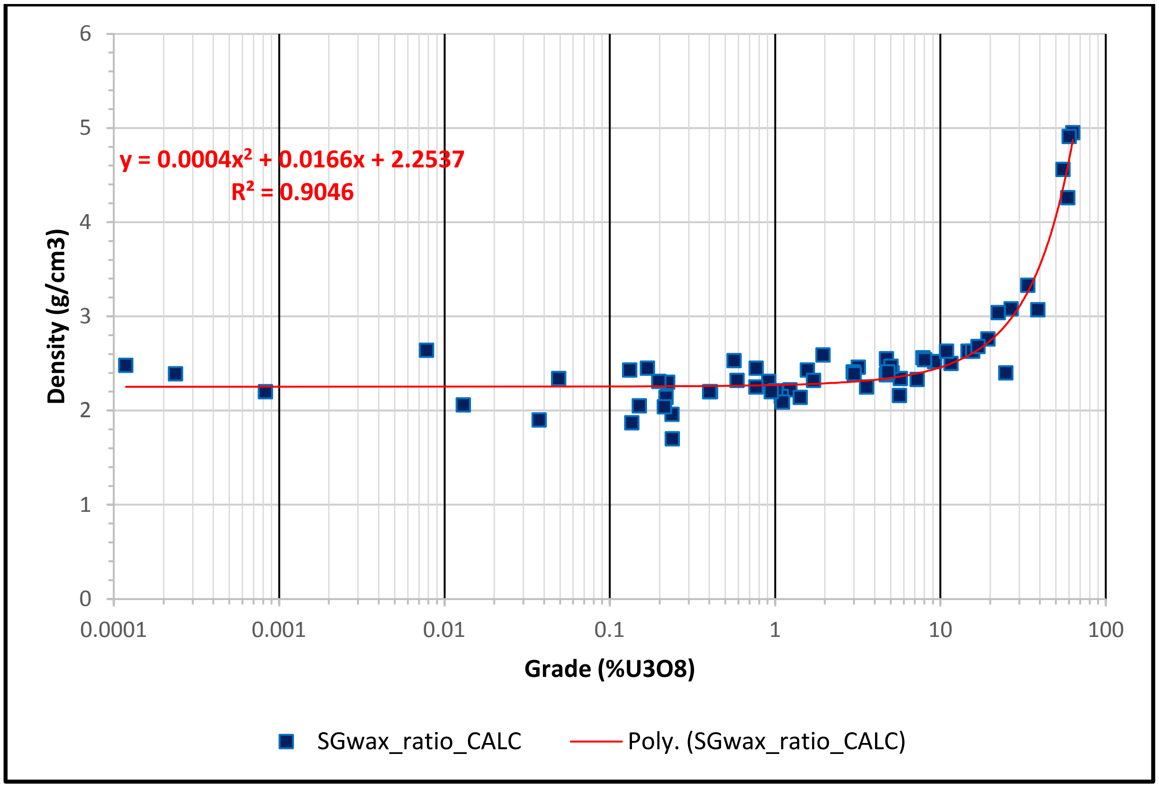



**Figure 14-13: Logarithmic Plot of Dry Bulk Density versus Uranium Grade – Gryphon Deposit**

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

Assay values located inside the wireframe models were tagged with domain identifiers and exported for statistical analysis. Results were used to help verify the modelling process. Basic statistics by domain are summarized in Table 14-4.

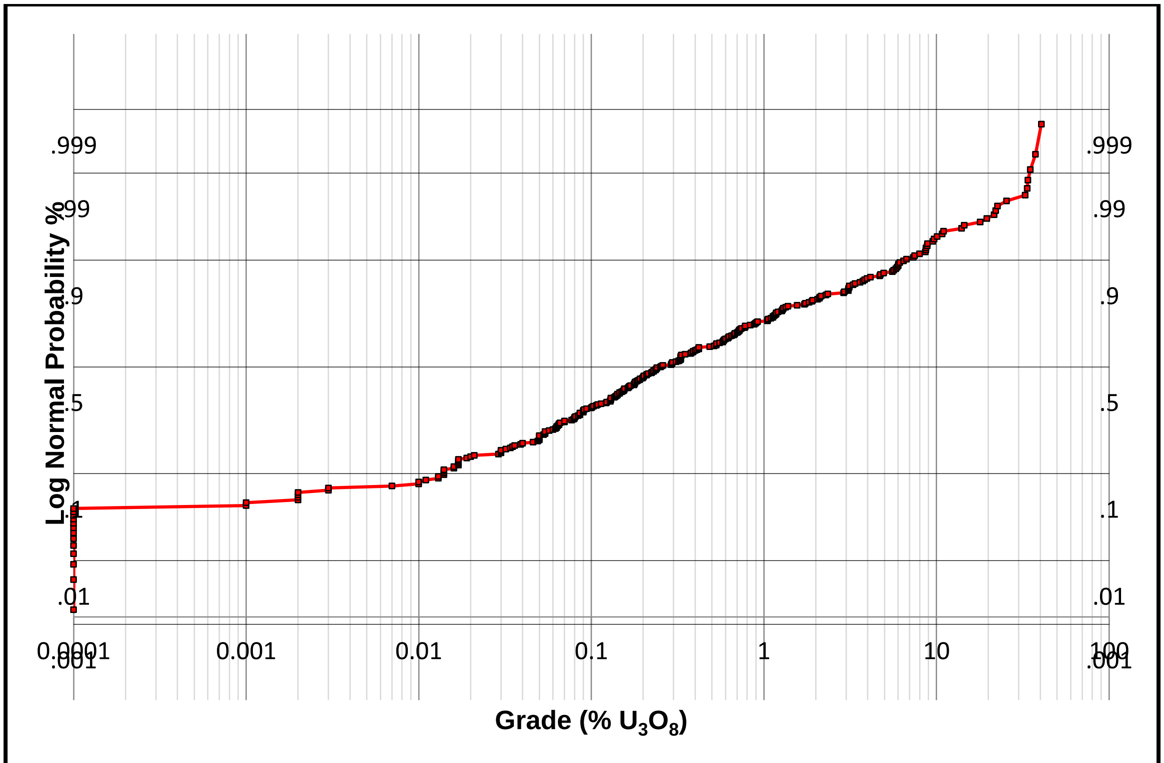
**Table 14-4: Descriptive Statistics of Gryphon Uranium Assay by Domain**

| Descriptive Statistic | Zone A1 | Zone A2 | Zone A3 | Zone B1 |
|---|---|---|---|---|
| Count | 281 | 102 | 17 | 190 |
| Mean | 2.46 | 0.76 | 0.55 | 0.58 |
| Median | 0.25 | 0.08 | 0.16 | 0.09 |
| Std. Dev. | 6.22 | 1.75 | 1.14 | 2.00 |
| Variance | 38.73 | 3.05 | 1.30 | 4.00 |
| Kurtosis | 17.03 | 8.02 | 6.38 | 46.77 |
| Skewness | 4.00 | 2.94 | 2.73 | 6.44 |
| Range | 40.60 | 8.67 | 4.56 | 17.10 |
| Minimum | 0.00 | 0.00 | 0.00 | 0.00 |
| Maximum | 40.60 | 8.67 | 4.56 | 17.10 |
| Coefficient of Variation | 2.53 | 2.29 | 2.06 | 3.47 |
| | **Zone B2** | **Zone B3** | **Zone C1** | **Zone C2** |
| Count | 49 | 24 | 46 | 15 |
| Mean | 2.60 | 3.89 | 4.75 | 0.26 |
| Median | 0.25 | 0.43 | 0.42 | 0.16 |
| Std. Dev. | 4.77 | 8.37 | 10.39 | 0.29 |
| Variance | 22.79 | 70.12 | 107.91 | 0.09 |
| Kurtosis | 3.31 | 6.73 | 5.68 | 0.47 |
| Skewness | 2.09 | 2.70 | 2.60 | 1.21 |
| Range | 18.80 | 36.00 | 42.50 | 1.01 |
| Minimum | 0.00 | 0.00 | 0.00 | 0.00 |
| Maximum | 18.80 | 36.00 | 42.50 | 1.01 |
| Coefficient of Variation | 1.84 | 2.15 | 2.19 | 1.14 |

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 30% (Figure 14-14), which only impacts zones A1, B3, and C1. Results of the capping impacted 10 (1.2%) values out of 834 assays. Table 14-5 lists descriptive statistics for the domains affected by cutting.

**Table 14-5: Statistics of Gryphon Capped Assays by Domain**

| | Zone A1 | | Zone B3 | | Zone C1 | |
|---|---|---|---|---|---|---|
| **Descriptive Statistic** | **Raw** | **Cap** | **Raw** | **Cap** | **Raw** | **Cap** |
| Count | 281 | 281 | 24 | 24 | 46 | 46 |
| Mean | 2.46 | 2.34 | 3.89 | 3.64 | 4.75 | 4.20 |
| Median | 0.25 | 0.25 | 0.43 | 0.43 | 0.42 | 0.42 |
| Std. Dev. | 6.22 | 5.61 | 8.37 | 7.61 | 10.39 | 8.66 |
| Variance | 38.73 | 31.49 | 70.12 | 57.91 | 107.91 | 75.04 |
| Kurtosis | 17.03 | 13.45 | 6.73 | 6.21 | 5.68 | 4.36 |
| Skewness | 4.00 | 3.61 | 2.70 | 2.56 | 2.60 | 2.36 |
| Range | 40.60 | 30.00 | 36.00 | 30.00 | 42.50 | 30.00 |
| Minimum | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Maximum | 40.60 | 30.00 | 36.00 | 30.00 | 42.50 | 30.00 |
| Coefficient of Variation | 2.53 | 2.39 | 2.15 | 2.09 | 2.19 | 2.06 |



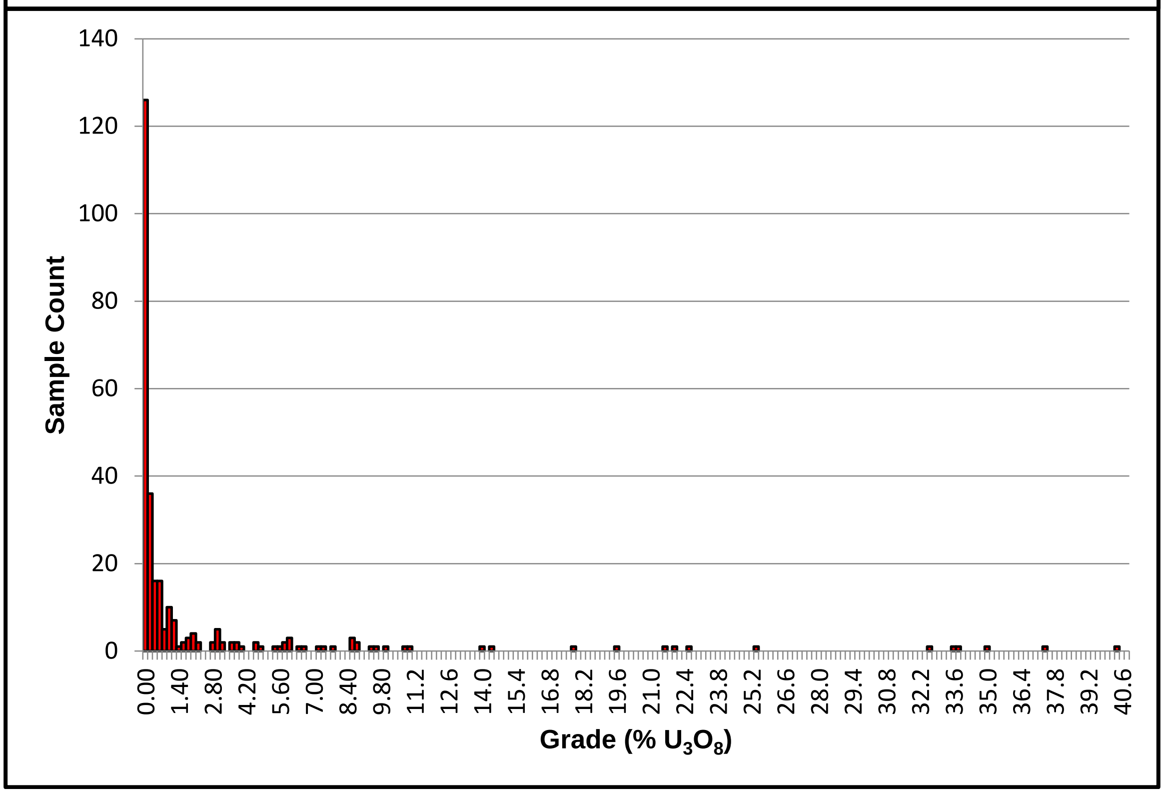




**Figure 14-14: Zone A1 Log Normal Probability and Histogram Plot – Gryphon Deposit**

## 14.4.2 Composites

As discussed in Section 10 Drilling and Section 11 Sample Preparation, Analyses and Security, 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%.

Sample lengths range from 0.5 cm to 1.0 m within the wireframe models, however, 99.85% of the samples were taken at 0.5 m intervals. Given this distribution, and considering the width of the mineralization, RPA composited uranium grade (G), bulk density (D), and uranium grade multiplied by density (GxD) values over 1 m run-length intervals to create a composite database for statistical analysis and block estimation purposes. Assay grades are weighted by both sample length and density when compositing. Compositing was restricted to within the wireframe models (hard boundaries). This can result in residual short composites at the bottom of the wireframes. These short composites were retained if they were between 0.5 m and 1.0 m long, and were added to the previous full length composite if they were less than 0.5 m long. As discussed below, block estimation was done by interpolating GxD and density and dividing them to obtain a density-weighted grade estimate for each block.

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 |

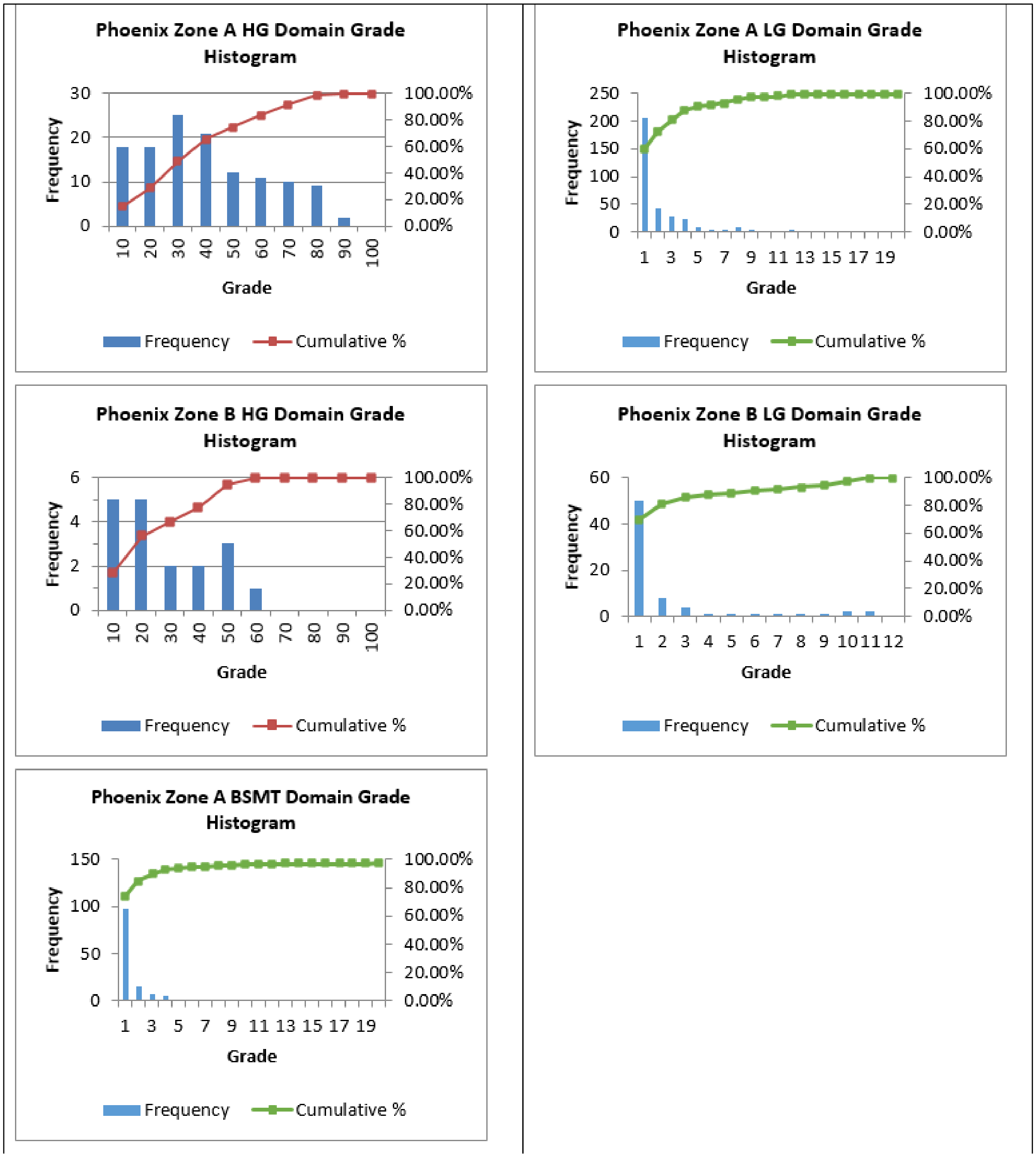



**Figure 14-15: Grade Composite Histograms for Phoenix Deposit Zones A and B HG and LG Domains**

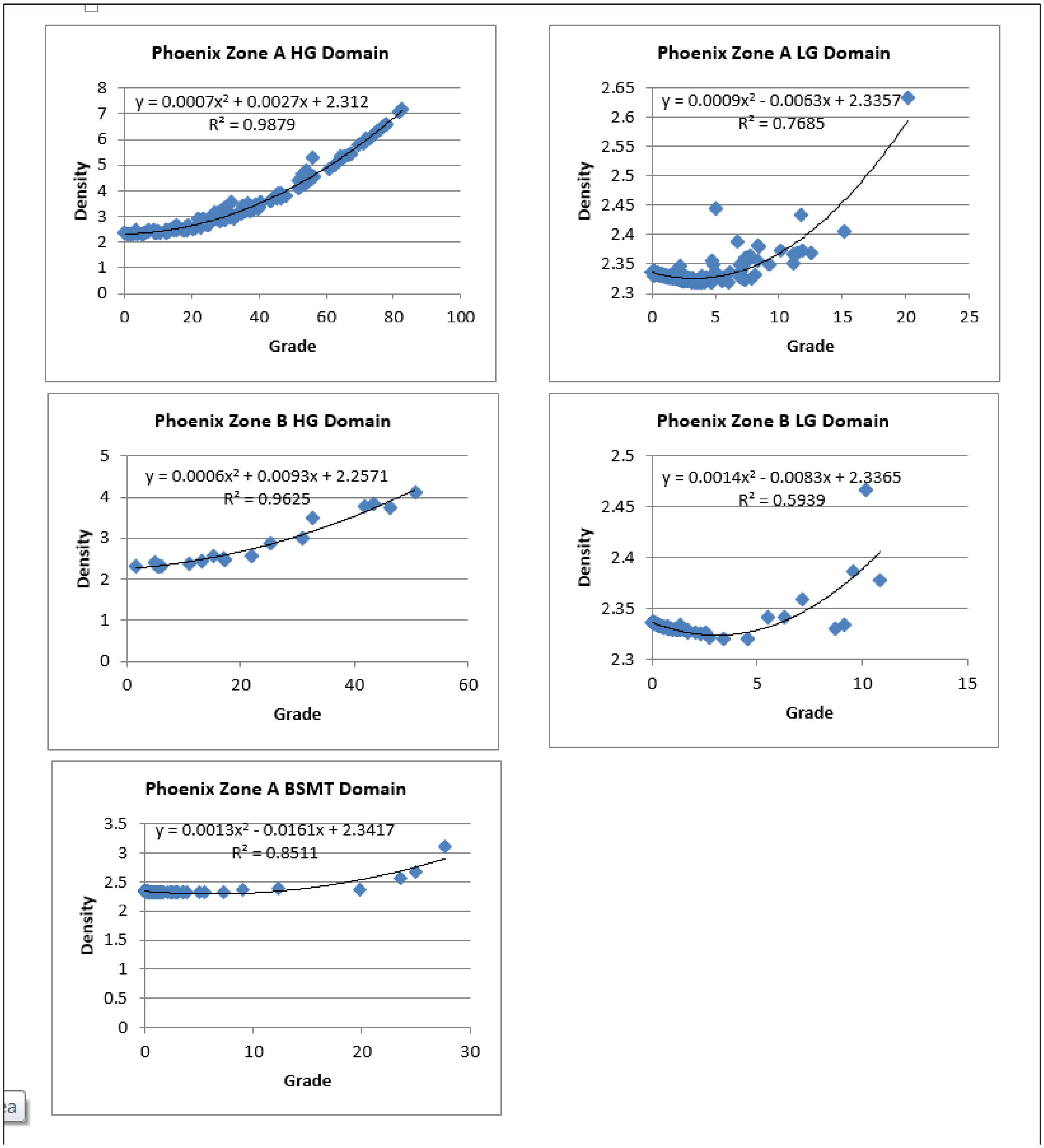



**Figure 14-16: Grade versus Density Plots for Phoenix Deposit Zones A and B HG and LG Domains**

### 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**

| Descriptive Statistic | Zone A1 | Zone A2 | Zone A3 | Zone B1 |
|---|---|---|---|---|
| Count | 131 | 51 | 4 | 51 |
| Mean | 2.25 | 0.76 | 0.94 | 0.89 |
| Median | 0.38 | 0.16 | 0.67 | 0.18 |
| Std. Dev. | 4.53 | 1.32 | 0.96 | 2.34 |
| Variance | 20.52 | 1.74 | 0.92 | 5.46 |
| Kurtosis | 12.36 | 3.74 | -1.28 | 16.26 |
| Skewness | 3.26 | 2.16 | 0.55 | 4.10 |
| Range | 30.00 | 5.43 | 2.42 | 12.34 |
| Minimum | 0.00 | 0.00 | 0.00 | 0.00 |
| Maximum | 30.00 | 5.43 | 2.42 | 12.34 |
| Coefficient of Variation | 2.02 | 1.74 | 1.02 | 2.64 |

| | Zone B2 | Zone B3 | Zone C1 | Zone C2 |
|---|---|---|---|---|
| Count | 26 | 15 | 26 | 15 |
| Mean | 2.33 | 2.76 | 3.48 | 0.19 |
| Median | 0.33 | 0.44 | 0.56 | 0.01 |
| Std. Dev. | 3.85 | 4.45 | 6.70 | 0.35 |
| Variance | 14.81 | 19.84 | 44.90 | 0.12 |
| Kurtosis | 1.97 | 1.46 | 3.15 | 5.94 |
| Skewness | 1.81 | 1.67 | 2.16 | 2.56 |
| Range | 13.54 | 14.77 | 24.03 | 1.39 |
| Minimum | 0.00 | 0.00 | 0.00 | 0.00 |
| Maximum | 13.54 | 14.77 | 24.03 | 1.39 |
| Coefficient of Variation | 1.65 | 1.61 | 1.92 | 1.83 |

## 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 has not been undertaken because the current drill hole spacing and number of samples are not adequate to generate meaningful variograms.

## 14.6 Interpolation Parameters

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 variables used.

**Table 14-8: Phoenix and Gryphon Block Model Parameters and Variables**

| **Model name:** | **H:\PROJECTS\2534 - Denison Mines Corp - Gryphon Deposit\Work\VULCAN_30Sept2015\wr_gryphon_sept2015_2m_capped-30** |
|---|---|
| History list: | wr_gryphon_sept2015_2m_capped-3014Oct2015.bhst |
| Format: | extended |
| Structure: | regular |
| Smooth: | no |
| Number of blocks: | 19,250,000 |
| Number of variables: | 13 |
| Number of schemas: | 1 |
| Origin: | 474,768.281 6,376,260.0 -400.0 |
| Bearing/Dip/Plunge: | 110.0 0.0 0.0 |
| Offset: | 550.0 700.00 1000.0 |
| Model name: | H:\PROJECTS\2299-Denison Mines Corp.-Phoenix Uranium Deposits\Work\Vulcan\Phoenix\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: | H:\PROJECTS\2299-Denison Mines Corp.-Phoenix Uranium Deposits\Work\Vulcan\Phoenix\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: | H:\PROJECTS\2299-Denison Mines Corp.-Phoenix Uranium Deposits\Work\Vulcan\Phoenix\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: | H:\PROJECTS\2299-Denison Mines Corp.-Phoenix Uranium Deposits\Work\Vulcan\Phoenix\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 |

| 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 neighbor |
| 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 |

### 14.6.1 Phoenix Deposit

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 14-9 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 14-9: 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% U3O8 | 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% U3O8 | 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% U3O8 | 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% U3O8 | 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% U3O8 | 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

Following the generation of the wireframes for each zone, the wireframes were filled by a block model. The wireframes were used to assign domain codes to the blocks in the block model and for generating and coding composited assays (Figure 14-19). RPA determined that the 2 m by 10 m by 1 m block size was appropriate for modelling the individual mineralized units.

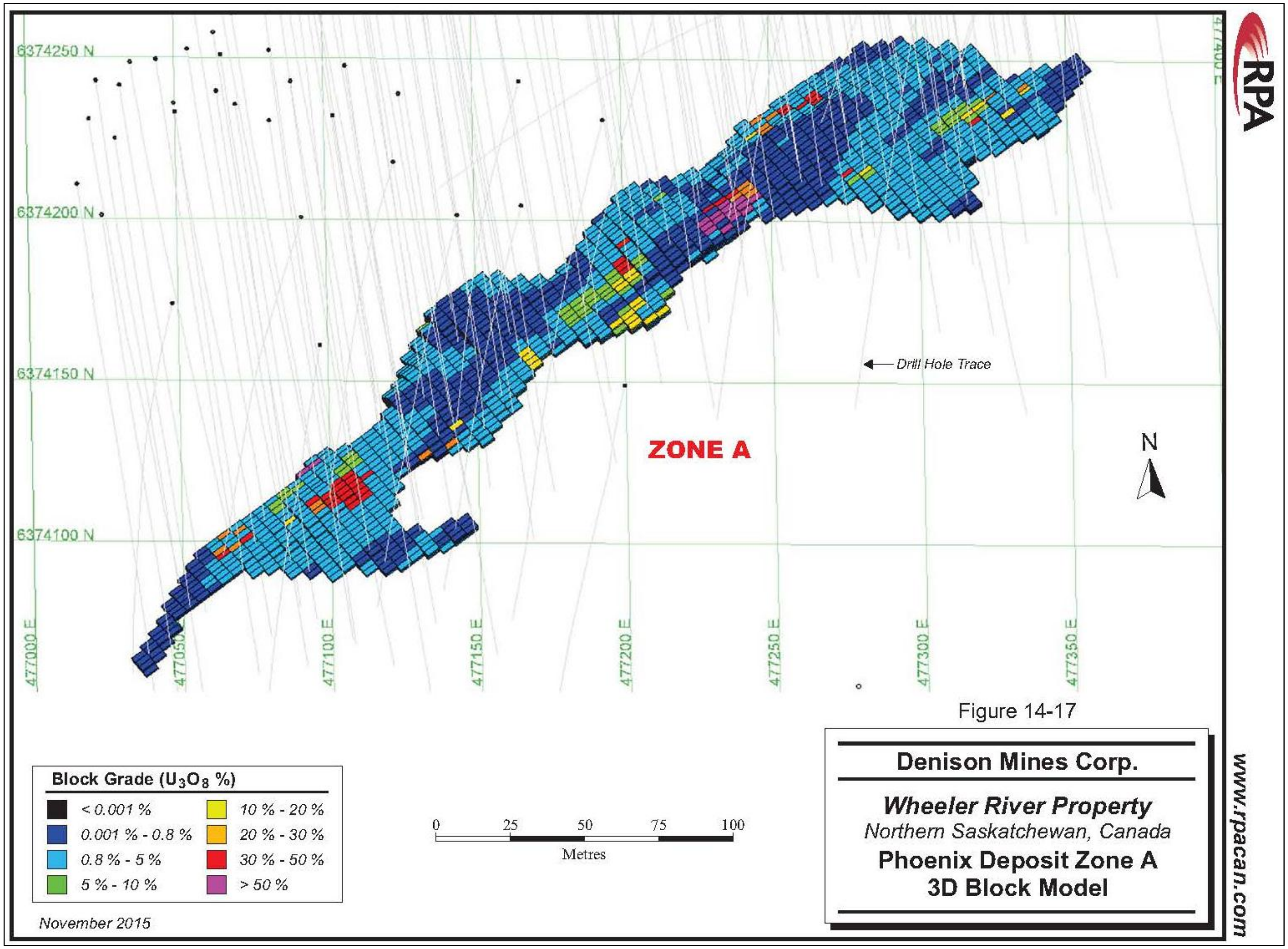



**Figure 14-17: Phoenix Deposit Zone A Block Model**

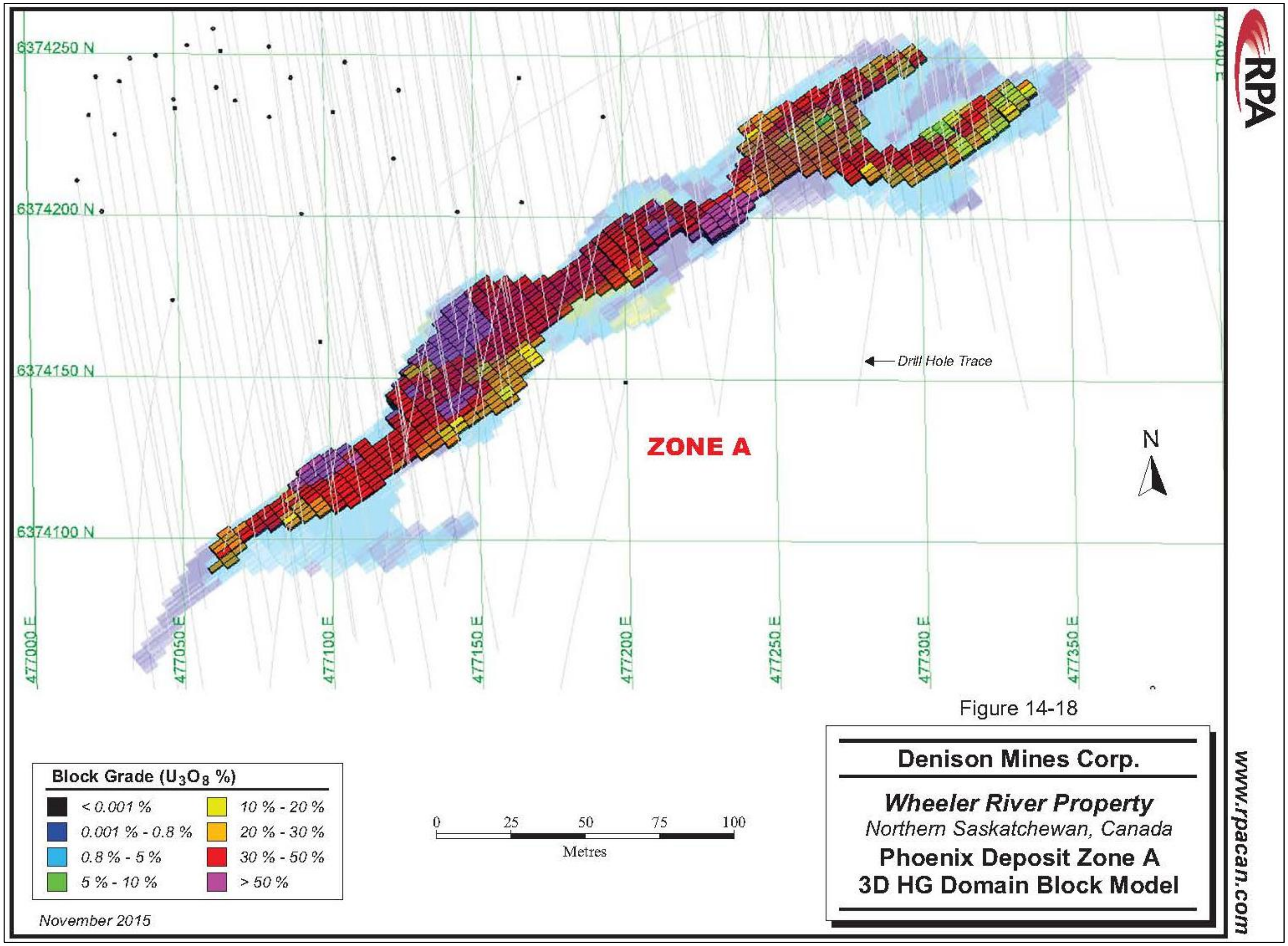



**Figure 14-18: Phoenix Deposit Zone A 3D HG Domain Block Model**

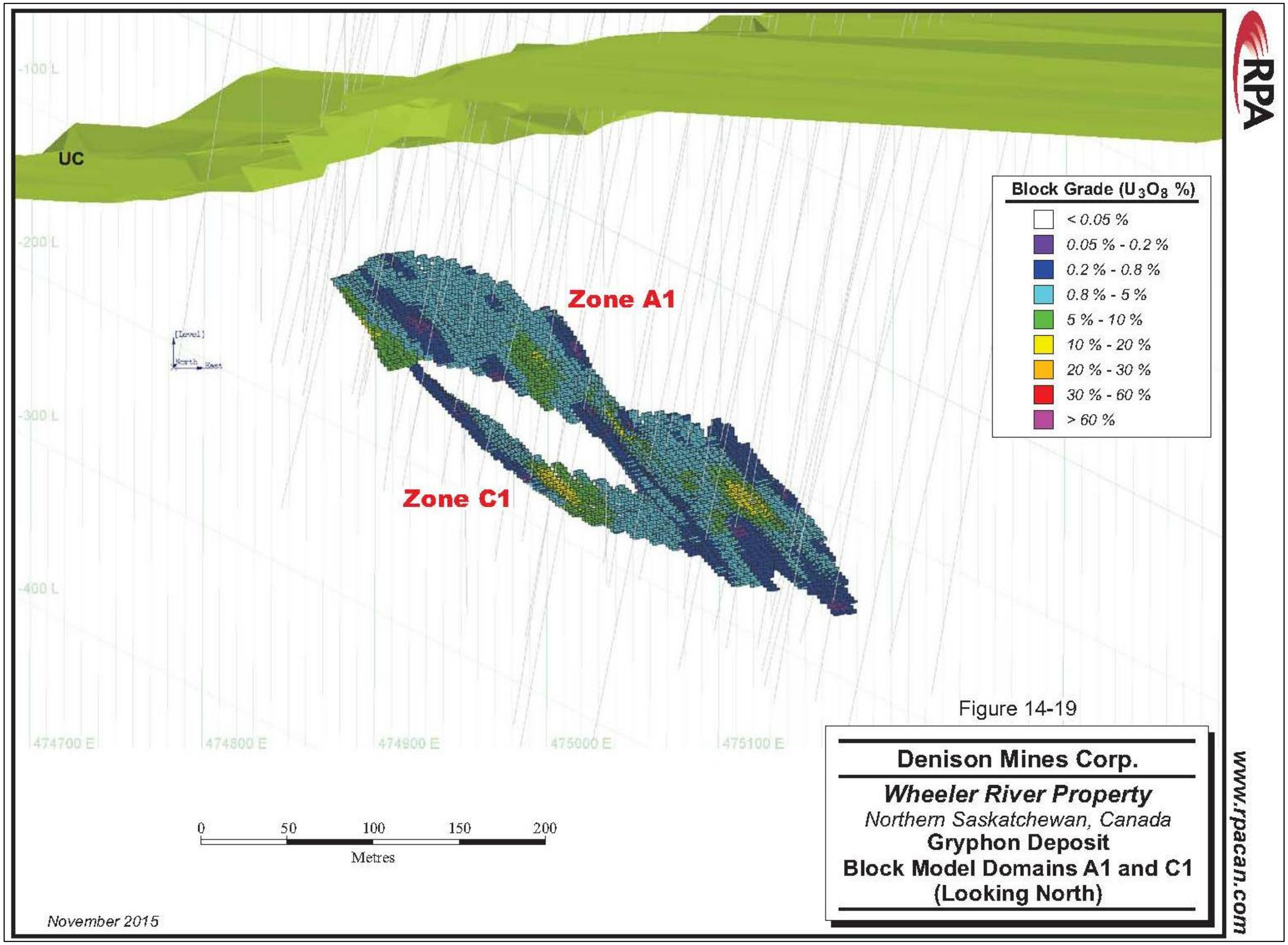



**Figure 14-19: Gryphon Deposit Block Model Domains A1 and C1 (Looking North)**

Composited GxD values and D values were interpolated into each block model domain using an ID2 algorithm for each mineralized domain. Domain boundaries were treated as hard boundaries, so that composites from any given domain could not influence block grades in other domains. Block grade was derived from the interpolated GxD value divided by the interpolated D value for each block (GxD/D). Block tonnage was based on volume times the interpolated D value.

The interpolation strategy involved setting up search parameters in two passes for each individual mineralized wireframe. Table 14-10 provides a list of the estimation parameters used for each pass, and all wireframes were subject to the same estimation parameters.

**Table 14-10: Gryphon Block Model Estimation Parameters**

| Estimation_ id | Flag_var | Flag_ value | Alpha | Zeta | Beta | Major | Semi | Minor | Min samples | Max samples | Dh_limit |
|---|---|---|---|---|---|---|---|---|---|---|---|
| id1_a1 | est_flag_id | 1 | 40 | -30 | -45 | 120 | 60 | 12 | 6 | 9 | 3 |
| id1_a2 | est_flag_id | 1 | 40 | -25 | -45 | 120 | 60 | 12 | 2 | 6 | 2 |
| id1_a3 | est_flag_id | 1 | 40 | -25 | -45 | 120 | 60 | 12 | 2 | 6 | 2 |
| id1_b1 | est_flag_id | 1 | 40 | -25 | -45 | 120 | 60 | 12 | 6 | 9 | 3 |
| id1_b2 | est_flag_id | 1 | 40 | -25 | -45 | 120 | 60 | 12 | 2 | 6 | 2 |
| id1_b3 | est_flag_id | 1 | 40 | -25 | -45 | 120 | 60 | 12 | 3 | 9 | 3 |
| id1_b4 | est_flag_id | 1 | 40 | -25 | -45 | 120 | 60 | 12 | 2 | 6 | 2 |
| id1_c1 | est_flag_id | 1 | 40 | -20 | -40 | 120 | 60 | 12 | 6 | 9 | 3 |
| id1_c2 | est_flag_id | 1 | 40 | -20 | -35 | 120 | 60 | 12 | 2 | 6 | 2 |
| id1_d1 | est_flag_id | 1 | 40 | -30 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id1_d2 | est_flag_id | 1 | 40 | -30 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id1_d3 | est_flag_id | 1 | 40 | -30 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id1_d4 | est_flag_id | 1 | 40 | -30 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id2_a1 | est_flag_id | 2 | 40 | -30 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id2_a2 | est_flag_id | 2 | 40 | -25 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id2_a3 | est_flag_id | 2 | 40 | -25 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id2_b1 | est_flag_id | 2 | 40 | -25 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id2_b2 | est_flag_id | 2 | 40 | -25 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id2_b3 | est_flag_id | 2 | 40 | -25 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id2_b4 | est_flag_id | 2 | 40 | -25 | -45 | 120 | 60 | 12 | 2 | 4 | 2 |
| id2_c1 | est_flag_id | 2 | 40 | -20 | -40 | 120 | 60 | 12 | 2 | 4 | 2 |
| id2_c2 | est_flag_id | 2 | 40 | -20 | -35 | 120 | 60 | 12 | 2 | 4 | 2 |
| id2_d1 | est_flag_id | 2 | 40 | -30 | -45 | 120 | 60 | 12 | 1 | 2 | 2 |
| id2_d2 | est_flag_id | 2 | 40 | -30 | -45 | 120 | 60 | 12 | 1 | 2 | 2 |
| id2_d3 | est_flag_id | 2 | 40 | -30 | -45 | 120 | 60 | 12 | 1 | 2 | 2 |
| id2_d4 | est_flag_id | 2 | 40 | -30 | -45 | 120 | 60 | 12 | 1 | 2 | 2 |

## 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
- 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 14-11 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, A3, and B2 domains where the difference ranges from 3.5% to 6% due to the small volume of the wireframe combined with the whole block approach.

### 14.7.2 Visual Comparison

Block grades were visually compared with drill hole composites on cross-sections, longitudinal sections, and plan views. The block grades and composite grades correlate very well visually within 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 14-12 for all blocks and composites within the Phoenix deposit Zones A and B, HG, and LG domains. Table 14-13 lists the composites versus block grades for Gryphon. Grades are weighted by density for the composites and tonnage for the blocks. In some cases, the average block grades are higher than the average composite grades, which RPA attributes to density weighting of the block grades or distribution of the drill holes within relatively small zones.

**Table 14-11: 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 | 4.05% |
| Zone B LG | 1,604 | 3,254 | 14,911 | 15,142 | 1,492 | 14,920 | -1.49% |
| Zone A Basement | 132 | 260 | 2009 | 2 | 1,115 | 2,230 | -1.02% |
| **Gryphon Deposit** | | | | | | | |
| A1 | 783 | 1,562 | 101,254 | 172,570 | 8614 | 172,280 | 0.17% |
| A2 | 770 | 1,536 | 76,882 | 86,959 | 4346 | 86,920 | 0.04% |
| A3 | 206 | 404 | 22,676 | 20,305 | 955 | 19,100 | 5.94% |
| B1 | 589 | 1,174 | 73,444 | 101,644 | 5156 | 103,120 | -1.45% |
| B2 | 268 | 532 | 32,089 | 38,599 | 1861 | 37,220 | 3.57% |
| B3 | 82 | 160 | 11,205 | 14,929 | 762 | 15,240 | -2.08% |
| C1 | 343 | 682 | 31,363 | 33,766 | 1648 | 32,960 | 2.39% |
| C2 | 267 | 530 | 23,213 | 21,189 | 1053 | 21,060 | 0.61% |

**Table 14-12: Statistics of Block Grades Compared to Composite Grades by Domain – Phoenix**

| Statistic | Zone A HG | | Zone A LG | | Zone A BSMT | | Zone B HG | | Zone B LG | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Blocks | Comps | Blocks | Comps | Blocks | Comps | Blocks | Comps | Blocks | Comps |
| Mean (%$U_3O_8$) | 39.18 | 34.86 | 1.73 | 1.77 | 1.35 | 1.56 | 25.71 | 21.65 | 1.34 | 1.57 |
| Standard Error | 0.37 | 1.93 | 0.02 | 0.14 | 0.35 | 0.36 | 0.59 | 3.74 | 0.04 | 0.31 |
| Median (%$U_3O_8$) | 36.51 | 31.52 | 1.22 | 0.59 | 0.14 | 0.32 | 26.63 | 17.14 | 0.69 | 0.53 |
| Mode (%$U_3O_8$) | N/A | N/A | 0.44 | 0.18 | 0.00 | 0.00 | N/A | N/A | N/A | 0.25 |
| Standard Deviation (%$U_3O_8$) | 15.63 | 21.62 | 1.72 | 2.69 | 4.11 | 4.26 | 10.66 | 15.85 | 1.65 | 2.64 |
| Sample Variance | 244.16 | 467.56 | 2.98 | 7.23 | 16.91 | 18.12 | 113.73 | 251.25 | 2.71 | 6.99 |
| Kurtosis | -0.13 | -0.69 | 16.02 | 10.25 | 25.63 | 23.16 | -1.18 | -1.02 | 5.04 | 4.65 |
| Skewness | 0.67 | 0.45 | 3.05 | 2.81 | 4.90 | 4.72 | -0.08 | 0.54 | 2.23 | 2.36 |
| Range (%$U_3O_8$) | 77.76 | 82.31 | 19.85 | 20.13 | 27.82 | 27.66 | 44.86 | 49.24 | 10.48 | 10.86 |
| Min (%$U_3O_8$) | 4.62 | 0.29 | 0.03 | 0.01 | 0.00 | 0.00 | 3.46 | 1.46 | 0.01 | 0.01 |
| Max (%$U_3O_8$) | 82.38 | 82.60 | 19.88 | 20.14 | 27.82 | 27.66 | 48.32 | 50.69 | 10.49 | 10.87 |
| Sum | 70,832 | 4,357 | 9,354 | 608 | 186.78 | 215 | 8,329 | 390 | 2,025 | 113 |
| Count | 1,808 | 125 | 5,417 | 344 | 138 | 138 | 324 | 18 | 1,506 | 72 |
| Coefficient of Variation | 0.40 | 0.62 | 1.00 | 1.52 | 3.04 | 2.73 | 0.41 | 0.73 | 1.23 | 1.68 |

**Table 14-13: Statistics of Block Grades Compared to Composite Grades by Domain – Gryphon**

| Statistic | Zone A1 | | Zone A2 | | Zone A3 | | Zone B1 | |
|---|---|---|---|---|---|---|---|---|
| | Comps | Blocks | Comp | Blocks | Comp | Blocks | Comp | Blocks |
| Count | 131 | 8,486 | 51 | 3,528 | 4 | 398 | 51 | 3,219 |
| Mean | 2.25 | 2.72 | 0.76 | 0.88 | 0.94 | 0.97 | 0.89 | 1.34 |
| Median | 0.38 | 1.51 | 0.16 | 0.52 | 0.67 | 0.99 | 0.18 | 0.87 |
| Std. Dev. | 4.53 | 2.89 | 1.32 | 0.92 | 0.96 | 0.22 | 2.34 | 1.31 |
| Variance | 20.52 | 8.34 | 1.74 | 0.84 | 0.92 | 0.05 | 5.46 | 1.71 |
| Kurtosis | 12.36 | 5.48 | 3.74 | 2.12 | -1.28 | 0.39 | 16.26 | 3.40 |
| Skewness | 3.26 | 2.00 | 2.16 | 1.59 | 0.55 | 0.46 | 4.10 | 1.68 |
| Range | 30.00 | 25.47 | 5.43 | 5.06 | 2.42 | 1.32 | 12.34 | 9.90 |
| Minimum | 0.00 | 0.05 | 0.00 | 0.05 | 0.00 | 0.47 | 0.00 | 0.05 |
| Maximum | 30.00 | 25.52 | 5.43 | 5.11 | 2.42 | 1.78 | 12.34 | 9.95 |
| Coef. of Var. | 2.02 | 1.06 | 1.74 | 1.04 | 1.02 | 0.23 | 2.64 | 0.98 |
| | **Zone B2** | | **Zone B3** | | **Zone C1** | | **Zone C2** | |
| | **Comp** | **Blocks** | **Comp** | **Blocks** | **Comps** | **Blocks** | **Comp** | **Blocks** |
| Count | 26 | 1,757 | 15 | 580 | 26 | 1,513 | 15 | 454 |
| Mean | 2.33 | 1.72 | 2.76 | 3.94 | 3.48 | 3.13 | 0.19 | 0.18 |
| Median | 0.33 | 0.74 | 0.44 | 2.18 | 0.56 | 1.38 | 0.01 | 0.10 |
| Std. Dev. | 3.85 | 2.03 | 4.45 | 4.17 | 6.70 | 4.14 | 0.35 | 0.21 |
| Variance | 14.81 | 4.14 | 19.84 | 17.43 | 44.90 | 17.13 | 0.12 | 0.04 |
| Kurtosis | 1.97 | 2.50 | 1.46 | -0.71 | 3.15 | 4.04 | 5.94 | 6.48 |
| Skewness | 1.81 | 1.75 | 1.67 | 0.87 | 2.16 | 2.09 | 2.56 | 2.66 |
| Range | 13.54 | 9.77 | 14.77 | 14.98 | 24.03 | 22.08 | 1.39 | 1.11 |
| Minimum | 0.00 | 0.05 | 0.00 | 0.11 | 0.00 | 0.07 | 0.00 | 0.05 |
| Maximum | 13.54 | 9.82 | 14.77 | 15.10 | 24.03 | 22.15 | 1.39 | 1.17 |
| Coef. of Var. | 1.65 | 1.18 | 1.61 | 1.06 | 1.92 | 1.32 | 1.83 | 1.15 |

### 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 metre 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 ID2 block model estimate.

## 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 US$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 14-14 and Figure 14-20 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 14-14: Phoenix Deposit Indicated Mineral Resource Sensitivity to Cut-Off Grade**

| Cut-off % $U_3O_8$ | Grade % $U_3O_8$ | Tonnes | Lb $U_3O_8$ Millions |
|---:|---:|---:|---:|
| 0.50 | 16.94 | 188,900 | 70.5 |
| 0.80 | 19.13 | 166,200 | 70.2 |
| 1.00 | 20.60 | 154,000 | 69.9 |
| 1.50 | 24.23 | 129,800 | 69.3 |
| 2.00 | 27.40 | 113,700 | 68.7 |
| 3.00 | 32.42 | 94,700 | 67.7 |
| 5.00 | 38.07 | 79,100 | 66.3 |

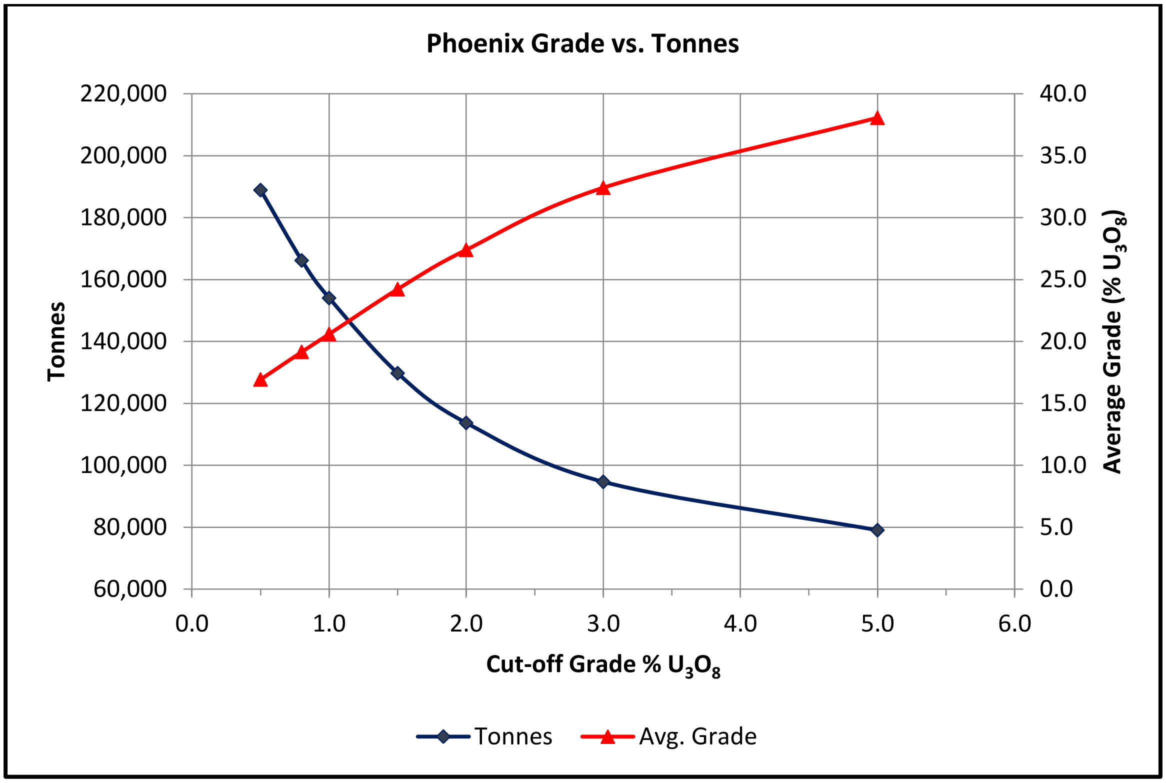



**Figure 14-20: Phoenix Indicated Mineral Resource Tonnes and Grade at Various Cut-Off Grades**

### 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 14-15 shows the breakeven cut-off grade estimate by RPA using a price of US$50/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 Cameco's Rabbit Lake mine, which is basement mineralization similar geologically to Gryphon.

**Table 14-15: Gryphon Deposit Cut-Off Grade Calculation**

| Item | Quantity |
|---|---|
| Price in US$/lb $U_3O_8$ | US$50 |
| Processing plant recovery | 90% |
| Operating cost per tonne | US$270 |
| Incremental operating cost component (75%) | US$200 |
| Cut-off grade | 0.2% |

Table 14-16 and Figure 14-21 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 14-16: Gryphon Deposit Inferred Mineral Resource Sensitivity to Cut-Off Grade**

| Cut-off % $U_3O_8$ | Grade % $U_3O_8$ | Tonnes (000) | Mlb $U_3O_8$ |
|---|---|---|---|
| 0.20 | 2.342 | 834 | 43 |
| 0.40 | 2.679 | 716 | 42 |
| 0.60 | 2.981 | 629 | 41 |
| 0.80 | 3.367 | 538 | 40 |
| 1.00 | 3.701 | 474 | 39 |

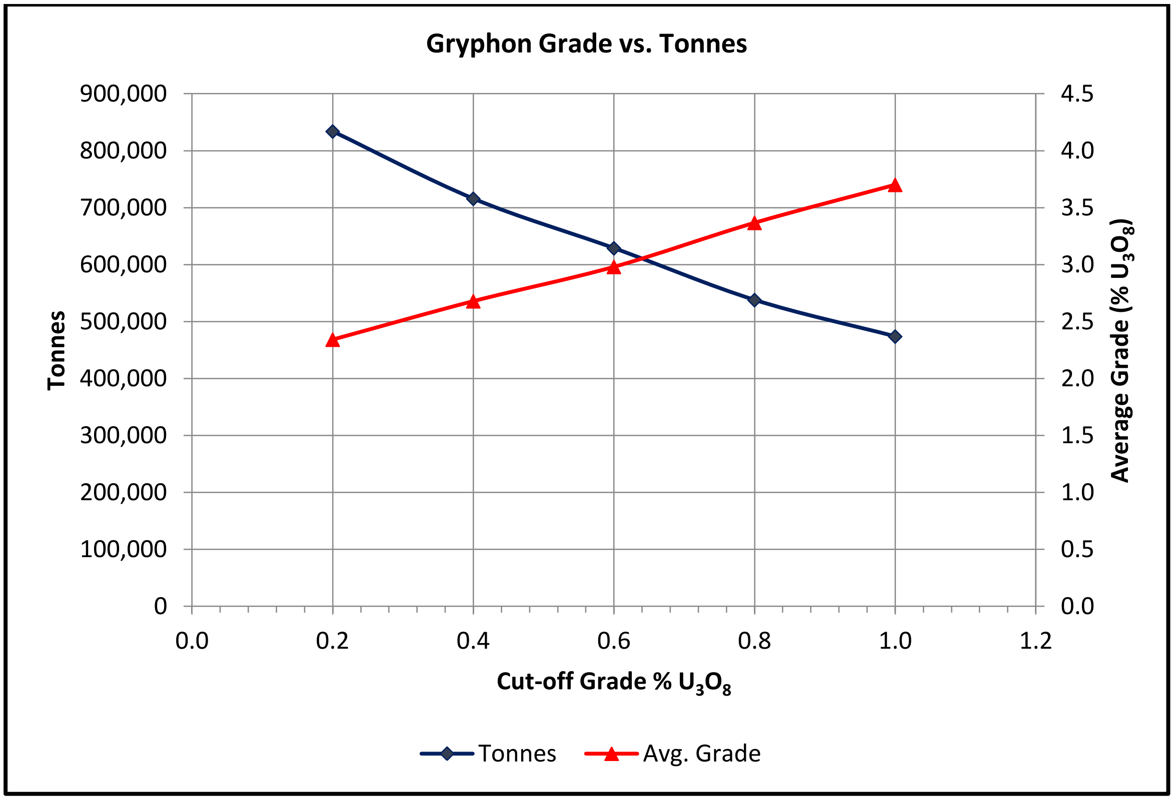



**Figure 14-21: Gryphon Inferred Mineral Resource Tonnes and Grade at Various Cut-Off Grades**

## 14.9 Classification

Definitions for resource categories used in this report are consistent with those in the CIM (2014) and adopted by NI 43-101. In 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 pre-feasibility or feasibility level as appropriate. Mineral reserves are classified into Proven and Probable categories. No mineral reserves have been estimated for the property.

### 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 14-22 shows the area of Inferred mineral resources along with Indicated mineral resources at Zone B.

### 14.9.2 Gryphon Deposit

The mineral resources for the Gryphon deposit are classified as Inferred based on drill hole spacing and apparent continuity of mineralization.

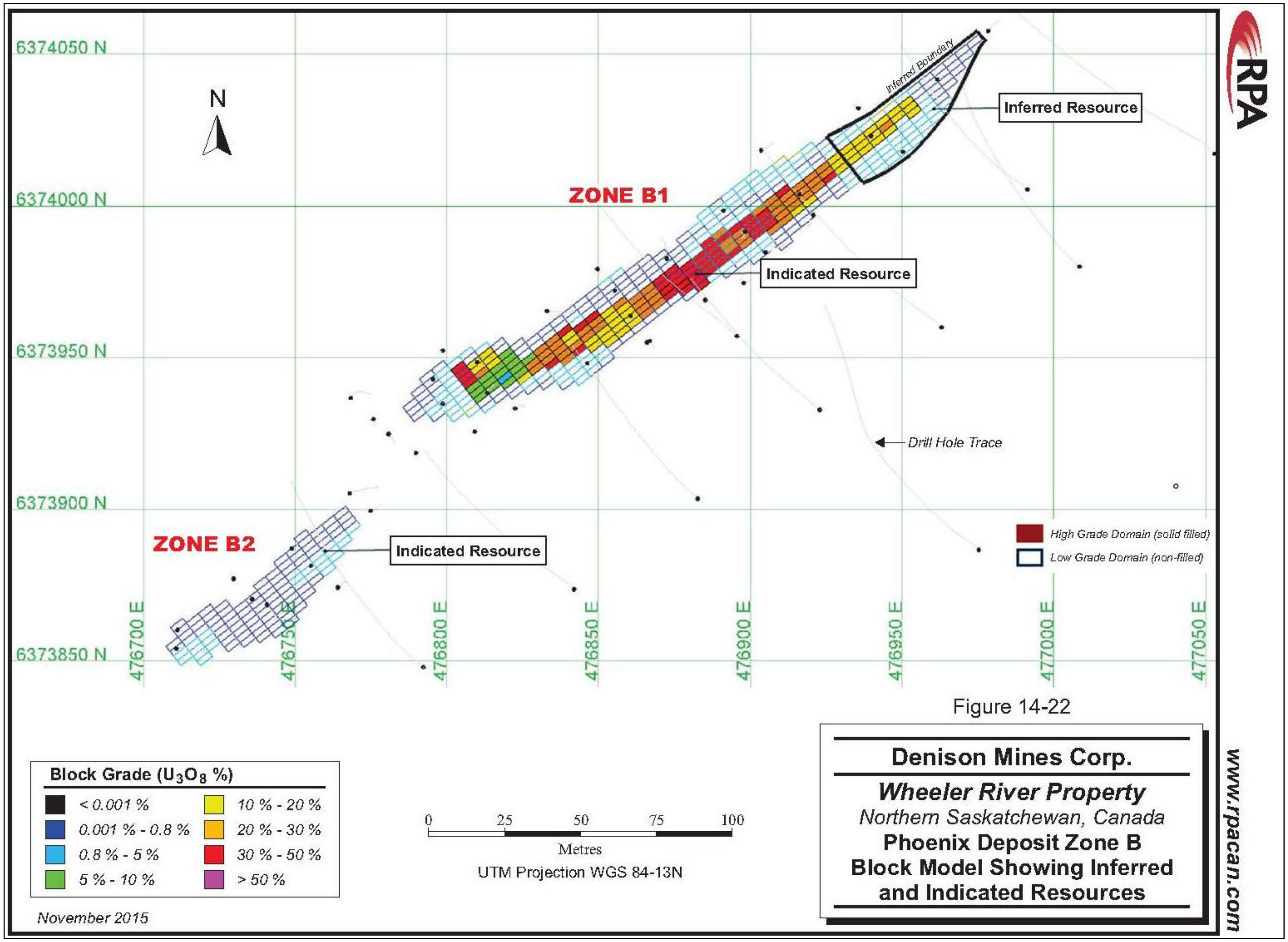



**Figure 14-22: Phoenix Deposit Zone B Block Model Showing Inferred and Indicated Resources**

## 14.10 Mineral Resource Estimate

Table 14-17 lists the mineral resource estimate for the Wheeler River property by domain and resource category. The effective date of the resource estimate is September 25, 2015. The Phoenix cut-off grade of 0.8% $U_3O_8$ is based on internal conceptual studies by Denison and a price of US$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 US50/lb $U_3O_8$.

For the Phoenix and Gryphon deposits, total Indicated mineral resources are estimated at 166,400 tonnes at an average grade of 19.13% $U_3O_8$ containing 70.2 million pounds of $U_3O_8$. Total Inferred mineral resources are estimated at 842,600 tonnes at an average grade of 2.37% $U_3O_8$ containing 44.1 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 14-17: RPA Mineral Resource Estimate - Wheeler River Project - September 25, 2015**

| Category | Deposit and Domain | Tonnes | Grade (% $U_3O_8$) | Contained Metal (million lb $U_3O_8$) |
|---|---|---|---|---|
| Indicated | Phoenix Zone A HG | 62,900 | 43.24 | 59.9 |
| Indicated | Phoenix Zone A LG | 84,300 | 2.37 | 4.4 |
| Indicated | Phoenix Zone B HG | 8,500 | 28.02 | 5.2 |
| Indicated | Phoenix Zone B LG | 10,700 | 2.91 | 0.7 |
| Subtotal Indicated | Phoenix Zone A | 147,200 | 19.81 | 64.3 |
| Subtotal Indicated | Phoenix Zone B | 19,200 | 13.94 | 5.9 |
| **Total Indicated** | | **166,400** | **19.13** | **70.2** |
| | | | | |
| Inferred | Phoenix Zone A HG | 0 | 0 | 0 |
| Inferred | Phoenix Zone B HG | 700 | 14.48 | 0.2 |
| Inferred | Phoenix Zone B LG | 4,800 | 1.79 | 0.2 |
| Inferred | Phoenix Zone A Basement | 3,100 | 10.24 | 0.7 |
| Subtotal Inferred | Phoenix Zone A | 0 | 0 | 0.0 |
| Subtotal Inferred | Phoenix Zone B | 5,500 | 3.30 | 0.4 |
| Subtotal Inferred | Phoenix Zone A Basement | 3,100 | 10.24 | 0.7 |
| **Subtotal Inferred** | **Phoenix Deposit** | **8,600** | **5.80** | **1.1** |
| | | | | |
| Inferred | Gryphon A1 | 387,200 | 2.89 | 24.6 |
| Inferred | Gryphon A2 | 125,200 | 1.10 | 3.0 |
| Inferred | Gryphon A3 | 18,100 | 0.97 | 0.4 |
| Inferred | Gryphon B1 | 137,500 | 1.43 | 4.3 |
| Inferred | Gryphon B2 | 73,300 | 1.90 | 3.1 |
| Inferred | Gryphon B3 | 19,000 | 5.72 | 2.4 |
| Inferred | Gryphon C1 | 69,700 | 3.33 | 5.1 |
| Inferred | Gryphon C2 | 3,900 | 0.54 | 0.1 |
| **Subtotal Inferred** | **Gryphon Deposit** | **834,000** | **2.37** | **43.0** |
| | | | | |
| **Total Inferred** | **Phoenix and Gryphon** | **842,600** | **2.37** | **44.1** |

Notes:
1. CIM definitions were followed for classification of mineral resources.
2. Mineral resources for Phoenix are reported above a cut-off grade of 0.8% $U_3O_8$, which is based on internal Denison studies and a price of US$50 per lb $U_3O_8$.
3. Mineral resources for Gryphon are reported above a cut-off grade of 0.2% $U_3O_8$, which is based on RPA assumptions and a price of US$50 per lb $U_3O_8$.
4. High grade composites are subjected to a high grade search restriction without capping at Phoenix.
5. High grade mineralization was capped at 30% with no search restrictions at Gryphon.
6. Bulk density is derived from grade using a formula based on 196 measurements at Phoenix and 65 measurements at Gryphon.
7. Numbers may not add due to rounding.

# 15 Mineral Reserve Estimates

This PEA does not support an estimate of mineral reserves.

# 16 Mining Methods

## 16.1 Hydrogeology

### 16.1.1 Regional Operating Mine Experience

Cameco's McArthur River and Cigar Lake operations have both experienced significant groundwater inflow events (two at McArthur River and three at Cigar Lake) associated with mine workings above the unconformity in Athabasca sandstone that have resulted in mine flooding and significant production and development delays. At least one major event was associated with ground failure.

### 16.1.2 Work Completed to Date and Findings

Hydrogeological investigations conducted at site to date include:

- Packer-based hydraulic testing of a single diamond drill hole at Phoenix (11 successful tests: 8 - sandstone, 2 - crystalline basement, 1 - unconformity), 1 water sample collected from lower Athabasca sandstone (Golder, 2014)
- Packer-based hydraulic testing of 12 diamond drill holes, 10 at Gryphon, 2 at Phoenix (39 successful tests: 6 - sandstone, 6 - unconformity, 23 - crystalline basement), open hole water level monitoring (SRK, 2015)

Data collected in 2015 by SRK appears to be consistent both with data collected by Golder in 2014 and with regional investigations of the Athabasca sandstone and underlying crystalline basement rocks.

Athabasca sandstone hydraulic conductivity at the Wheeler River project is largely in the moderate permeability range, moderately variable, with localized high permeability sections. There is little matrix permeability in these rocks, and highly fractured sandstone and conglomeratic sections tend to be highly permeable. The limited data indicate a potential reduction in permeability with depth in this unit (Figure 16-1).

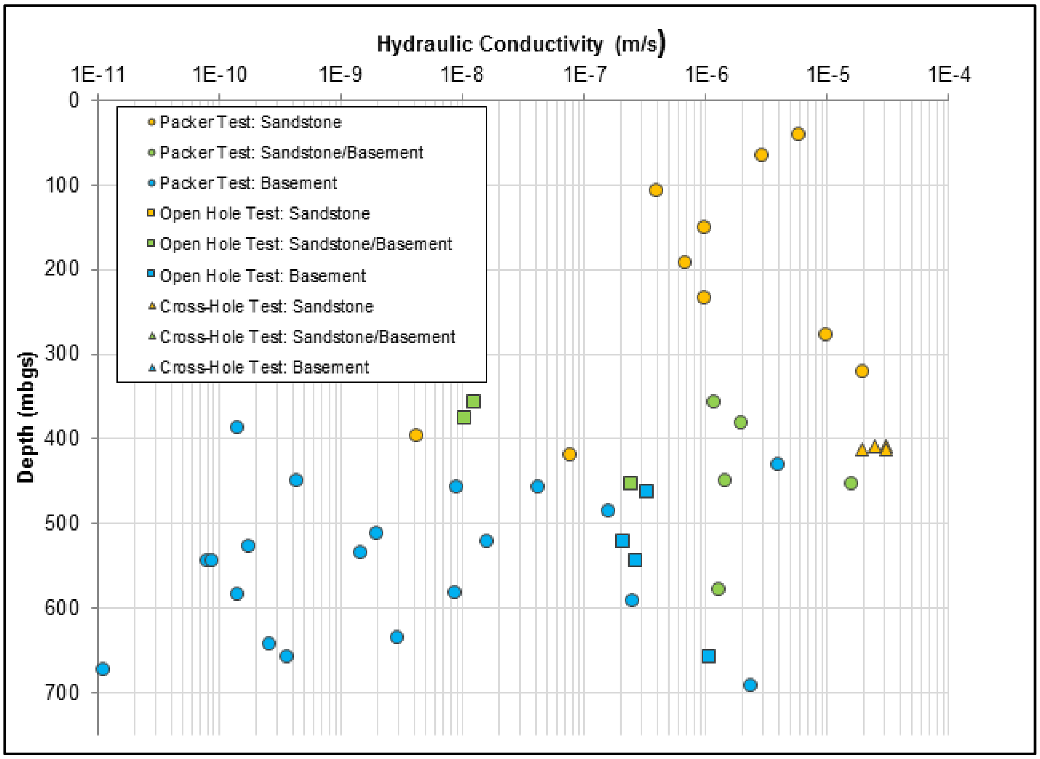



**Figure 16-1: Hydraulic Conductivity Data Compiled to Date, Wheeler River**

Crystalline basement rocks are highly variable in terms of bulk hydraulic conductivity, with a geometric mean hydraulic conductivity in the low permeability range. Hydraulic conductivity does not appear to decrease with increasing depth within the range of testing (370 m to 795 m depth) at Wheeler River to date. Significant sub-vertical structures may locally be highly permeable over widths of 10 m or more, as indicated by 2 of 25 tests conducted in the crystalline basement. Testing undertaken in 2015 indicated a permeable zone associated with the contact of pelitic gneisses with underlying quartzite, a feature also observed at McArthur River (Cameco, 2012). There does not appear to be a significant correlation between bulk hydraulic conductivity and average fracture frequency, however, testing at smaller scales (e.g., <10 m) could change this interpretation.

Groundwater quality data is limited to date, however, the single sample collected at Phoenix indicates lower Athabasca Formation water is fresh, of neutral pH, soft with elevated iron (2.5 mg/L), manganese (0.3 mg/L) and uranium (37 µg/L) contents together with detectable radioisotopes.

## 16.1.3 Groundwater Management

At Gryphon, where development is proposed in deeper basement rocks, pressures will be higher than those at McArthur River. There is potential of hydraulic connection of the crystalline basement mine workings with the overlying permeable, and largely un-depressurized Athabasca sandstone via sub-vertical joints or faults, or through open exploration holes, particularly where workings are located close (e.g., < 20 m) to the unconformity. This may be managed by:

- Maintaining development workings a minimum of 20 m below the unconformity, or greater where uncertainty exists with respect to the depth

- Collecting data, where possible, to evaluate the locations and hydraulic characteristics of potential structures
- Use of cover drilling and grouting in sensitive areas

The current practice of grouting exploration drill holes from the bottom up must be continued, particularly where these holes extend into the crystalline basement, as it is imperative that no additional pathways for water into the unit be created by poorly abandoned drill holes.

Other inflow risk mitigation measures employed by Cameco (2012) (for development located close to the unconformity) include:

- Use of probe and grout cover prior to excavation
- Utilization of freeze cover
- Plan for mine development to take place away from known groundwater sources
- Conservative ground support designs to reduce the risk of failures
- Minimization of opening size where practical
- Tight filling of mined-out areas to minimize ground movement
- Use of excavation methods that minimize ground damage and disturbance (e.g., mechanical development in high-risk areas)
- Use of third party experts as required when developing in high-risk areas

Groundwater pressures encountered at Phoenix are likely to be comparable to those at McArthur River. Cameco's standard for McArthur River is to maintain a pumping capacity equivalent to 1.5 times the maximum estimated sustained inflow. Dewatering requirements are reviewed once per year, and before working on any new zone (Cameco, 2012). No assessment of groundwater inflows has been conducted for Wheeler River project to date, however, it is considered likely that total maximum sustained routine inflows are likely to be less than 300 $m^3$/h (83 L/s), assuming the currently proposed mining methods and prudent groundwater management. However, much will depend on the mining and water management approach applied at Phoenix.

Future hydrogeological investigation should focus on identification and hydraulic testing of permeable structures, and should also be sufficiently flexible to accommodate changes in the current PEA mine plan.

## 16.2 Mine Geotechnical

A preliminary geotechnical evaluation has been completed to assess and characterize the rock mass conditions at the Phoenix and Gryphon deposits for the proposed underground mining. Input recommendations for mine design have been provided at a scoping level and recommendations have been made as to what will be required to move the geotechnical level of understanding to pre-feasibility and feasibility level of study.

### 16.2.1 Geotechnical Context

The critical geotechnical aspects that typically require consideration in the Athabasca Basin uranium deposits are:

- Proximity to the regional unconformity and potential for high pressure and large volumes of water associated with the Athabasca sandstone
- Rock mass conditions within and immediately adjacent to mineralized zone associated with de-silicification of Athabasca sandstone, and weakening clay alteration of basement rocks from mineralizing events
- Rock mass conditions in infrastructure areas

- Presence of major structures with potential for poor ground conditions or hydraulic connection to aquifers within the Athabasca sandstone

The general context of Phoenix and Gryphon deposits relative to the critical geotechnical aspects is illustrated in Figure 16-2.

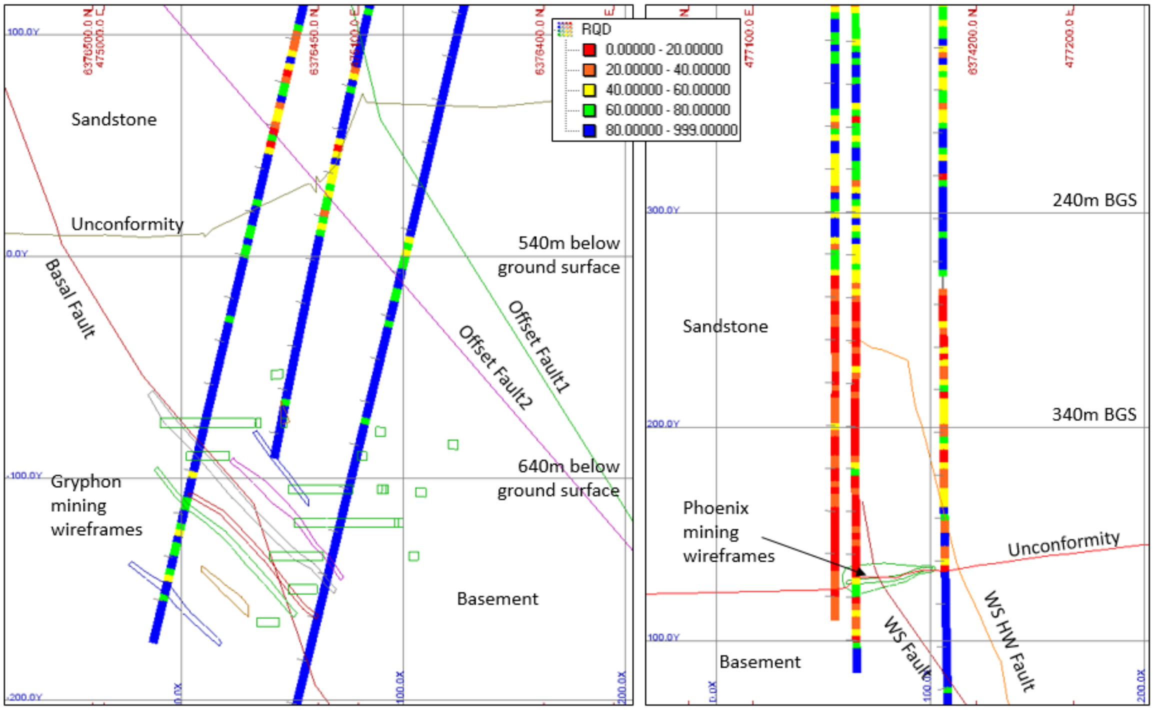



**Figure 16-2: Vertical Sections through Gryphon (left) and Phoenix (right) Deposits with Geotechnical Aspects Noted**

## 16.2.2 Data Availability

Rock geotechnical investigations and data collection completed include:

- Geotechnical data collected during routine exploration drilling by Denison based on the Bieniawski (1989) rock mass classification system
- Detailed geotechnical logging completed by Golder Associates of one drill hole at Phoenix (WR-555; Golder, 2014)
- Detailed geotechnical logging completed by SRK at Denison's core logging facility of one hole each at Phoenix and Gryphon (WR-605, WR-604 respectively; SRK, 2015)

A data set consisting of a Gemcom model is available with the following components (relevant to geotechnical evaluation):

- Wireframe models representing Phoenix and Gryphon mining shells
- Interpreted major structures at Phoenix and Gryphon
- Drill hole database with lithology and rock friability for 178 drill holes intersecting Phoenix mining wireframes and 31 intersecting Gryphon mining wireframes
- Drill hole database for all 178 holes with basic geotechnical data consisting of core recovery and rock quality designation
- Drill hole database for 77 holes at Phoenix and 15 holes at Gryphon with detailed geotechnical data collected using the Bieniawski rock mass classification system

- Core photos for all diamond drill holes in both the Phoenix and Gryphon zones

### 16.2.3 Large and Small Scale Structure

SRK has previously worked with Denison to review the structural setting of the deposits (SRK, 2014). Wireframe models of the main faults at Phoenix and Gryphon are illustrated in Figure 16-3. The faults at Phoenix are reasonably well constrained by drill hole data, while faults at Gryphon are less well constrained and will require review following the collection of additional drilling data.

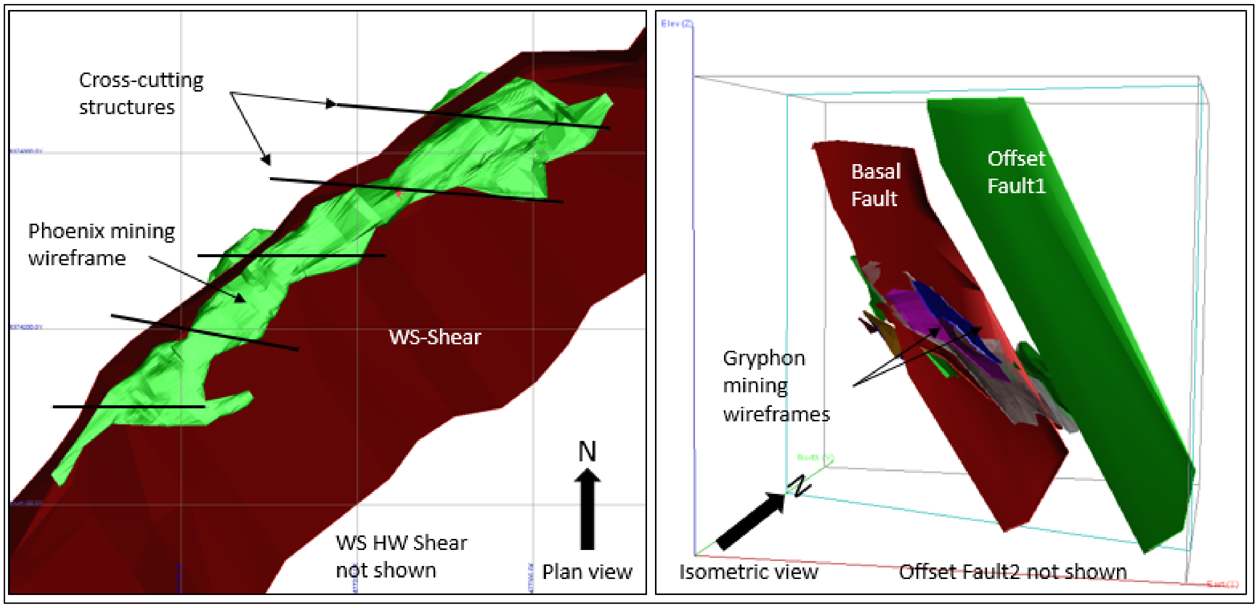



**Figure 16-3: Phoenix and Gryphon Structural Models**

The WS Shear at Phoenix is described by "deformation within the WS fault has occurred partly by ductile shearing, but mainly by fracturing." The shear dips at 55° to the south-east and cuts below the mineralized zone.

At Gryphon, the Offset and Basal Fault (or G Fault) are described as "a combination of cataclasites and gouges, and intervals of blocky and friable core". An example of rock mass conditions within the Offset Fault are shown in Figure 16-4.

Small scale structures for the basement below the unconformity have been identified from oriented core drilling completed by Denison. The primary joint set (60°/115°, dip/dip direction) is consistent with regionally persistent foliation. Three other pole clusters are present, conjugate to foliation (38°/310°), and two intermediate northeast and southwest dipping sets (40°/220° and 60°/060°, Figure 16-5).

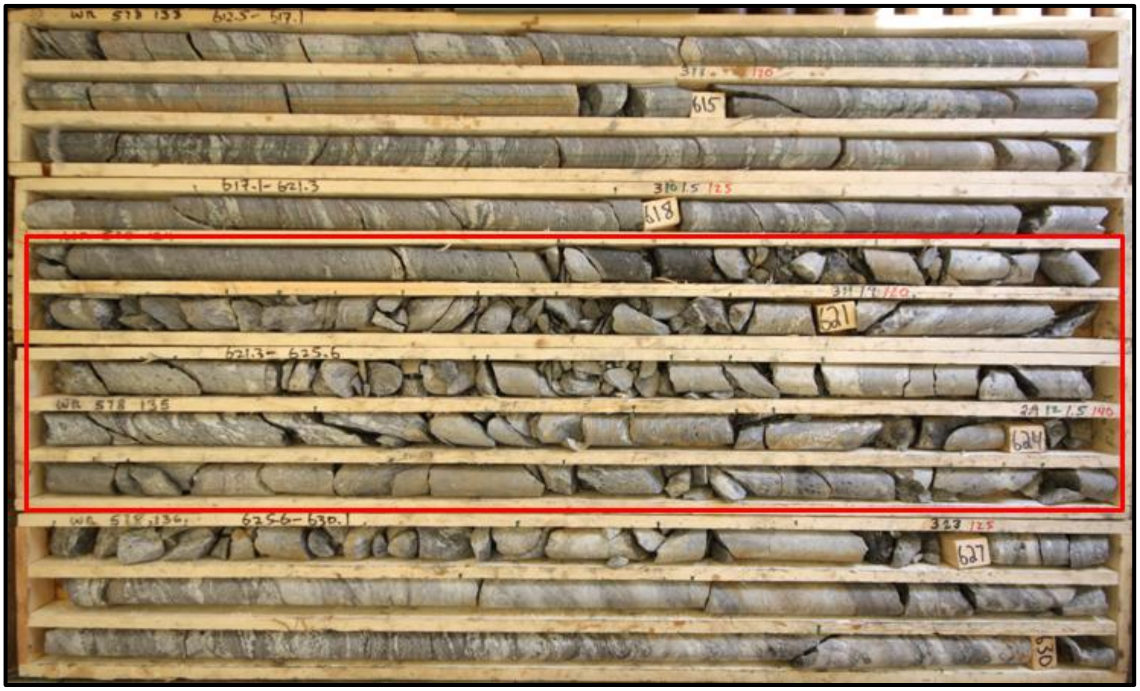


**Figure 16-4: Example of Rock Mass Condition in the Gryphon Offset Fault**
(indicated by red box)

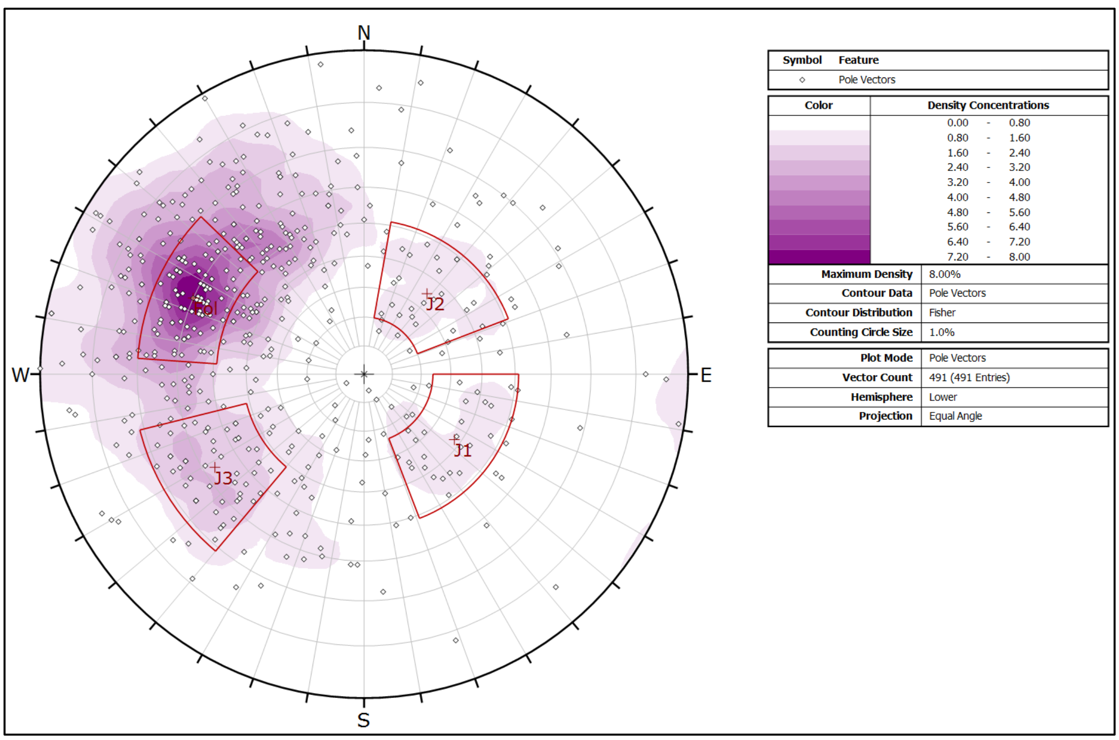



**Figure 16-5: Small Scale Structural Features in Basement Lithologies Collected by SRK from 2015 Diamond Drilling**

### 16.2.4 Rock Geotechnical

The following points summarize the findings of the geotechnical evaluation. Preliminary geotechnical domains have been developed and are described in the following section.

#### Phoenix

- Geotechnical data coverage at Phoenix is considered to be extensive, enabling a good 3D spatial assessment to be completed using the available down-hole parameters.
- Desilicification of the hangingwall (HW) rock mass has resulted in a significant zone of broken sandstone and unconsolidated sand intervals. Significant intervals of core loss are observed. The zone has 80 m to 250 m vertical thickness with a positive correlation to uranium grade.
- A Leapfrog Geo interpolant representing Rock Quality Designation (RQD) <60% has been modelled to illustrate the extent of poor HW rock mass conditions immediately above the mineralized zone (Broken Zone domain; Figure 16-6). The lateral extent of the Broken Zone domain is not well understood due the drilling being vertically oriented and largely focused immediately above the mineralization.
- Weakening clay alteration is pervasive throughout the mineralized zone. A good correlation is observed between poor ground conditions (with higher intensity alteration) and high uranium grade.
- Footwall (FW) basement rocks, except in close proximity to mineralization or where fault structures are present, are considered to be of good rock mass quality.

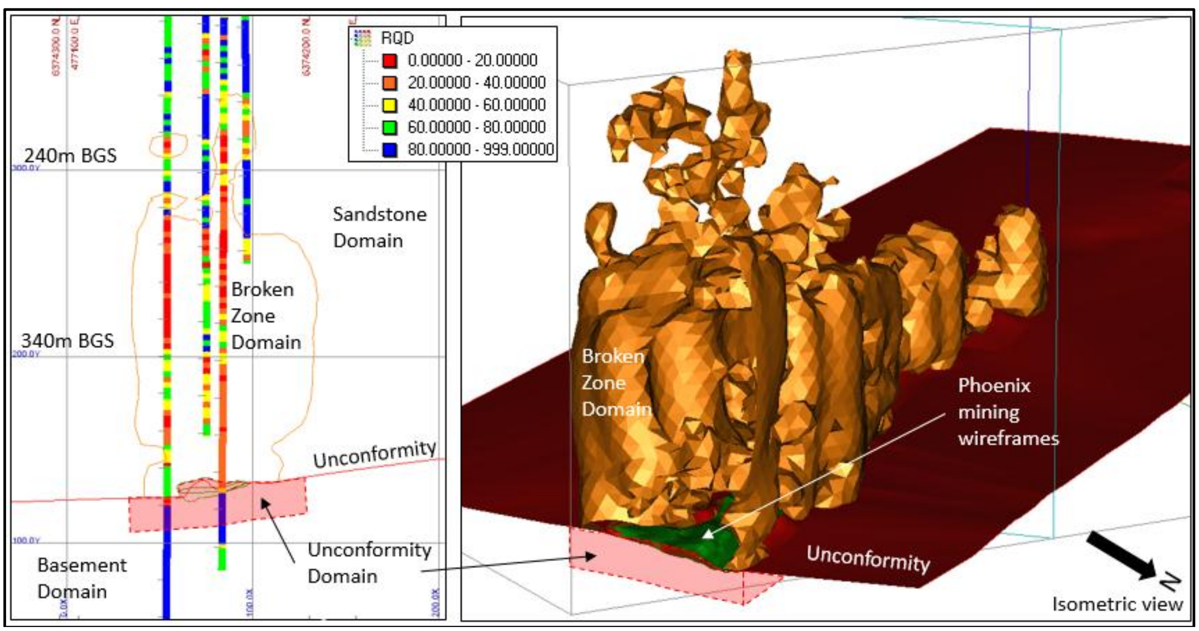



**Figure 16-6: Phoenix Conceptual Geotechnical Domains, Cross-section (left) and Isometric View**

### Gryphon

- Geotechnical data coverage is more limited consisting of approximately 31 holes intersecting the seven primary mineralized lenses (Figure 16-7).
- As the mineralization is entirely within the basement rocks, high water pressures and geotechnical issues usually associated with unconformity deposits are not considered critical geotechnical factors at Gryphon. Fault structures and open joints still represent a risk of water inflows.
- Although clay alteration is present, in general due to the lower grade mineralization the clay alteration intensity is also lower.
- Localized zones of reduced rock mass quality are associated with higher grade mineralized zones, and the modelled Basal and Offset fault structures.
- Overall, rock mass conditions are of fair to good rock mass quality throughout the HW, mineralized veins, and FW package.

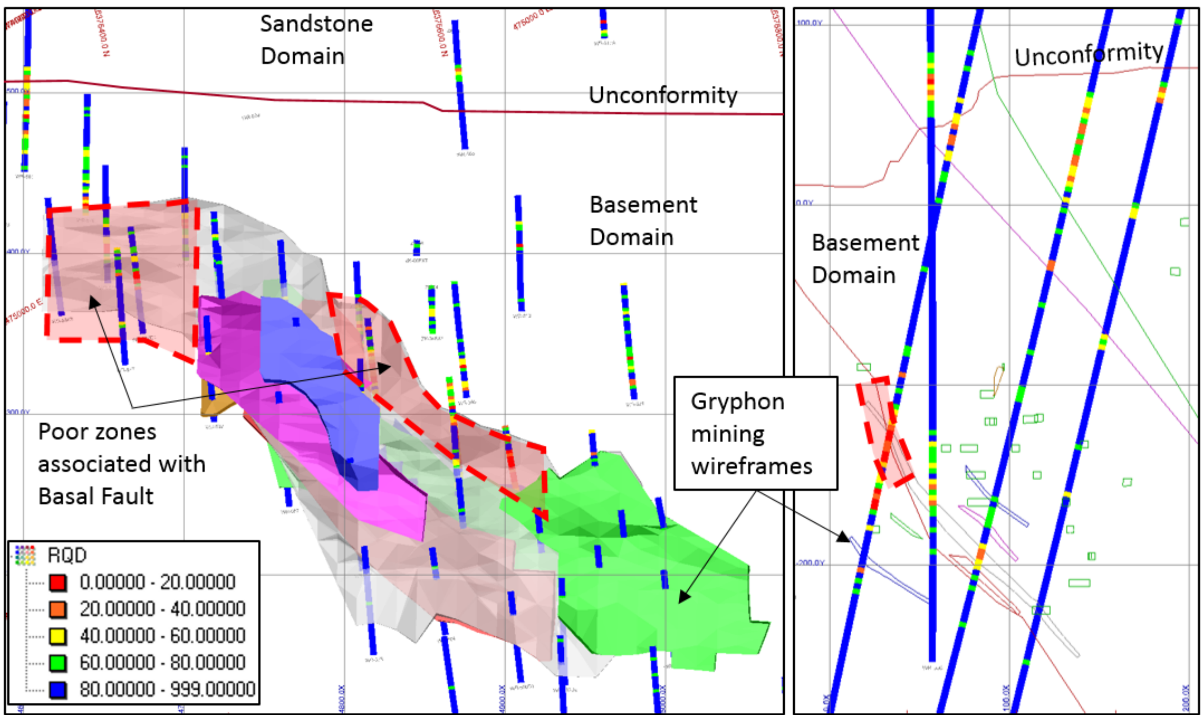



**Figure 16-7: Gryphon Conceptual Geotechnical Domains, Isometric View (left) and Cross-section**

## 16.2.5 Preliminary Geotechnical Domains

The following section provides a description of the conceptual geotechnical domains established for each mining area, as shown in Figure 16-6 and Figure 16-7.

## Sandstone Domain (Phoenix and Gryphon)

Above the regional unconformity, the Sandstone domain contains the sandstone/conglomerate units of generally fair to good rock mass quality, with intact rock strength estimated at 80 to 120 MPa (Figure 16-8). Weaker and more friable/de-silicified zones should be expected in close proximity to major structures and the unconformity. Shaft sinking is proposed through the sandstone in the HW to Gryphon mineralization. A thorough assessment of geotechnical conditions within the Sandstone domain should be completed for all areas proximal to planned development.

**Figure 16-8: Fair to Good Rock Mass Conditions in the Sandstone Domain**

## Broken Zone Domain (Phoenix)

Within the Sandstone domain, the Broken Zone domain contains the volume of poor quality sandstone and desilicified loose sand in the immediate HW to the Phoenix mineralized zone (Figure 16-9). This zone extends vertically above the unconformity, and is up to 80 m vertical thickness above the Phoenix Zone B, and up to 250 m above the Phoenix Zone A. This domain is of poor to very poor rock mass quality.

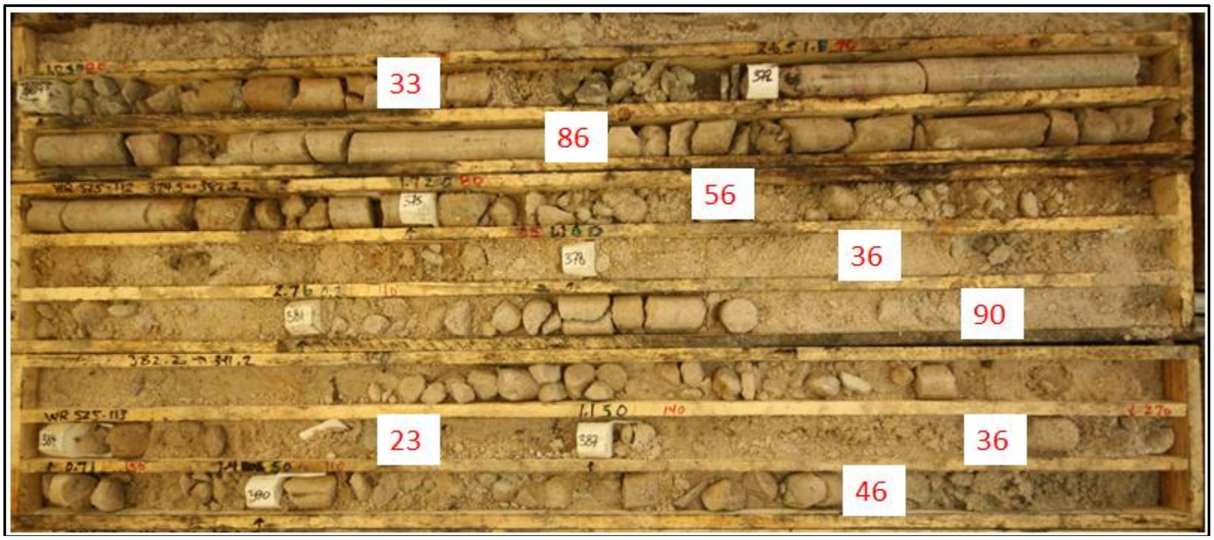



**Figure 16-9: Poor to Very Poor Rock Mass Conditions in the Broken Zone Domain (% Core Recovery Shown)**

## Unconformity Domain (Phoenix)

The Unconformity domain (Figure 16-10) encompasses a zone of ground within and approximately 20 m around mineralized zones where ground conditions are interpreted to exhibit a wider variability compared to the Basement domain. An increased frequency of core loss, elevated clay alteration intensity, and rubble zones can be observed. Poorer ground conditions are correlated with higher $U_3O_8$ grade.

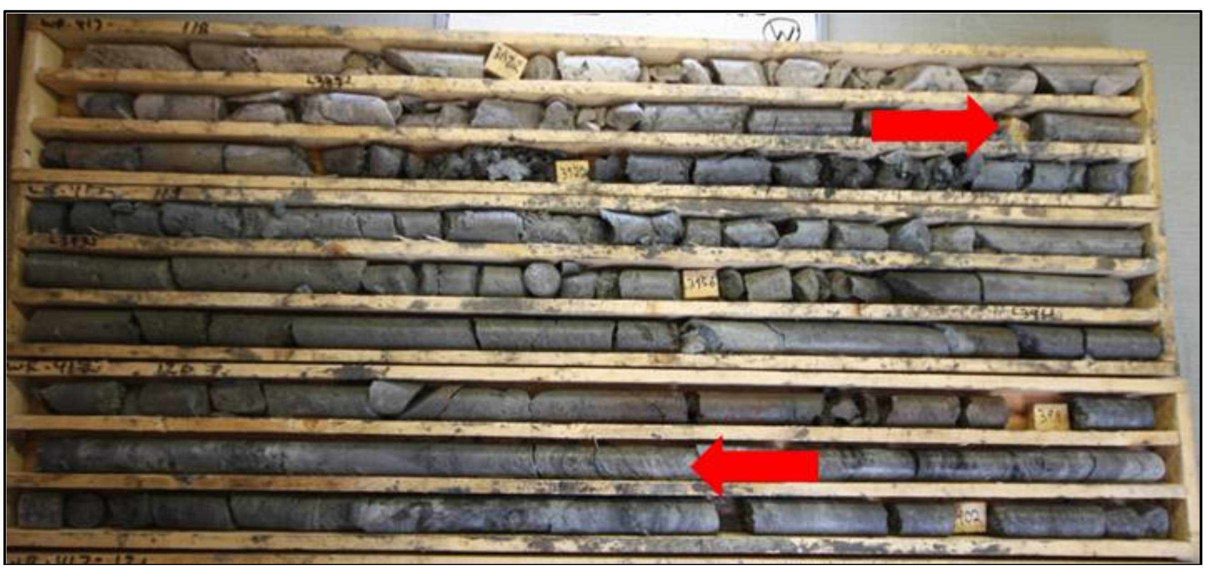


**Figure 16-10: Poor to Fair Rock Mass Conditions in the Unconformity Domain**
(arrows indicate Phoenix mineralized zone)

**Basement Domain (Phoenix and Gryphon)**

The Basement domain (Figure 16-11) encompasses the bulk of the basement lithologies. Similar to the Sandstone domain, low variability is expected with predominantly Fair to Good rock mass conditions and intact rock strength in the range 80 to 150 MPa. Reduced rock mass conditions, similar to those observed in the Unconformity domain, are observed around fault structures in basement rocks.

**Figure 16-11: Fair Rock Mass Conditions in the Basement Domain (Gryphon Mineralization)**

### 16.2.6 Excavation Design

This report section provides comments on excavation design considering the prevailing geotechnical conditions.

#### Phoenix

- Lateral or vertical development is not recommended within the Broken Zone domain due to the prevailing ground conditions and anticipated high water pressures.
- Frozen ground conditions will be required to provide strength enhancement in the Broken Zone domain to enable the opening of non-man entry mining excavations.
- Even with frozen ground conditions, it is likely that instability will occur in some areas when opening spans greater than 4 m diameter. Allowances for mineralized dilution and additional backfill volumes are required.
- Infrastructure developed below the mineralized zone should have a minimum vertical stand-off distance of 15 m to avoid poor rock mass conditions. Along the middle of the mining zone where mineralization is thicker, a stand-off of 30 m should be maintained (Figure 16-12). It is estimated that 20% of the development immediately below the mineralization will require enhanced ground support consisting of 100 mm of shotcrete (in addition to standard pattern support) to prevent squeezing due to freeze development.
- Within the Basement domain it is recommended that development should be completed under continuous probe and cover drilling along the main development headings. Probe and cover

drilling can assist with mitigating the risks associated with fault structures and open fractures potentially connected to water in the Sandstone domain.

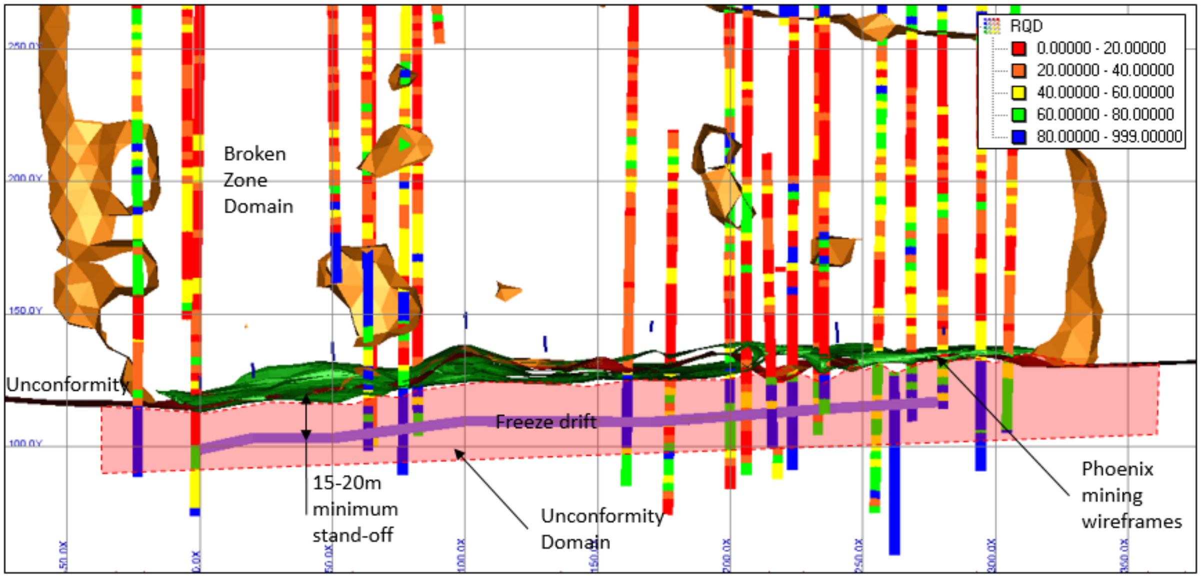



**Figure 16-12: Long Section along Phoenix Freeze Drift showing Expected Rock Mass Conditions**

### Gryphon

- The rock mass conditions at Gryphon are suitable for open stoping.
- Man-entry spans have been reviewed based on the critical span curve presented in Ouchi et al (2004). Based on this empirical graph stable spans of up to 10 m are considered reasonable for the fair to good conditions in the Basement domain. A maximum span of 5 m is recommended in poor ground conditions associated with faults or clay alteration.
- Non man-entry spans (for open stoping) have been established using the modified Matthew’s stability curve after Potvin and Hadjigeorgiou (1998; further modified by Trueman, 2000). A stable stope longitudinal geometry of 15-20 m vertical height, 15 m strike length, and 10 m width is recommended.
- Where multiple veins may be mined in a package, wide spans and intersections greater than 7 m inscribed circle will require long secondary support according to the one-third to one-half span rule of thumb.
- Within the Basement domain it is recommended that development should be completed under continuous probe and cover drilling along the main development headings. Probe and cover drilling can assist with mitigating the risks associated with fault structures and open fractures potentially connected to water in the Sandstone domain.

## 16.2.7 Ground Support

The recommended support requirements for the production excavations are based on using the Q Support Chart of Grimstad, Barton, and Loset (1993) and practical experience. The Q Support Chart estimated values are tempered by the fact that the jointing and foliation orientations are not considered in the empirical guidelines (Figure 16-13).

- For all lateral development (including mineralization development at Gryphon), the recommended primary support is 2.1 m fully grouted resin rebar on a 1.2 m by 1.2 m spacing with #7 gauge weld wire mesh.
- Additional plain shotcrete (75-100 mm) will be required in weaker zones. This includes fault structures, adverse rock mass conditions associated with clay alteration and weaker areas at Phoenix where freezing could impact tunnel stability.
- Spacing of rebar should be reduced based on ground conditions at the face. In particularly weak areas such as fault intersections, spilling and additional application of early shotcrete may be required.
- Wide spans and intersections greater than 7 m inscribed circle should be supported with long secondary support anchors according to the one-third to one-half span rule of thumb.

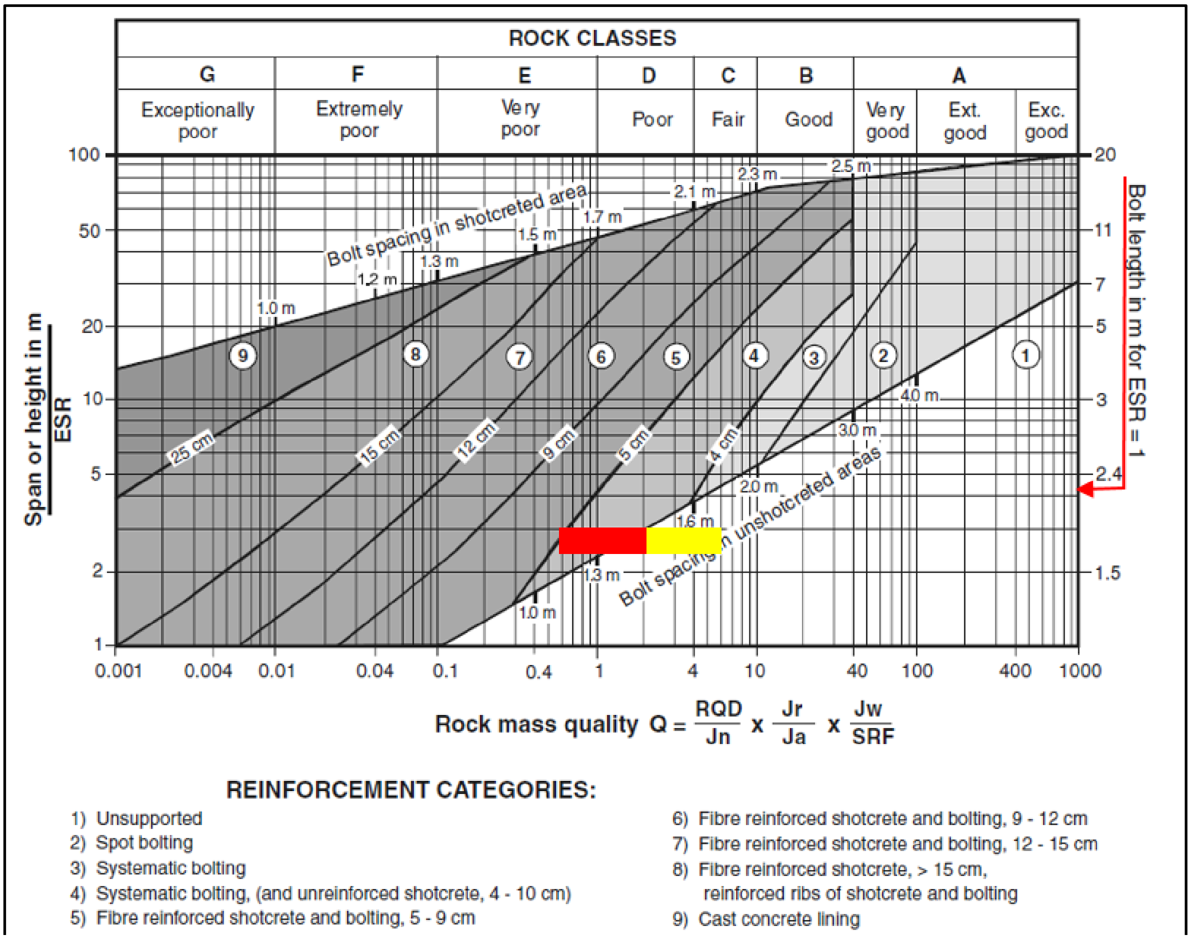



**Figure 16-13: Recommended Support Requirements for the Production Excavations (Q Support Chart of Grimstad and Barton, 1993), ESR=1.6.**

## 16.3 Mineral Resources within PEA Design Plan

This mining study is based on the Indicated and Inferred mineral resources of the Phoenix and Gryphon deposits as disclosed November 25, 2015 by Roscoe Postle Associates Inc. in their National Instrument 43-101 report, "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada" (RPA, 2015) The effective date of the resource estimate is September 25, 2015. Refer to Table 14-17.

SRK followed the steps listed below in estimating the mineral resources to be included in the PEA design plan.

- Mining methods were selected
- A base case uranium price of US$44.00 per pound, and an exchange rate of 1.35 CAD/USD were selected for mine planning
- A cut-off grade of 0.4% $U_3O_8$ was estimated for the longhole mining method selected for Gryphon
- Jet bore cut-off criteria was based on a combination of grade and vertical thickness
- Mineralization wireframes were evaluated at zero cut-off grade for underground mining
- Some wireframes were clipped to remove low grade areas using the established cut-off grades as a guide
- The final wireframes were evaluated in Gemcom to determine in situ tonnes and uranium grades within the wireframes at a zero cut-off grade
- Factors for external dilution and mining recovery were applied

SRK notes that this PEA mining study is preliminary in nature. The "mineral resources within PEA design plan" (MR within PEA) disclosed in the mine plans include a portion of Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the results of this study will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

### 16.3.1 Phoenix Deposit

For the Phoenix deposit, mining shapes suitable for jet boring were created for Zone A, Zone A Basement, and Zone B1. Zone B2 is small and of lower grade and cannot support the cost of jet bore mining.

Zones A and B1 resource wireframes both include a high grade (HG) core surrounded by lower grade (LG) mineralization. These designations were established for the purpose of mineral resource estimation, not for mine production planning. For both Zones A and B1, parts of the LG mineralization have been clipped off and excluded from jet bore mining. A cut-off grade of 2% $U_3O_8$ was used as a guide, similar to the cut-off grade at Cigar Lake mine. The clipped areas are generally on the fringes of the LG solids, and are thinner and/or of lower grade. Entire HG resource wireframes have been designated for jet bore mining.

Figure 16-14 shows how the Phoenix Zone A LG resource wireframe was trimmed to clip off outer portions of lower grade (dark blue). The central corridor is the high grade jet boring target area (cyan).

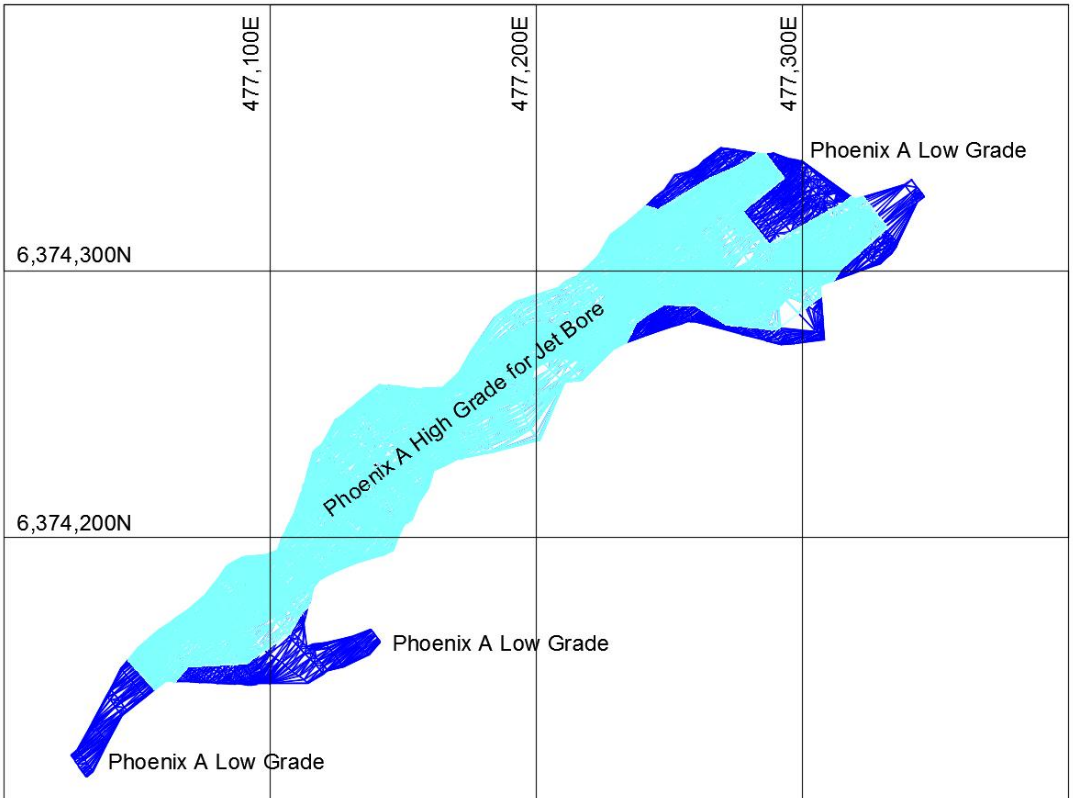



**Figure 16-14: Phoenix Zone A Plan View – Low Grade Fringe on Mining Shape**

Figure 16-15 shows how the Phoenix Zone B1 LG resource wireframe was trimmed to clip off outer portions of lower grade (black). The central corridor is the high grade jet boring target area (red).

The Phoenix Zone A Basement resource wireframe was clipped to exclude a very low grade area from the jet bore mine plan. The low grade area to be clipped off was determined by inspection at the point of a dramatic drop in grade from less than 1% $U_3O_8$ to high grade material ranging from 3% to over 20% $U_3O_8$.

Table 16-1 shows the conversion of in situ Indicated mineral resources to MR within PEA for the Phoenix deposit.

Table 16-2 shows the conversion of in situ Inferred mineral resources to MR within PEA for the Phoenix deposit.

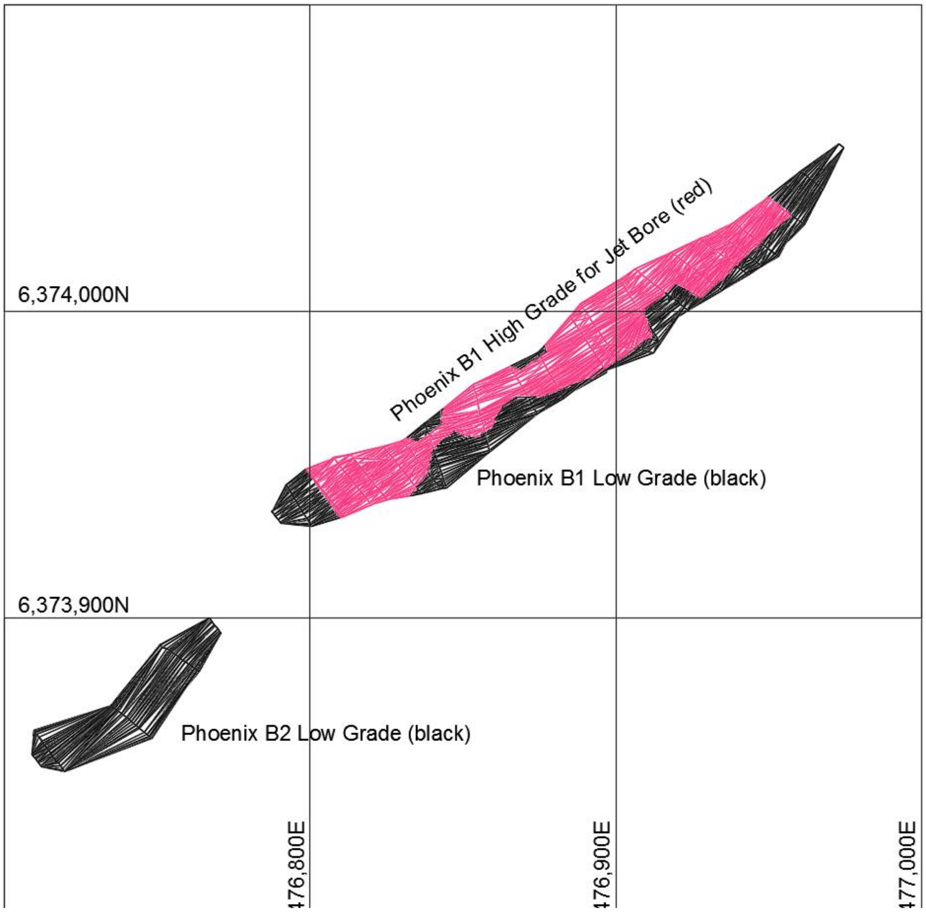



**Figure 16-15: Phoenix Zones B1/B2 Plan View – High Grade and Low Grade Areas**

**Table 16-1: Phoenix – Conversion of Indicated Mineral Resources to MR within PEA**

| | | | | | Indicated Classification | | |
|---|---|---|---|---|---|---|---|
| **Zone** | **Type** | **Mining Method** | **Mining Wireframe** | **Cut-off % $U_3O_8$** | **Kilo-tonnes** | **Grade % $U_3O_8$** | **Mlbs $U_3O_8$** |
| **A** | in situ | JBS | High Grade -full | 0.0% | 62.8 | 43.2 | 59.9 |
| | in situ | JBS | Low Grade - clipped | 0.0% | 109 | 1.85 | 4.4 |
| **B1** | in situ | JBS | High Grade -full | 0.0% | 8.47 | 27.0 | 5.04 |
| | in situ | JBS | Low Grade - clipped | 0.0% | 18.3 | 1.57 | 0.63 |
| | **in situ** | **JBS** | **All JBS wireframes** | **0.0%** | **198** | **16.0** | **70.0** |
| | | | **Conversion** | **Factor** | **Kilo-tonnes** | **Grade % $U_3O_8$** | **Mlbs $U_3O_8$** |
| | | | External Dilution | 30% | 59.5 | 0.0 | 0.0 |
| | | | Diluted Tonnes | | 258 | 12.3 | 70.0 |
| | | | Mining Recovery | 90% | | | |
| | | **Jet Bore** | **MR within PEA** | | **232** | **12.3** | **63.0** |

**Table 16-2: Phoenix – Conversion of Inferred Mineral Resources to MR within PEA**

| | | | | | Inferred Classification | | |
|---|---|---|---|---|---|---|---|
| Zone | Type | Mining Method | Mining Wireframe | Cut-off % $U_3O_8$ | Kilo-tonnes | Grade % $U_3O_8$ | Mlbs $U_3O_8$ |
| **A** | in situ | JBS | A basement - clipped | 0.0% | 3.65 | 8.41 | 0.68 |
| **B1** | in situ | JBS | High Grade -full | 0.0% | 0.74 | 27.0 | 0.44 |
| | in situ | JBS | Low Grade - clipped | 0.0% | 2.26 | 1.57 | 0.08 |
| | **in situ** | **JBS** | **All JBS wireframes** | **0.0%** | **6.64** | **8.15** | **1.19** |
| | | | **Conversion** | **Factor** | **Kilo-tonnes** | **Grade % $U_3O_8$** | **Mlbs $U_3O_8$** |
| | | | External Dilution | 30% | 1.99 | 0.0 | 0.0 |
| | | | Diluted Tonnes | | 8.63 | 6.27 | 1.19 |
| | | | Mining Recovery | 90% | | | |
| | | **Jet Bore** | **MR within PEA** | | **7.77** | **6.3** | **1.07** |

The total potential mill feed from the Phoenix deposit is estimated at 240 kt at an average grade of 12.1% $U_3O_8$ containing 64.1 Mlb of $U_3O_8$.

The jet bore external dilution and mining recovery factors are based on the publicly disclosed reserve parameters used at the Cigar Lake high grade uranium project in northern Saskatchewan where a jet bore mining method is used.

### SRK Comment

Areas of low grade mineral resources excluded from the mine plan are shown in Figure 16-14 (dark blue) and Figure 16-15 (black). This material has been excluded from the PEA mine plan due to the challenges related to mining these areas:

- These are small mining targets distributed over a large area and will require extensive ramp and lateral development to access and ventilate.
- These areas are close to the freeze wall, requiring drilling and blasting within a 5 m distance. This would increase the risk of freeze hole damage and mine flooding.
- Ground conditions would be variable, and difficult in proximity to the HW.
- The freeze wall would have to be extended along strike to recover portions of this material. This would be expensive.

## 16.3.2 Gryphon Deposit

An economic break even mine planning cut-off grade of 0.4% $U_3O_8$ was estimated for Gryphon by SRK based on a base case uranium price of US$44.00 per pound, and an exchange rate of 1.35 CAD/USD. A mining cost of $144 per tonne was estimated based on longhole mining at 400 tonnes per day. Other operating cost inputs to the cut-off grade estimate included $68 per tonne for surface plant feed transportation, a processing cost of $82 per tonne for processing at cut-off grade, and a G&A operating cost of $174 per tonne. A process uranium recovery of 97% was assumed.

Gryphon Inferred mineral resources estimated by RPA are comprised of geologic wireframes reported at a cut-off grade of 0.2% $U_3O_8$.

The geologic wireframes were interrogated at a zero cut-off grade and at a 0.4% $U_3O_8$ cut-off grade, to determine the percentage of internal dilution within each geologic wireframe. Internal dilution was found to range from 0% to 90%, averaging 23%.

Each geologic wireframe was examined to determine if low grade portions could be practically clipped off to improve the mining grade. Material grading less than a 0.4% $U_3O_8$ was targeted for clipping. Of the eight geologic wireframes, SRK was able to clip six.

Clipped wireframes were created to remove very low grade areas from the mine plan. This was done by viewing each of the lenses in a vertical long section view showing the distribution of $U_3O_8$ grades. Lower grade portions of the geologic wireframes below a cut-off grade of 0.4% $U_3O_8$ were identified and polylines were drawn to act as clipping boundaries. Clipped wireframes were then created that excluded the low grade portions of the lenses. The overall results of the clipping were a grade increase of 11% with a uranium metal loss of only 1%. Total mineralized material clipped off was 105,000 tonnes at 0.19% $U_3O_8$.

The six clipped and two whole geologic wireframes (now referred to as mining wireframes) were interrogated at zero cut-off grade to determine the in situ tonnes and grades contained within them. The true in-situ thickness of each mining wireframe was calculated to provide a guide for estimating external dilution.

The planned mining method for Gryphon is conventional longhole stoping with backfill. External dilution averaging 23% and a mining recovery factor of 95% were based on this mining method being applied to the moderately dipping lenses at Gryphon. These factors were applied to the in situ tonnes of the mining wireframes to obtain an estimate of MR within PEA.

Table 16-3 shows the conversion of in situ Inferred mineral resources to MR within PEA for the Gryphon deposit.

**Table 16-3: Gryphon – Conversion of Inferred Mineral Resources to MR within PEA**

| Gryphon Type | Mining Method | Mining Wireframe | Cut-off % $U_3O_8$ | Inferred Classification Kilo-tonnes | Grade % $U_3O_8$ | Mlbs $U_3O_8$ |
|---|---|---|---|---|---|---|
| in situ | longhole | A1_1 | 0.0% | 394 | 2.84 | 24.7 |
| in situ | longhole | C1_1 CLIP | 0.0% | 65.7 | 3.51 | 5.09 |
| in situ | longhole | B1_1 CLIP | 0.0% | 117 | 1.62 | 4.18 |
| in situ | longhole | A2_1 CLIP parts A & B | 0.0% | 137 | 1.01 | 3.06 |
| in situ | longhole | B2_1 CLIP | 0.0% | 71.9 | 1.92 | 3.05 |
| in situ | longhole | B3_1 CLIP | 0.0% | 24.5 | 4.47 | 2.42 |
| in situ | longhole | A3_1 | 0.0% | 18.1 | 0.97 | 0.39 |
| in situ | longhole | C2_1 CLIP | 0.0% | 3.13 | 0.61 | 0.04 |
| **in situ** | **longhole** | **Total** | **0.0%** | **831** | **2.34** | **42.9** |
| | | **Conversion** | **Factor** | **Kilo-tonnes** | **Grade % $U_3O_8$** | **Mlbs $U_3O_8$** |
| | | External Dilution | 23% | 195 | 0.0 | 0.0 |
| | | Diluted Tonnes | | 1,026 | 1.90 | 42.9 |
| | | Mining Recovery | 95% | | | |
| | **longhole** | **MR within PEA** | | **975** | **1.90** | **40.7** |

The total potential mill feed from the Gryphon deposit is estimated at 975 kt at an average grade of 1.90% $U_3O_8$ containing 40.7 Mlb of $U_3O_8$.

## 16.3.3 Wheeler River Project Potential Mill Feed

The total potential mill feed for the Wheeler River project is estimated at 1.22 Mt containing 105 Mlb of $U_3O_8$.

SRK notes that this PEA mining study is preliminary in nature. The MR within PEA include a portion of Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

# 16.4 Mining Context

## 16.4.1 Phoenix

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 to 50 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, over the 700 m strike length of Zones A and B1, the deposit gradient is only -3% to the northeast
- The deposit sits on the unconformity at a nominal 420 m depth, and is subject to the high water pressures in the overlying sandstone
- It is a very high grade, high value deposit requiring a high mining recovery
- Due to the high uranium grade, special mining methods will be required to minimize the exposure of workers to radiation
- 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
- Development in basement rock, sufficiently below the unconformity and associated alteration or palaeoweathering, will encounter fair to good rock mass conditions with some risk of encountering fault structures
- Mineralization/waste contacts are easily visible
- Mineralization continuity is excellent at mining cut-off grades

## 16.4.2 Gryphon

The relevant characteristics of the Gryphon zone 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 drilled at a nominal 50 m spacing
- The zone is comprised of stacked lenses, dipping approximately 50 degrees
- Average true thickness of individual resource wireframes ranges from 1.8 to 4.0 m, averaging about 3 m
- In some areas, the lenses are close enough together to be impacted by adjacent lens mining
- The deposit is located well below the unconformity and high-pressure groundwater will not likely be encountered
- The moderate grade of the mineralization will not require special mining methods; conventional mining methods can be used
- Geotechnical assessment indicates generally fair to good rock mass conditions
- Mineralization/waste contacts are easily visible
- Mineralization continuity is good at mining cut-off grades

# 16.5 Mining Methods

## 16.5.1 Introduction

Mining methods introduced in previous report sections are described in this section. The selected underground mining methods for the Wheeler River project are:

- Jet bore system (JBS) mining for the Phoenix Zones A and B1. This is planned to be done using freeze wall protection.
- Conventional longhole open stoping with backfill is planned for the Gryphon deposit. No freeze wall protection is needed.

Table 16-4 shows the relative significance of each planned mining method.

**Table 16-4: Relative Distribution of Mining Methods**

| Mining Method | Deposit | Mining Method Distribution by Tonnes | by lbs $U_3O_8$ |
|---|---|---|---|
| Jet Bore System | Phoenix | 20% | 61% |
| Longhole Stoping | Gryphon | 80% | 39% |
| **Total** | | 100% | 100% |

## 16.5.2 Phoenix – Jet Bore System

### Introduction

The Phoenix Zones A and B1 planned mining method is similar to the method currently used at the Cigar Lake uranium mine in northern Saskatchewan. SRK has used publicly available information to assess the method for application at the Phoenix deposit. Comparing the Cigar Lake mine to the Phoenix deposit, SRK notes that:

- The vertical mineralization thickness is similar, 5 to 7 m at Cigar Lake compared to averages of 6.3 m at Zone A and 5.0 m at Zone B1.
- Cigar Lake reserve grade is 17.8% $U_3O_8$ compared to a planned 12.1% at Phoenix.
- The target mining area mineralization is more competent at Phoenix than at Cigar Lake. Cigar Lake mine uses a bulk freeze approach to strengthen the area of the deposit mined.
- At both projects, the immediate HW is weak. Cigar Lake mine relies on ground freezing to strengthen the HW. A similar approach is planned for the Phoenix HW.
- Cigar Lake mine is scheduled to produce 18 Mlbs/year $U_3O_8$ compared to an average planned rate of 7 Mlbs/year for Phoenix.

### Cigar Lake Jet Bore System

The information in this report section has mainly been extracted from a Cameco public report, “Cigar Lake Project, Northern Saskatchewan, Canada,” February 24, 2012, (Cameco, 2012). In addition, SRK has relied on information from Cameco’s website and news releases.

Prior to mining at Cigar Lake, the mineralization and 10 m of the HW is frozen using a bulk freezing technique. This both prevents water inflows from the sandstone unit and strengthens the weak ground.

This mining method requires an access drill drift in basement (waste) rock below the mineralization to be mined. An oversized pilot hole is drilled up into the deposit and a casing is installed. A pipe string equipped with a high pressure side firing water jet nozzle at the top, will be installed inside the casing. While the pipe string is rotated, the water jet will cut a cavity in the surrounding mineralization. A slurry of water and loose broken rock flows by gravity out of the cavity created, through a blade screen, down through the annulus between the rotating pipe string and the casing, and into a receiving car next to the jet bore machine. The jet bore machine has successfully excavated cavities in the range of 4 to 7 m in diameter.

After cavity completion, the pipe string will be removed and the jet boring mining system moved to the next mining location. To complete the cavity cycle, mined out cavities are completely backfilled with concrete that is of sufficient strength to withstand the force of the water jet when an adjacent cavity is mined. Due to the value of the mined material, a very high mining recovery percentage is planned.

The water and high grade mineralization (broken rock at this stage) slurry is processed underground by crushing and grinding so it can then be pumped to surface as a slurry, as discussed in report Section 17.1.2. The entire system is designed to remove operators from having any direct contact with the high grade uranium mineralization.

Figure 16-16 is a 3D view of mined cavities from JBS early test results in year 2000 during development of the mining system. Figure 16-17 shows a schematic view of the JBS mining method.

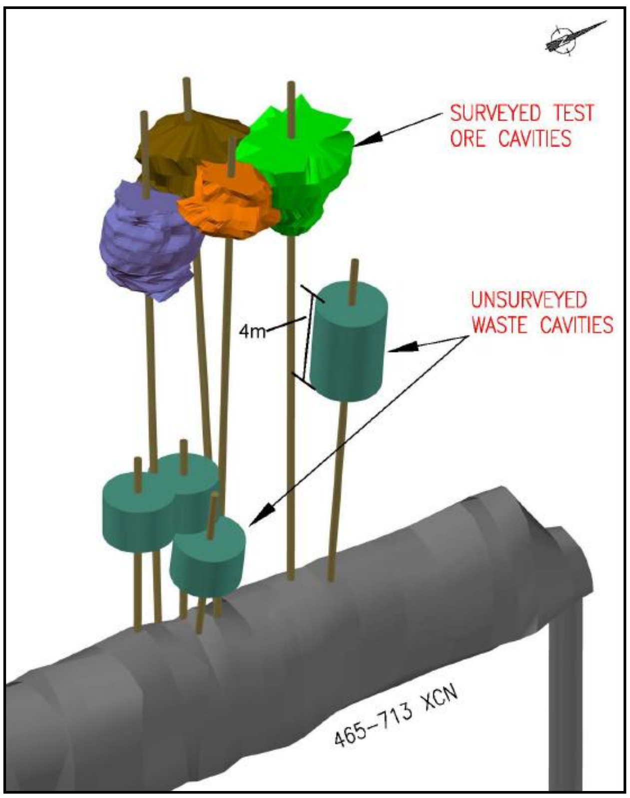



**Figure 16-16: 3D View of Year 2000 Initial Cigar Lake JBS Test Cavities**

Source: Cameco, November 12, 2012

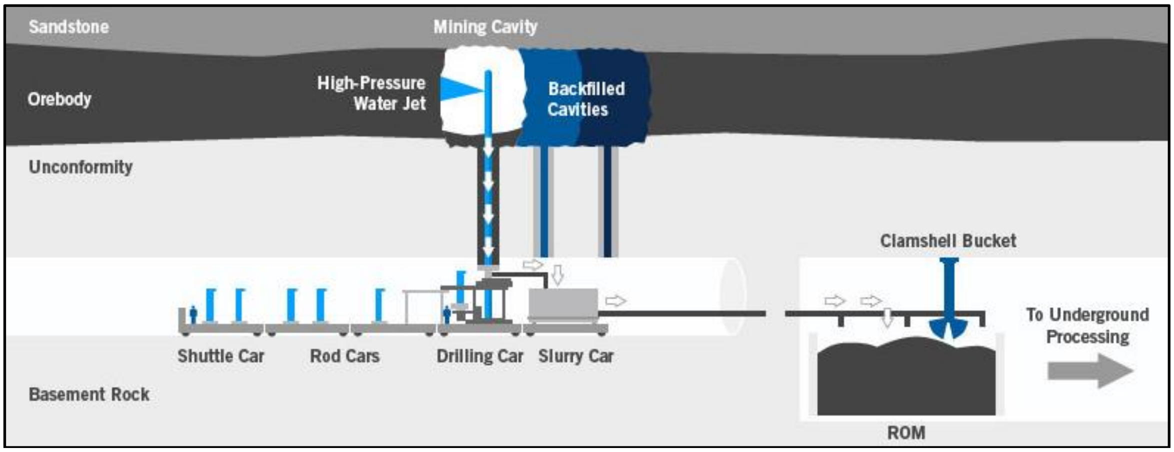



**Figure 16-17: Schematic View of JBS**

Source: Cameco, November 12, 2012

Technical information about the year 2000 JBS test results made public by Cameco includes:

- The ability to excavate roughly circular cavities with an average diameter of more than 4 m, without attempting any optimization.
- The achievement of an average productivity rate of 7 to 10 t/h while jetting.
- Cycle times were determined to be approximately 152 hours for the four test cavities mined in the deposit. This factor has since been revised to 160 hours reflecting changes to the mining horizon and other process changes.
- The ability of the broken rock and water to flow from the cavity, through the preventor and slurry car, and pumping of the slurry down the pipeline to the RoM storage facility.
- The ability to use 40 MPa concrete as backfill and its ability to withstand jetting from an adjacent cavity.

In addition, a February 2014 news release by Cameco stated,

> *"Cigar Lake will eventually have four JBS units on site, with each new one being improved upon by the lessons of the previous iteration. The mine will be capable of hitting full production capacity with two units in operation."*

Table 16-5 shows estimated JBS productivity values comparing the Cigar Lake application to the Phoenix deposit planned application. Two JBS units are required for the Wheeler River project.

**Table 16-5: JBS Method Productivity Comparison**

| Item | Units | Cigar Lake | Phoenix |
|---|---|---|---|
| Operating JBS units | number | 3.0 | 1.8 |
| Mine production rate | Mlbs/year | 18.0 | 7.0 |
| Mining grade | % $U_3O_8$ | 17.8% | 12.1% |
| Mining grade | lbs/tonne | 392 | 267 |
| Tonnes mined per year | tonnes/year | 45,869 | 26,210 |
| Tonnes/year per operating unit | tonnes/year/unit | 15,290 | 14,977 |
| Tonnes/day per operating unit | tpd/unit | 42 | 42 |
| Tonnes/hour per operating unit | tonnes/hour | 8.0 | 8.0 |
| Average jetting time per cavity | hours/cycle | 41 | 37 |
| Deposit vertical thickness | metre | 6.0 | 6.0 |
| Diameter of jet bore cavity | metre | 5.0 | 5.0 |
| Jet bore cavity volume | cubic metre | 118 | 118 |
| Mineralization density | tonne/cubic m | 2.75 | 2.50 |
| Tonnes per mined cavity | tonnes/cavity | 324 | 295 |
| Days to cycle a mined cavity | day/cycle | 7.6 | 7.1 |
| Total cycle duration | hours/cycle | 183 | 170 |

## Planned Jet Bore Mining at Phoenix Deposit

Figure 16-18 shows a long section view of the portion of the Phoenix Zone A planned for jet bore mining. A jet boring drill access drift is shown 30 m below the deposit, which is 320 m in length in this view. Only some of the “tent” configuration freeze holes are shown. They are actually planned at a spacing of 4 m along strike.

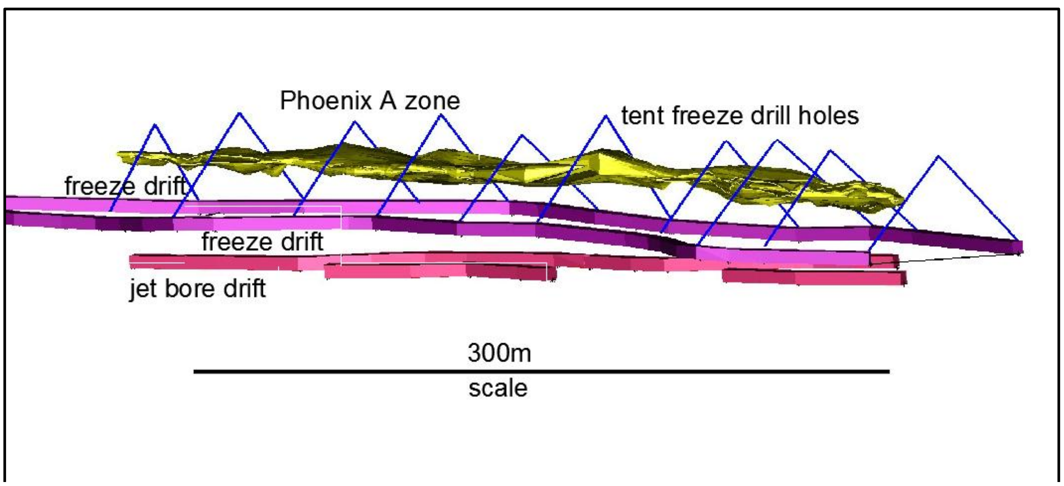



**Figure 16-18: Isometric View - Phoenix A Zone Tent Freeze and Jet Bore Drift**
(Looking E)

Figure 16-19 shows a vertical cross-section through the Phoenix Zone A area planned for JBS mining. The magenta coloured resource blocks in the centre of the deposit grade more than 20% $U_3O_8$. A jet bore mining drill drift is shown positioned below the deposit.

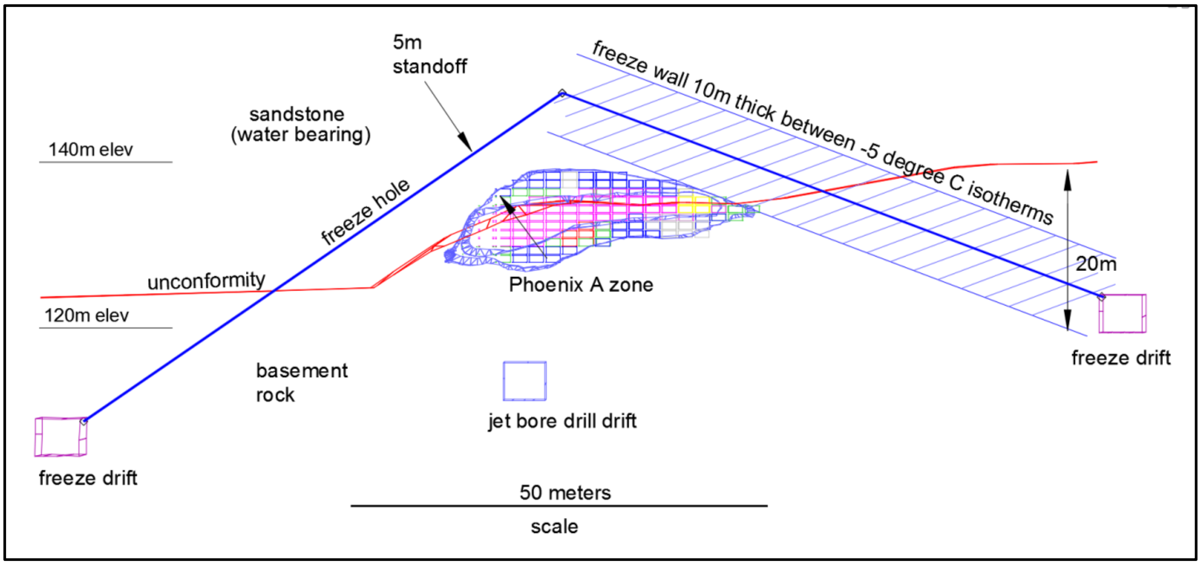



**Figure 16-19: Phoenix A Zone Cross-Section - Tent Freeze Wall Arrangement (Looking NE)**

Freeze drifts on either side are positioned 15 to 20 m below the unconformity into the basement rock. Freeze hole drilling will be done from these drifts at a spacing of 4 m along strike. A freeze wall of 10 m thickness is expected to develop after operating the freeze system for 16 months, at which time jet boring can begin.

The freeze holes will ideally be positioned no closer than 5 m from the nearest planned excavation (jet bore cavity). The intention with this layout is to have frozen ground in close proximity to the immediate HW of the deposit to strengthen the rock and limit external dilution, and prevent any uncontrolled caving.

More than one jet boring drift will be required where the deposit is wide. SRK notes that the JBS unit has the capability of angling the pilot hole a maximum of 18° off vertical to either side, and this will maximize the tonnes mined from each machine set up.

Concrete will be used to backfill mined cavities by pumping it into each cavity after installing a breather pipe to the top of the cavity.

Backfill for Phoenix will be prepared at the Gryphon mine site. Aggregates will be prepared by crushing and screening development waste rock during the summer. The concrete will be prepared inside a backfill batch plant by mixing aggregates, cement, water and additives in a 10 $m^3$ ready mix truck. The truck will deliver the concrete to the slick line installed in the Phoenix ventilation raise. The average requirement will be for 26 $m^3$ per day.

### 16.5.3 Gryphon – Longhole Stoping

The average mining grade at Gryphon is estimated at 1.90% $U_3O_8$, however, there are areas of much higher grade locally. Conventional mining methods are planned for Gryphon.

The geometry of the mineralized lenses at Gryphon is well suited to longhole mining using cemented waste rock backfill. In some areas, the lenses may have to be mined together due to their close proximity to each other. A production rate of 400 tonnes per day has been selected to yield an annual uranium production rate of six million pounds.

Sublevels spaced 15 m were selected due to the narrow vein nature of the deposit considering blast hole deviation in the down dip dimension, and also due to the variable shapes of the mining wireframes as viewed in a vertical projection. Closely spaced sublevels will facilitate access to the upper and lower elevation limits of each mining wireframe. Refer to Figure 16-20.

Table 16-6 provides a list of the mining wireframes shown in Figure 16-20 working from the FW to the HW. It also shows the nominal separation distance between them. In some limited areas, it is likely that two adjacent lenses will be mined with one stope encompassing both.

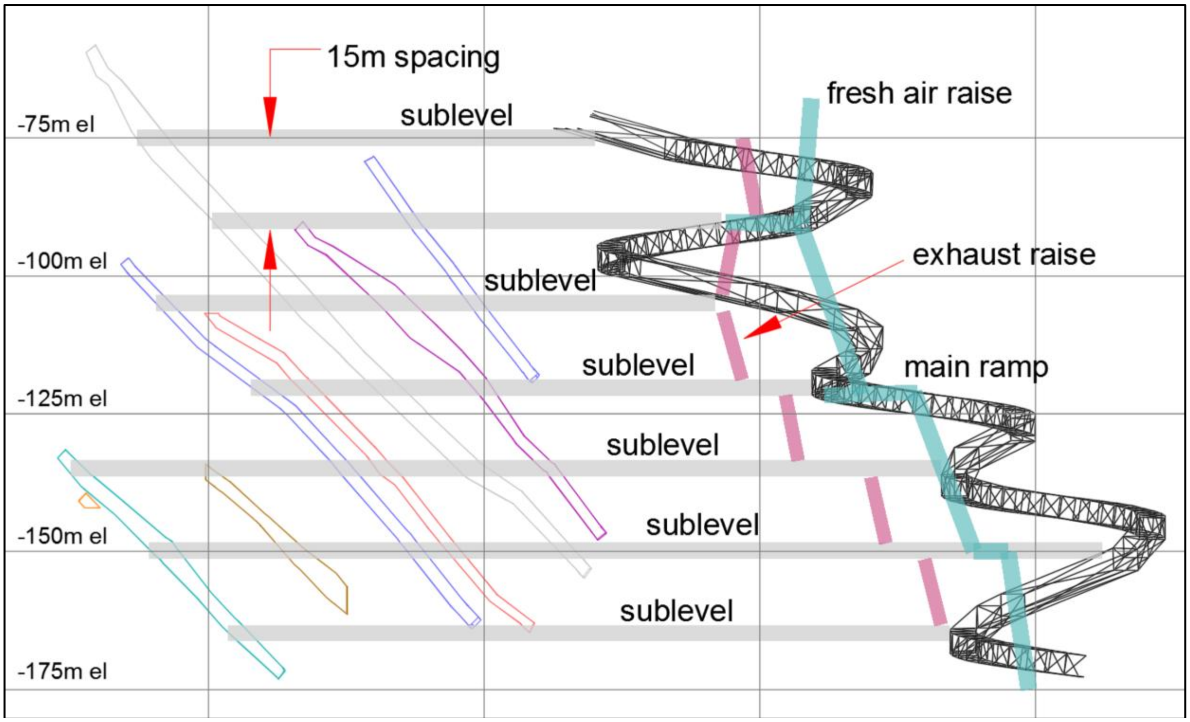



**Figure 16-20: Gryphon Vertical Section – Longhole Stoping Layout (Looking NE)**

**Table 16-6: Gryphon Stacked Mining Wireframes (Lenses)**

| Mining Wireframe | Average Lens True Thickness (m) | True Lens Separation Distance (m) |
|---|---|---|
| C1_1_Clip | 2.5 | |
| | | 14 |
| B3_1_Clip | 2.8 | |
| | | 15 |
| B2_1_Clip | 2.8 | |
| | | 3 |
| B1_1_Clip | 3.2 | |
| | | 8 |
| A1_1 | 4.0 | |
| | | 5-8 |
| A2_1_Clip | 2.9 | |
| | | 6-12 |
| A3_1 | 1.9 | |

Longhole drilling will be done with a small top hammer drill such as the Boart StopeMate ideally suited to narrow vein mining. Blast hole will be roughly 16 m in length and 63 mm (2.5 inch) in diameter. Anfo will be used for blasting as much as possible where conditions are dry.

Stope size will be roughly 15 m along strike by full lens width and one sublevel in height for an approximate 1,500 tonnes per stope. A mining rate of roughly two stopes per week will be required. The small stopes, fast mining cycle and cemented rock backfill will help maintain excavation stability and mining control.

Stope sequencing will be longitudinal along strike, retreating towards the level crosscuts, and will progress from the footwall towards the hangingwall.

Figure 16-21 shows typical level development (sublevel -150 m) for longhole stoping at Gryphon. In this case seven of the lenses are accessed by three mining crosscuts.

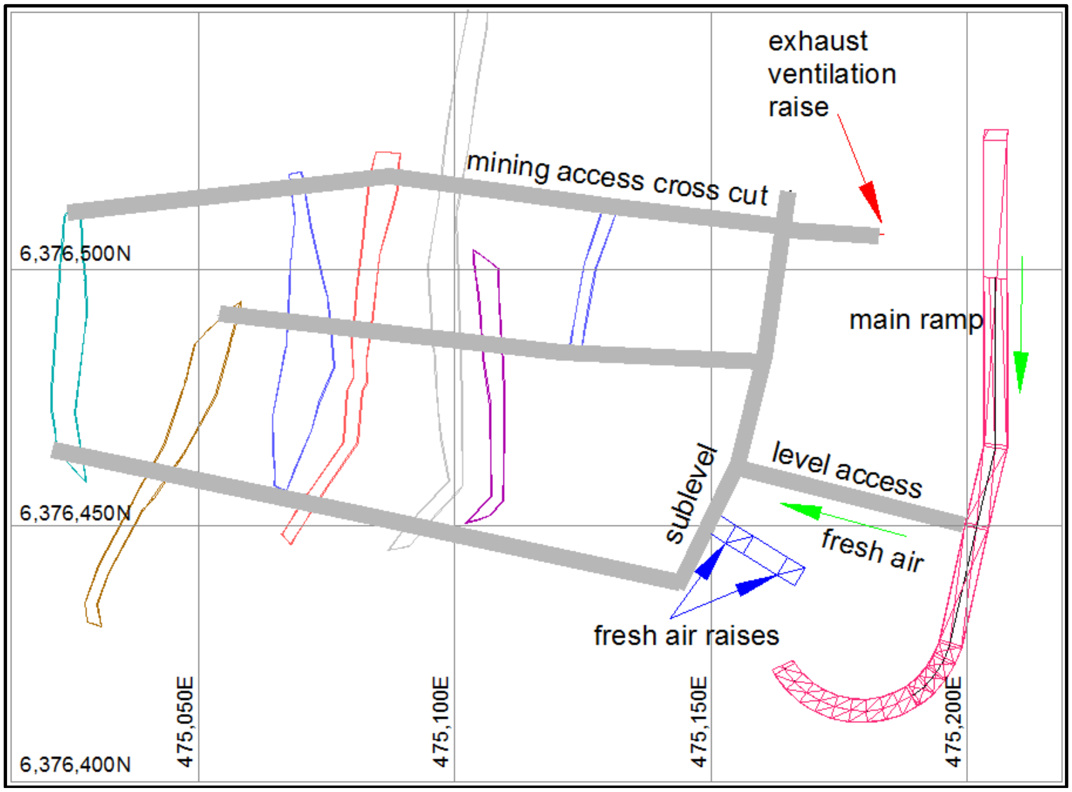



**Figure 16-21: Gryphon Plan View – Typical Level for Longhole Mining**

The Gryphon main ramp has planned dimensions of 4.5 m wide by 4.0 m height. Level development is sized at 4.0 m by 3.8 m.

Mining activity in a crosscut will be ventilated by auxiliary fan and ducting. An exhaust system is planned whereby exhaust air is drawn into rigid ducting near the workplace and ducted to the exhaust raise.

Development rock and stope muck will be loaded by LHD into 20 tonne trucks on the level such that any dust created and radon gas emitted will be drawn to the exhaust raise on the level. Waste rock and mineralized material will be trucked up to a truck dump and rock breaker installation near the shaft at -6 m elevation. There will be separate dump and rock handling systems for waste rock and mineralized material to prevent cross contamination.

Development waste rock with 4% by weight cement slurry added will be used to backfill stopes. A portable cement slurry plant, fed by bulk one tonne bags of cement will be used (Figure 16-22). A

50 mm diameter pipe will deliver a metered amount cement slurry from the slurry mixer into the bucket of an LHD close to the dumping point into the mined out stope.


**Figure 16-22: Skid Mounted Portable Slurry Plant**

### 16.5.4 Other Mining Methods Considered

#### Phoenix – Blind Raise Bore Mining

The geometry at the Phoenix Zones A and B is well suited for blind raise boring (Figure 16-23). This method was successfully tested at the McArthur River mine, but it was not incorporated into their life-of-mine (LOM) plan.

Figure 16-24 shows an enlarged view of a possible blind raise bore mining method for the Phoenix deposit higher grade areas.

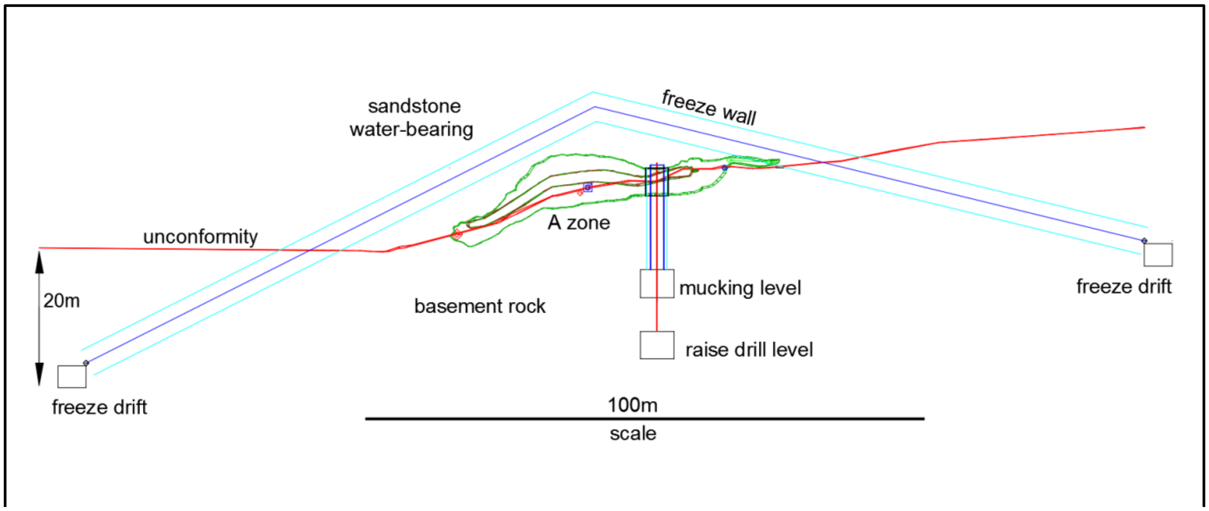



**Figure 16-23: Possible Blind Raise Bore Layout for the Phoenix Deposit (Looking NE)**

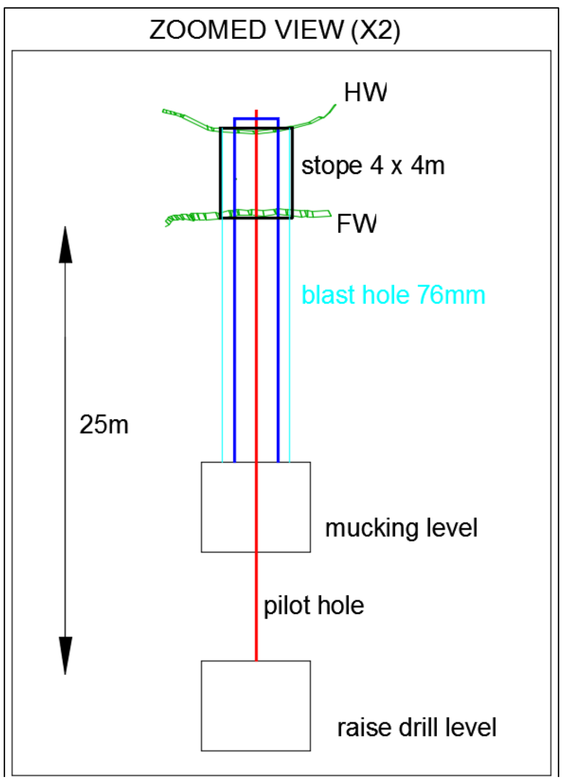



**Figure 16-24: Enlarged View of Possible Blind Raise Bore Method for Phoenix Deposit**

This mining requires two stacked drifts below the deposit. The raise drill would be positioned in the clean environment of the lower drift. The pilot hole would be drilled vertically up to the deposit HW, and then retracted to the mucking level to attach the reaming head.

The raise drill would blind bore upwards, represented by the dark blue outline in Figure 16-24. This bored excavation alone does not produce very much mineralized raise drill cuttings due to the relatively thin vertical dimension of the deposit (Zone A 6.3 m, Zone B1 5 m).

To increase the tonnes produced per raise drill set up, parallel blast holes pre-drilled adjacent to the planned location of the bored raise could be blasted into the raise void. Blasting would take place only within the mineralized portion of the raise (in the deposit). Blasted rock would be ejected out of the small, square stope, down through the raise to the mucking level.

Preliminary estimates by SRK indicate that opening up the stope size to 4.0 x 4.0 m in plan view would make the method viable (even with limited vertical thickness). This assessment is based on making a comparison to the McArthur River mine where the publicly stated cut-off criteria for conventional (reaming upwards) raise bore mining is to generate a minimum of 5,000 lbs of $U_3O_8$ at a minimum grade of 0.8% $U_3O_8$ per raise drill set up.

It is likely that this mining method would require more lateral development than using the JBS method.

This method was not selected for the following reasons:

- On an overall basis it was considered less productive than JBS
- Increased lateral development
- Blasting would be required near the freeze wall and next to the weak HW
- The blind boring method described here would involve more exposure of personnel to the high grade mineralization – longhole drillers drilling up holes, and blasters loading blast holes next to the open (safety bulkhead installed) raise hole

#### SRK Comment

At this level of study, the blind raise bore method cannot definitively be ruled out. It should be reconsidered as part of any higher level, more detailed, technical study.

#### Gryphon – Conventional Raise Bore Mining

Conventional raise bore mining was considered for the higher grade portion of the Gryphon zone. The raise drill could be set up at roughly elevation -100 m and ream back 80 m vertical raises from undercut development at roughly -180 m elevation. A reaming head of 2.4 to 3.0 m diameter could be used.

This method was not selected for Gryphon because of its lower productivity and higher operating cost. It would be difficult to justify raise bore mining with an average mined grade of 1.89% $U_3O_8$.

## 16.6 Underground 3D Mine Model

### 16.6.1 Mine Access Methods

#### Introduction

Several different configurations are possible when considering how to provide underground mining access to the Gryphon and Phoenix deposits. An important aspect of the design approach is how to maximize synergy between the two deposits. The distance between the two, at roughly 3 km, is such that the question must be answered as to whether it is best to connect them underground, or to develop them with separate accesses from surface.

Another aspect that was assessed as part of the mining access design, is how much ventilation air each planned mine will require. Gryphon has been planned at 302 cms (640,000 cfm), and Phoenix at 240 cms (508,600 cfm). These estimates are discussed further in a subsequent report section.

Further, to optimize mine access, the sequence of deposit mining must be established. This study is based on mining Gryphon first, and planning for sequential production, with Phoenix production ramping up as Gryphon is exhausted. Mining Gryphon first provides the following advantages:

- Minimized risk (water inflow) and cost (no freeze wall) by mining in the basement rock units
- Minimizing risk, schedule and cost by applying a well-established conventional longhole mining method (as compared to the planned jet bore mining at Phoenix)

Sequential mining of the deposits will provide a relatively smooth production profile (uranium lbs/year) favourable for negotiating a custom milling contract. It will also reduce financing requirements by delaying much of the Phoenix capital to the period when Gryphon is producing positive cash flow.

Aspects considered in the mine access design included:

- Minimizing capital costs
- Maximizing synergy between the two deposits, including ability to move workers, equipment and materials between deposits
- Providing sufficient air flows without exceeding rule-of-thumb air velocities
- Transporting waste rock and low grade (conventionally handled) mineralization to surface
- Transporting waste rock from surface back underground, should it be needed for backfill
- Transporting workers and mining supplies to each deposit
- Moving the mobile mining fleets underground
- Providing for the following services to each deposit:
    - Mine dewatering sumps and pumps
    - Electrical power distribution
    - A second means of exit from the underground
- Providing for the following services to the Phoenix deposit:
    - Routing for brine piping for tent freeze walls
    - Routing for high grade uranium slurry transport
    - Slick line for concrete for jet bore cavity backfilling

## Selected Design

The design approach selected is to connect the two deposits underground with a 2.8 km connection drift. The drift will be driven only from the Gryphon side at a gradient of +4% and dimensions of 5.5 wide by 6.0 m height. It will be positioned about 30 m into the basement rock. Planned synergies between the two deposits include:

- Requirement for only one full service production shaft for hoisting waste rock and low grade mineralization
- Movement of workers and materials between deposits
- Requirement for only one vertical access to surface at Phoenix
- Requirement for only one main dewatering station pumping to surface (at Gryphon)
- Consolidation on surface to one set of water management ponds and one water treatment plant

- Consolidation of all waste rock handling and storage at the Gryphon site (the only exception will be 19,000 tonnes of reamer cuttings from blind boring the ventilation raise at Phoenix)
- Consolidation on surface of mine buildings at the Gryphon site (mine office, change house, warehouse, maintenance shop, camp, etc.)

Blind bored shafts have been selected for vertical access in favour of typical full face shaft sinking with cover grouting or freeze curtain protection.

The blind boring method selected will begin with a surface pre-grouting program, averaging approximately 45 full depth boreholes per shaft (primary plus secondary grouting). Pilot hole drilling into the basement rock will be followed by blind boring with the shaft full of drilling fluid. Following shaft boring, the drilling rig is disassembled and a concrete lining system including head frame, hoists, forms, work deck, ventilation, and concrete conveyance are erected over the shaft.

Installation of the concrete liner is completed from the top down using collapsible concrete forms that are expanded to proper diameter, casting in place approximately 6 m of the concrete liner per cycle. The fluid in the shaft is maintained at a level just below the work of concrete lining. This is done to maintain a hydrostatic force on the shaft walls as well as to provide additional safety for those working in the shaft.

The main advantage of the blind boring method is the security of virtually eliminating the risk of unexpected shaft water inflow during shaft construction. Blind bored shafts offer competitive costs and construction schedules.

For Gryphon, the mine design includes a full service production shaft and a bare ventilation raise (both blind bored and concrete lined) to support underground development and production. Heated fresh air will be delivered through the shaft with return air up the ventilation raise. Mobile equipment will be lowered underground through the 5.5 m diameter Gryphon production shaft. An emergency man hoist will be set up in the ventilation raise.

The connection drift (Figure 16-25) will be driven from Gryphon to Phoenix at a just-in-time line advance rate of 2.1 m/d. Development waste rock from this heading will provide 50% of the backfill rock needed at Gryphon while the drift is being driven.

This access drift will be extended to the location of the Phoenix ventilation raise where a connection will be established, creating a flow through ventilation circuit to surface. All Phoenix pre-production lateral development will be supported from Gryphon through the connection drift. At that time, the emergency egress man hoist at Gryphon will be relocated to the Phoenix ventilation raise.

Phoenix will receive fresh air from Gryphon through the connection drift, and exhaust air will be routed to surface through the Phoenix ventilation raise.

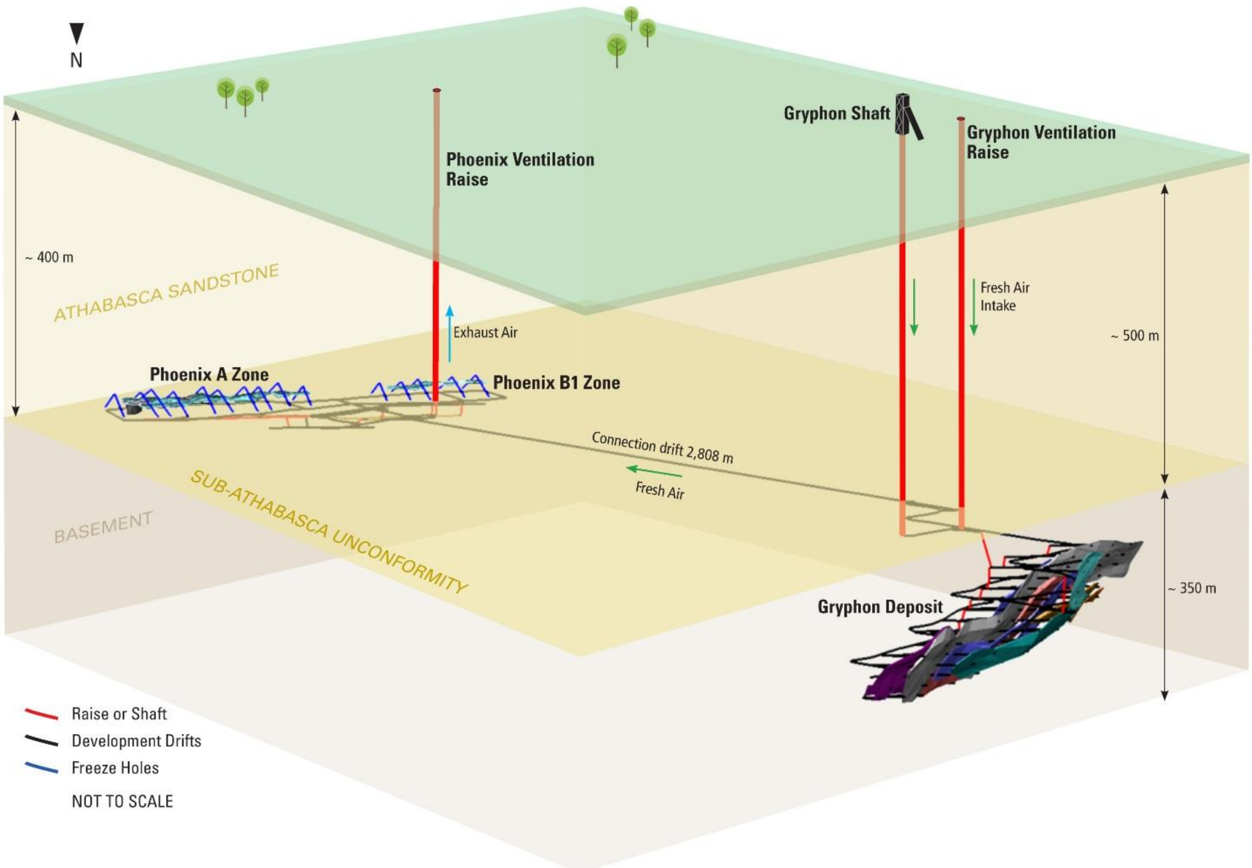



**Figure 16-25: Isometric View - Connection Drift - Phoenix Phase Air Flows**
(Looking S)

## 16.6.2 Phoenix Mine Model

Figure 16-26 shows the Phoenix mine model. Only development centre lines are shown for planned lateral development and the exhaust raise. The ventilation raise will be concrete lined, 4.5 m (15 feet) diameter blind bored raise. The connection drift from Gryphon, 2,808 m in length, is sized at 5.5 m x 6.0 m to handle the required ventilation air while keeping the air velocity within rule of thumb limits.

There are conceptually three levels to the mine layout: the uppermost freeze drifts, the mid elevation jet bore access at 95 m elevation, and the lower infrastructure access at 75 m elevation.

Freeze drifts are planned in basement rock along strike on both sides of the deposit. The elevation of the freeze drifts varies since the drift design elevation is relative to the profile of the unconformity surface. These drifts will accommodate freeze hole drilling and will also distribute fresh air along strike.

The jet bore access drifts are at mid elevation along strike and a central drift at 95 m elevation serves to access underground infrastructure. The lowest level at 75 m elevation provides access to centrally located infrastructure including dewatering sumps.

A dedicated exhaust ventilation drift at mid elevation is planned along strike to set up single pass ventilation through the active jet boring areas.

Figure 16-27 is an isometric view of the planned underground infrastructure in the central area of the Phoenix mine.

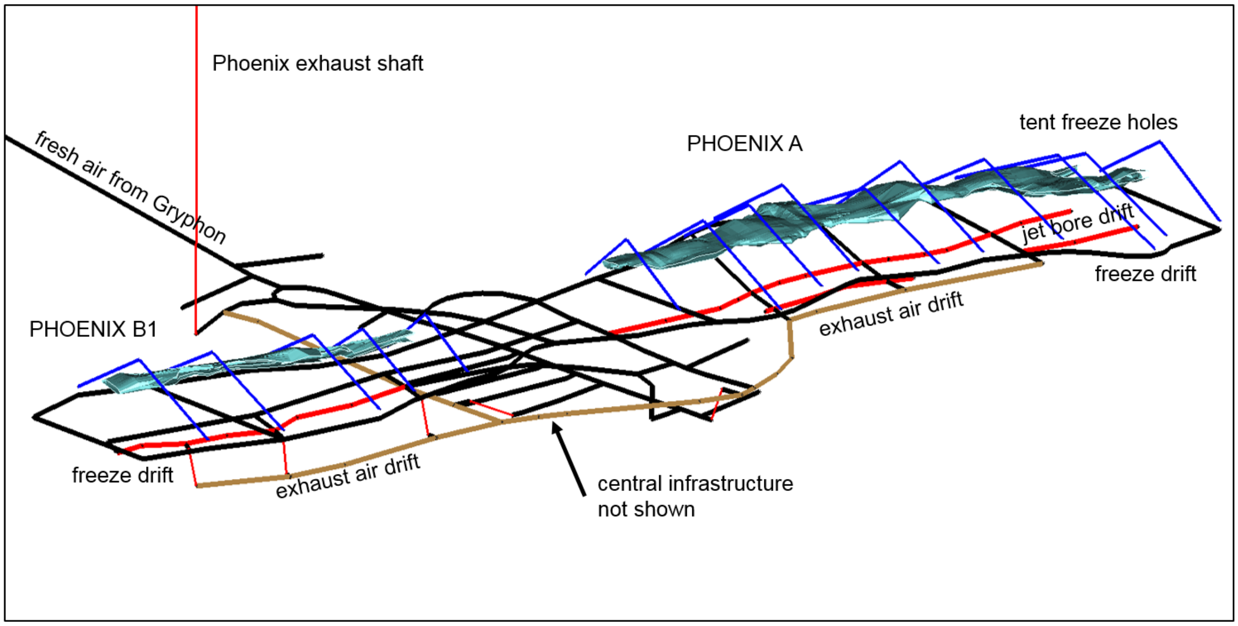



**Figure 16-26: Phoenix 3D Mine Model – Isometric View**
(Looking N)

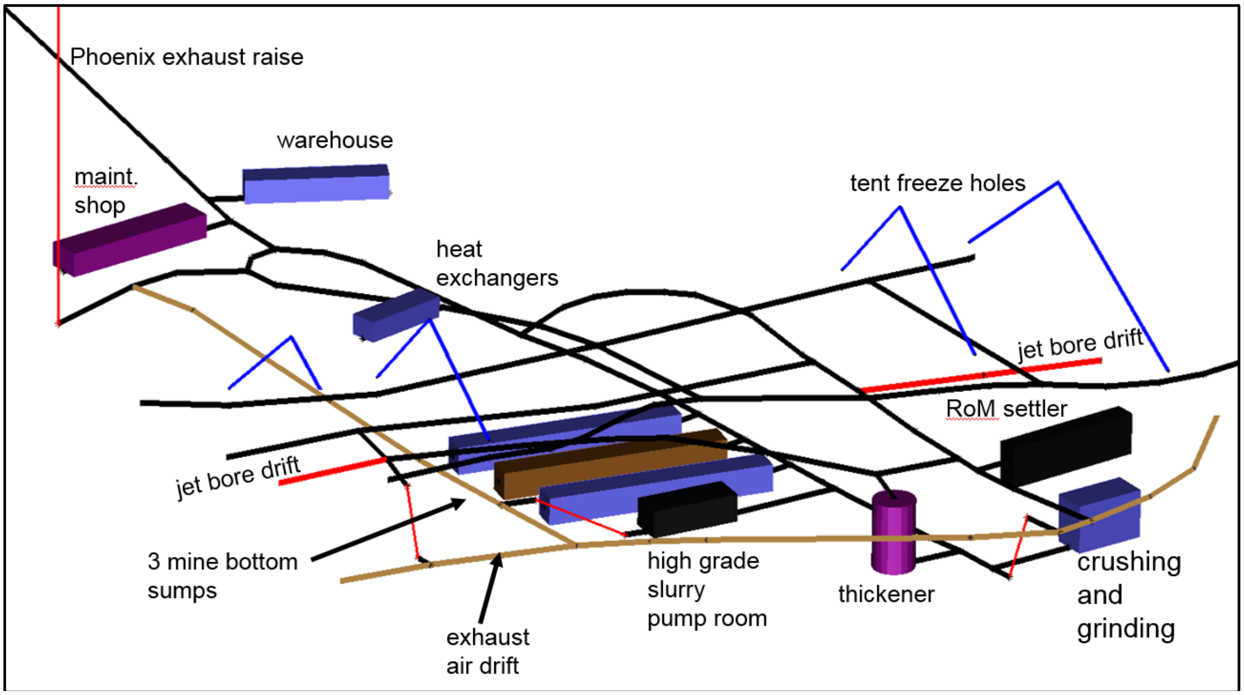



**Figure 16-27: Phoenix Underground Central Infrastructure – Isometric View**
(Looking N)

Figure 16-28 is a schematic vertical section through the central infrastructure area of the 3D Phoenix mine model. Key infrastructure shown includes:

- Freeze drifts and freeze holes; freeze holes range from 60 to 75 m in length
- Heat exchanger room for heat transfer from the high pressure surface to underground brine system to the lower pressure underground chilled brine distribution system
- A dedicated exhaust air crosscut will pass through the central area drawing exhaust air collected along strike directly to the Phoenix exhaust raise
- A run-of-mine (RoM) settling chamber will receive water/coarse rock slurry from jet boring operations and it will be equipped with an overhead clam shell excavator to feed the milling operations planned in close proximity
- The thickener will receive slurry from the mill and send thickened slurry to the slurry pumps that will deliver the slurry up the pipes installed in the Phoenix exhaust raise
- Three large sumps are planned for the lower level, each 60 m in length and 8 m x 8 m in section
- A maintenance shop and warehouse are planned at a central location near the connection drift from Gryphon

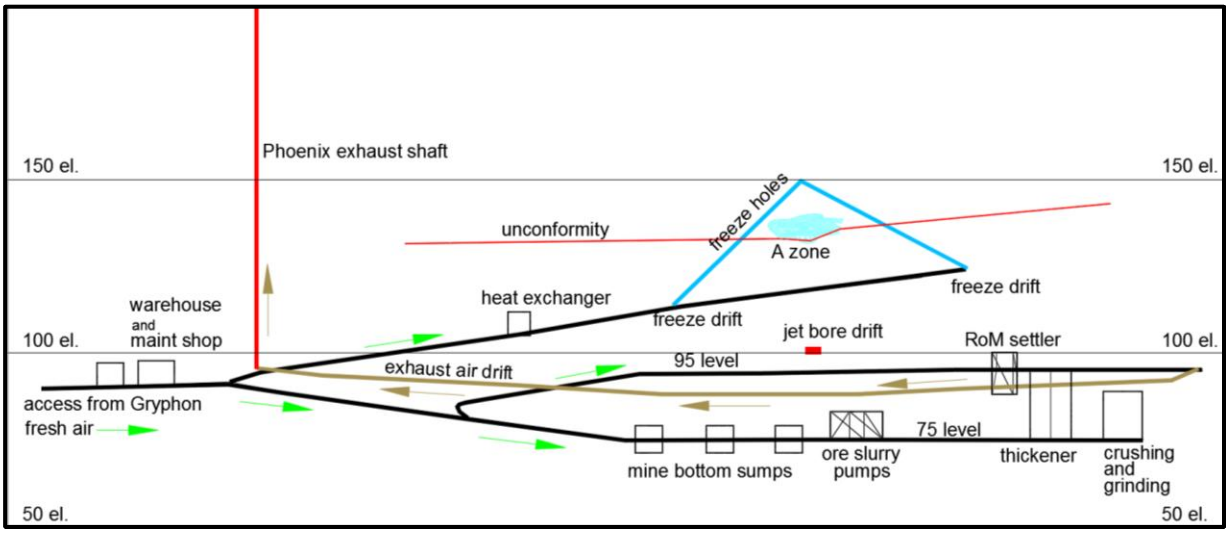



**Figure 16-28: Schematic Section showing Phoenix Central Infrastructure**

(Looking NE)

## 16.6.3 Gryphon Mine Model

Figure 16-29 shows the Gryphon 3D mine model. Only the main ramp is shown as a 3D solid. Other entities are shown as centerlines only – shaft, ventilation raise, mining sublevels, and internal exhaust raise system.

The Gryphon production shaft will be a blind bored, circular concrete lined 5.5 m (18 feet) diameter shaft, equipped with a production hoist  for skipping mineralization and waste rock, and a service hoist for moving men and materials, 583 m in length. The Gryphon ventilation raise will be a blind bored, circular concrete lined 4.50 m (15 feet) diameter raise, 550 m in length. Both the production shaft and the ventilation raise will be sunk a minimum of 25 m into the basement rock below the unconformity.

The main access ramp located on the 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 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 drift are included in the design to allow the ventilation raise systems to follow the plunge.

During the Gryphon mining phase of the project a connection drift to the Phoenix deposit will be developed such that Phoenix pre-production development can be completed before the end of Gryphon production. The line length of the drift is planned at 2,808 m, grading +4% from Gryphon towards Phoenix.

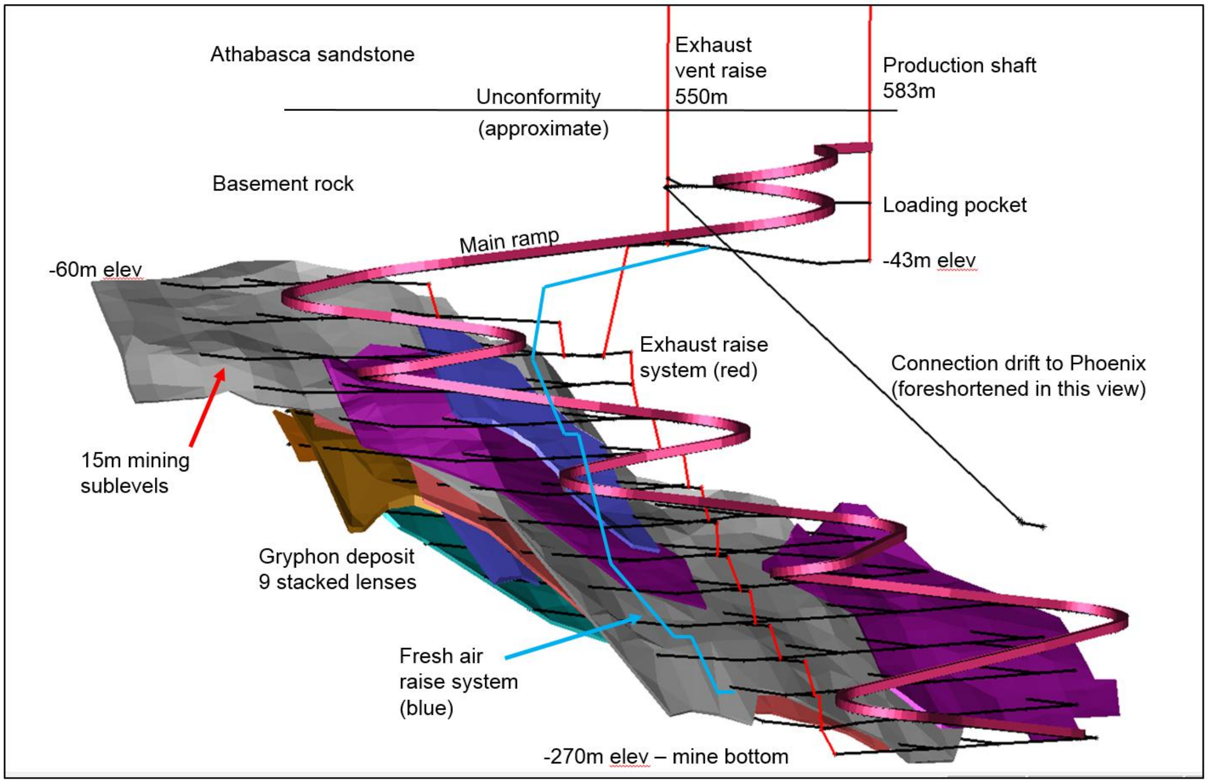



**Figure 16-29: Isometric View - Gryphon 3D Mine Model**
(Looking N)

Figure 16-30 is a vertical cross-section through the Gryphon 3D mine model where the surface elevation is 540 m. The locations shown for the unconformity and offset fault are approximate only. It is apparent, however, from the geometry that a vertical shaft located on the FW side of the deposit would intersect the projection of the offset fault.

The view is a full projection making the individual mining wireframes appear closer together than they actually are.

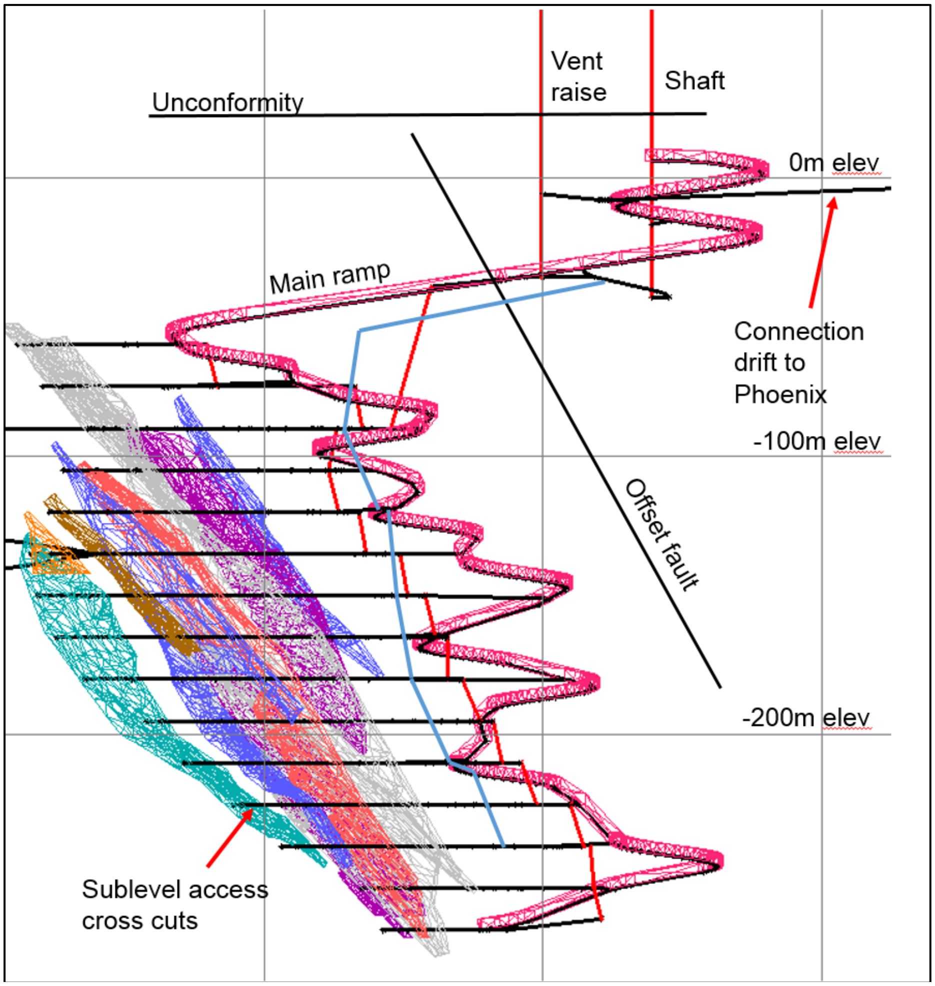



**Figure 16-30: Vertical Section - Gryphon 3D Mine Model**
(Looking NE)

## 16.6.4 Development Requirements

### Lateral Development

Based on the 3D mine models and a list of infrastructure requirements planned for each mine, SRK estimated LOM lateral development requirements as summarized in Table 16-7.

**Table 16-7: LOM Lateral Development Requirements**

| **Lateral Development** | **Gryphon (m)** | **Phoenix (m)** | **Total (m)** |
|---|---|---|---|
| Connection Drift | 3,239 | | 3,239 |
| Other Capitalized Development | 6,588 | 6,310 | 12,898 |
| **Total Capitalized** | 9,827 | 6,310 | 16,137 |
| **Expensed Development** | 4,160 | 4,651 | 8,811 |
| **Total Lateral Development** | **13,987** | **10,961** | **24,948** |

### Vertical Development

Table 16-8 shows the estimated LOM vertical development requirements planned for each mine.

**Table 16-8: LOM Vertical Development Requirements**

| Vertical Development | Gryphon (m) | Phoenix (m) | Total (m) |
|---|---|---|---|
| Production Shaft (5.50 m dia.) | 583 | | 583 |
| Ventilation Raise (4.50 m dia.) | 550 | | 550 |
| Ventilation Raise (4.50 m dia.) | | 440 | 440 |
| Other Ventilation Raise ( 4.5 x 4.5 m) | 404 | | 404 |
| Other Ventilation Raise ( 3.0 x 3.0 m) | | 80 | 80 |
| **Total Vertical Capitalized** | **1,537** | **520** | **2,057** |

## 16.6.5 Underground Equipment Requirements

Estimated underground equipment requirements are shown in Table 16-9.

**Table 16-9: Estimated Underground Equipment Requirements**

| Unit Type | Gryphon | Phoenix |
|---|---|---|
| Tractor | 5 | 5 |
| 2-Boom Jumbo | 2 | 1 |
| Scissor Lift | 3 | 2 |
| Mech Bolter | 1 | 1 |
| LHD 10-tonne | 3 | 3 |
| LHD 6.7-tonne | 2 | 1 |
| Truck 20-tonne | 3 | 2 |
| Shotcrete Unit | 1 | 1 |
| Transmixer | 1 | 1 |
| Boom Truck | 2 | 1 |
| Longhole Drill 64 mm | 2 | 0 |
| Jet Bore Units | 0 | 2 |
| Personnel Carrier | 1 | 2 |
| UG Forklift | 1 | 1 |
| Service Truck | 1 | 1 |
| Fuel/Lube Truck | 1 | 1 |
| Grader | 1 | 1 |
| **Total Underground Units** | **30** | **26** |

Mining equipment will be lowered underground in the Gryphon production shaft. When Phoenix mining begins, there will be some used equipment available from Gryphon to supplement the new Phoenix equipment purchases shown in the table.

## 16.6.6 Waste Rock Broken and Backfill Requirements

Table 16-10 shows estimated LOM quantities of development waste rock broken and rock required for backfilling.

**Table 16-10: Waste Rock Broken and Backfill Quantities**

| Waste Rock Broken | LOM Quantity (kilotonne) |
|---|---|
| Gryphon | 659 |
| Phoenix | 530 |
| **Total Broken** | **1,189** |
| **Backfill Required** | |
| Gryphon | 634 |
| Phoenix | 204 |
| **Total Rock for Backfill** | **837** |
| **Excess Waste Rock** | **352** |

SRK prepared a high-level annual schedule of these quantities that indicates that it should be possible to keep up with the Gryphon backfill demand using development waste broken underground. The rate of advance in the connection drift to Phoenix can be varied to help match the demand for broken waste rock.

SRK also notes that some of the excess waste rock shown in Table 16-10 will likely be used for construction on site.

### 16.6.7 Other Mine Infrastructure Options Considered

#### Independent Projects

The concept of having completely separate projects for the two deposits with independent mine access at each was rejected based on the potential to generate synergies by combining infrastructure items (see Section 16.6.1).

## Vertical Development

Options considered for vertical development are shown in Table 16-11.

**Table 16-11: Options Considered for Vertical Development**

| No. | Option | Key Points | Cost | Risk | Comments |
|---|---|---|---|---|---|
| 1 | Full face shaft sinking with cover grouting protection | Proven in basin Long term operational challenges | Moderate costs | Extreme risk due to potential for uncontrolled inflows | Eliminated due to high risk |
| 2 | Full face shaft sinking with freeze wall protection around the shaft | Proven in Sask. (potash)<br>Requires hydrostatic liner | Moderate cost for freezing and sinking; lining options are expensive | | Eliminated due to high cost |
| 3 | Raise bored shaft (pilot raise and slash) with freeze wall protection around the shaft | Requires access to bottom of raise | Reasonable costs | Risk of sloughing during piloting and slashing of raise | Option still plausible |
| 4 | Blind bored shafts with full surface pre-grouting program, shaft sinking with shaft full of water, concrete lined | Grouted, drilled and reamed from surface reducing HSE and inflow risks | Reasonable costs | New technology to the basin – capabilities to be determined | Selected due to moderate cost and low risk |
| 5 | Ramp from surface contained within freeze wall | Provides flexibility during operations | Moderate costs | Extended duration of freezing may develop long term challenges | Eliminated due to cost |

As previously discussed, option (4) above was selected for this study. The main advantage is the security provided by the initial surface grouting program and having the shaft full of water during sinking. Option (1), involving cover grouting, was considered as inadequate protection against unexpected water inflows.

Options (2), (3), and (5) are based on freeze wall protection. Common practice is the installation of a hydrostatic liner or maintenance of the freeze wall for the life of the project. Both are significant cost disadvantages.

Compared to full face shaft sinking, option (3) offers a faster construction schedule and capital cost saving if there is pre-existing access to the bottom of the shaft. For Gryphon, this would mean first sinking a shaft and then developing over to the ventilation raise location to set up the pilot raise. Such an approach would actually delay the construction of the ventilation raise compared to simply full face sinking or blind boring both the shaft and ventilation raise at the same time.

## Freeze Infrastructure Alternatives

Applications considered for freeze infrastructure included:

1. A tent freeze arrangement for Phoenix, installed from underground
2. A perimeter freeze wall installed from surface all around the entire Phoenix deposit, such a wall would have a perimeter length of 1.8 km

3. A bulk freeze pattern drilled vertically from surface to freeze the complete HW area of the Phoenix deposit
4. A circular freeze wall installed from surface at Phoenix protecting a spiral mine access ramp and ventilation raise, such a circular wall would have a diameter of 100 m

Table 16-12 presents a rough comparison of these possible freeze applications considering the amount of drilling required and approximate freeze infrastructure capital costs.

**Table 16-12: Freeze Infrastructure Alternatives Considered**

| No. | Freeze Infrastructure Alternative | Freeze Hole Spacing (m) | Freeze Hole Total Length (m) | Factored Infrastructure All-in Cost ($M) |
|---|---|---|---|---|
| 1 | Tent freeze above the Phoenix deposit HW | 4 | 24,000 | $74 |
| 2 | Perimeter freeze from surface around the Phoenix deposit | 3 | 200,000 | $440 |
| 3 | Bulk freeze from surface of the Phoenix HW area | 6 x 6 | 420,000 | $700 |
| 4 | Circular freeze wall to protect a spiral mine access ramp | 3 | 50,000 | $110 |

Option (1) above was selected for this study. The main advantages of the tent freeze wall arrangement located in the Phoenix deposit's immediate HW are the protection against water inflows and the strengthening of the HW rock mass.

Options (2) and (3) were rejected based on challenging logistics and very high capital costs.

Option (4) considers development of a spiral ramp from surface at Phoenix, protected inside a 100 m diameter circular freeze wall installed from surface. A vertical ventilation/service raise was to be positioned centrally inside the spiral (also protected by the freeze wall) as part of the concept. This option requires additional freezing infrastructure to be a complete, workable alternative, such as the addition of option (1) that would allow the mining of the Phoenix deposit.

This option was rejected because it is much more expensive than installing a blind bored ventilation raise at Phoenix instead of the spiral ramp. Ramp access is not needed or justified with a fully equipped shaft already planned at Gryphon and a connection drift between the two deposits.

## 16.7 Development and Production Schedule

### 16.7.1 Estimated Production Rates

The production rates selected for this study are shown below. Daily rates assume 360 days/year.

- Gryphon with a seven-year production period, at 6.0 Mlbs $U_3O_8$ per year, equivalent to 399 t/d of mineralization
- Phoenix with a nine-year production period, at 7.0 Mlbs $U_3O_8$ per year, equivalent to 73 t/d of mineralization

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.

It is SRK's opinion that the selected mining rates are reliably achievable, not necessarily maximum values for each deposit, but appropriate to assess the merits of the project. Higher mining rates should only be considered in the context of a more definitive technical study.

### 16.7.2 Development and Production Schedule

SRK prepared a Gantt schedule to show the permitting phase, development phase, and production phase of the Wheeler River project (Figure 16-31). SRK scheduled an approximate five-year pre-production period from the time the project is permitted in late Q3 2020, until it reaches commercial production (70% of planned production) in Q1 2026. The project production period is 16 years from Q1 2026 to end of 2041.

Milestone dates are:

| | |
|---|---|
| Q2 2019 | Receipt of EA approval |
| Q3 2019 | Production decision based on positive feasibility study |
| Q3 2019 | Start detailed engineering and prepare long lead equipment procurement |
| Q3 2020 | Project permitted |
| Q4 2020 | Start limited work on Gryphon site - roads, site preparation, and procurement |
| Q4 2020 | Shaft sinking contractor mobilizes to site, Gryphon shaft work starts - blind boring |
| Q4 2022 | Work starts on blind boring Gryphon ventilation raise |
| Q2 2024 | Lateral development work starts at Gryphon |
| Q1 2025 | Development in mineralization starts at Gryphon |
| Q4 2025 | First longhole stope blast at Gryphon |
| Q1 2026 | Commercial production achieved at Gryphon |
| Q2 2030 | Two mines are connected by underground drift |
| Q3 2030 | Freeze hole drilling starts for Phoenix tent freeze wall |
| Q2 2032 | Transition of production from Gryphon to Phoenix |

Table 16-13 shows the Wheeler River production schedule. LOM production totals 1.22 Mt of mill feed at an average grade of 3.91% $U_3O_8$ containing 105 Mlbs of $U_3O_8$.

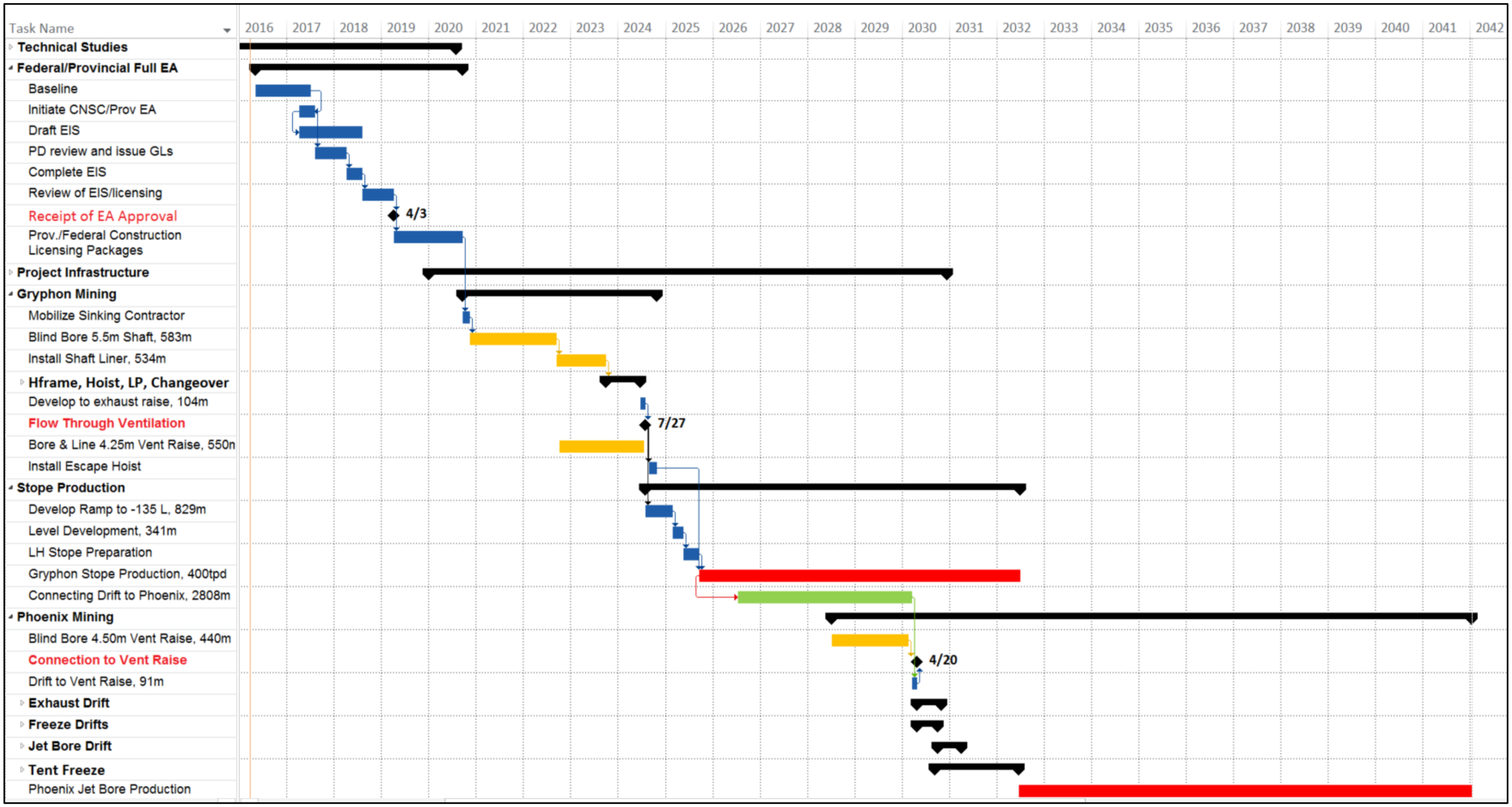



**Figure 16-31: Wheeler River Project Schedule**

**Table 16-13: Wheeler River Project Production Schedule**

| | | | Pre-production | | | | | Production | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Schedule Tasks** | | **2020** | **2021** | **2022** | **2023** | **2024** | **2025** | **2026** | **2027** | **2028** | **2029** | **2030** | **2031** | **2032** | **2033** | **2034** | **2035** | **2036** | **2037** | **2038** | **2039** | **2040** | **2041** |
| **Project Permitted** | | | | | | | | | | | | | | | | | | | | | | | |
| Site Infrastructure | | | | | | | | | | | | | | | | | | | | | | | |
| **Gryphon Shaft** | | | | | | | | | | | | | | | | | | | | | | | |
| | Bore & Line | | | | | | | | | | | | | | | | | | | | | | |
| | Equip | | | | | | | | | | | | | | | | | | | | | | |
| Gryphon Vent Raise | | | | | | | | | | | | | | | | | | | | | | | |
| Gryphon Decline | | | | | | | | | | | | | | | | | | | | | | | |
| Production Preparation | | | | | | | | | | | | | | | | | | | | | | | |
| **Gryphon Mining** | | | | | | | | | | | | | | | | | | | | | | | |
| Kilotonne | 975 | | | | | | 61.8 | 144 | 144 | 144 | 144 | 144 | 144 | 51.6 | | | | | | | | | |
| % $U_3O_8$ | 1.90 | | | | | | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | | | | | | | | | |
| Mlbs $U_3O_8$ | 40.7 | | | | | | 2.58 | 6.00 | 6.00 | 6.00 | 6.00 | 6.00 | 6.00 | 2.16 | | | | | | | | | |
| **Phoenix Mining** | | | | | | | | | | | | | | | | | | | | | | | |
| Kilotonne | 240 | | | | | | | | | | | | | 16.3 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 13.9 |
| % $U_3O_8$ | 12.1 | | | | | | | | | | | | | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 |
| Mlbs $U_3O_8$ | 64.1 | | | | | | | | | | | | | 3.84 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 4.21 |
| **Plant Feed** | | | | | | | | | | | | | | | | | | | | | | | |
| Kilotonne | 1,215 | | | | | | 62 | 144 | 144 | 144 | 144 | 144 | 144 | 68 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 13.9 |
| % $U_3O_8$ | 3.91 | | | | | | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 4.35 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 |
| **Mlbs $U_3O_8$** | **105** | | | | | | **2.6** | **6.0** | **6.0** | **6.0** | **6.0** | **6.0** | **6.0** | **6.5** | **7.0** | **7.0** | **7.0** | **7.0** | **7.0** | **7.0** | **7.0** | **7.0** | **3.7** |

# 16.8 Underground Mine Infrastructure and Services

## 16.8.1 Definition Drilling

The Phoenix deposit is relatively well drilled. For the Gryphon zone, SRK planned an additional 7,800 m of NQ underground core drilling to tighten the average pierce point spacing in the mineralized lenses. A provision has been included for 250 m of underground development coming off of the main ramp to expand drill coverage in the strike direction. The average borehole length will be approximately 100 m.

## 16.8.2 Mined Mineralization and Waste Rock Handling

### Waste Rock and Low Grade Mineralization

The average mining rate at Gryphon will be 400 t/d of low grade mineralization. All of the Phoenix high grade mineralization mined will be handled as a slurry.

Both Gryphon and Phoenix will generate waste rock from mine development, generally at rates that do not exceed 350 t/d, and averaging 200 t/d over the mine life.

The Wheeler River project strategy for handling low grade mineralization and waste rock described below is designed to minimize cross-contamination of the two materials. The intention is to avoid the contamination of the clean waste rock.

At the planned Gryphon mine, waste rock from development will be handled by load haul dump (LHD) vehicles and 20-tonne underground trucks. Remuck bays are planned along the internal ramp at every mining sublevel. Waste rock will either be hoisted or used for backfill.

Mined mineralization will also be handled underground by LHD and truck on the Gryphon internal ramp system.

Waste rock not needed for backfill will be trucked to a truck dump near the shaft. The dump will be equipped with a grizzly and rock breaker. Sized waste rock will enter a small surge bin, then pass through a short raise to a dedicated waste rock feeder on the loading pocket conveyor. Waste rock will be skipped from one side only of the loading pocket. The hoisting system will be designed for 700 t/d in 8 hours of hoisting using only one skip (50% efficiency).

Low grade conventionally mined mineralization will be hoisted at different times than waste rock using the other skip and other side of the loading pocket. It will be fed onto the loading pocket conveyor through a separate, dedicated feeder. A separate truck dump and surge bin is planned for low grade mineralization.

### Phoenix High Grade Mineralization Slurry

Broken mineralization (< 100 mm) and water from the jet boring unit will be deposited into a receiving steel bin (slurry car) next to the jet bore unit and from there, the slurry will be handled as described in report Section 17, Recovery Methods.

### 16.8.3 Freeze Wall Infrastructure

The planned mining method for Phoenix requires a "tent" freeze wall design to prevent water inflows. Freeze infrastructure will include:

- Five self-contained freeze plants on surface located at the Phoenix mine site - each plant is planned to have a power rating of 250 tons of refrigeration (TR) or 880 kW (one TR equivalent to 3.52 kW)
- 300 mm diameter insulated brine circulation piping installed in the Phoenix ventilation raise, and for underground connection to the heat exchangers
- Five underground heat exchangers rated at 250 TR each at 150 $m^3$/hour brine
- LOM total of 25,000 m of tent freeze hole drilling to cover Phoenix Zones A and B1 mining areas

Freeze holes will be drilled approximately 75 m in length, at a 4 m spacing, from two dedicated freeze drifts. Holes drilled at PQ core size will have collars equipped with freeze pipe well heads. HDPE piping will be installed inside the steel cased PQ holes to deliver chilled brine to the ends of the freeze holes.

An initial 100 m strike length of Phoenix Zone A will be frozen as the initial mining block. A freezing time of 16 months is estimated before jet bore mining can start. This will provide a planned 10 m freeze wall thickness between -5°C isotherms.

### 16.8.4 Mine Ventilation

Underground mine ventilation estimates were based on comparisons to other uranium mines and were selected to ensure the two planned uranium mines would be adequately ventilated. SRK estimated required mine ventilation at 302 cms (640,000 cfm) for Gryphon, and 240 cms (508,600 cfm) for Phoenix.

The Gryphon ventilation estimate is based on benchmarking the air flow used for similar uranium underground operations and project studies in the Athabasca basin, northern Saskatchewan. The Phoenix estimate was set as the same as the 2011 Cigar Lake mine ventilation rate, even though the Phoenix mine will have a much lower production rate.

Table 16-14 shows the planned ventilation flows for the two phases of mining.

**Table 16-14: Wheeler River Project – Planned Ventilation Phases**

| Phase | Type | Dimension (m) | Dimension (ft) | Area ($ft^2$) | Velocity (fpm) | Air Flow (cfm) | Air Flow (cms) | Velocity (m/s) |
|---|---|---|---|---|---|---|---|---|
| **Gryphon Mining Phase** | | | | | | | | |
| Gryphon Production Shaft | Intake | 5.50 | 18.0 | 256 | 2,500 | 640,000 | 302 | 12.7 |
| Gryphon Ventilation Raise | Exhaust | 4.50 | 14.8 | 171 | 3,740 | 640,000 | 302 | 19.0 |
| **Phoenix Mining Phase** | | | | | | | | |
| Gryphon Production Shaft | Intake | 5.50 | 18.0 | 256 | 1,000 | 256,400 | 121 | 5.1 |
| Gryphon Ventilation Raise | Intake | 4.50 | 14.8 | 171 | 1,480 | 252,200 | 119 | 7.5 |
| Phoenix Ventilation Raise | Exhaust | 4.50 | 14.8 | 171 | 2,970 | 508,600 | 240 | 15.1 |
| Connection Drift | Transfer | 5.5 x 6.0 | 18.0 x 19.7 | 390 | 1,300 | 508,600 | 240 | 6.6 |
| **Total - to surface** | Exhaust | | | | | 508,600 | 240 | |
| **Total - from surface** | Intake | | | | | 508,600 | 240 | |

For the Gryphon mining phase, heated fresh air will be drawn down the Gryphon production shaft and exhaust air will be routed up the Gryphon ventilation raise. Within the mine, fresh air will flow down the internal spiral ramp and internal fresh air raise, and then be drawn in on each mining sublevel as needed. The internal exhaust air raise will be extended to the depth of the lowest mining sublevel by drop raising. Exhaust air from the mining sublevels will be directed to the internal exhaust raise.

The Gryphon mine layout provides for the following:

- Dust generated at the shaft area truck dump/rock breaker will be directed into the nearby exhaust raise and will not enter the main ramp system.
- The driving of the connection drift to Phoenix is a significant, long term development project. It will be supplied with fresh air from the shaft. Exhaust from development activities will directly enter the main exhaust raise.

For the Phoenix mining phase, the air flow direction in the Gryphon ventilation raise will be reversed (changed from exhaust to intake) and it will be equipped with a mine air heater. Heated fresh air will enter the mine through both the Gryphon shaft and the Gryphon ventilation raise. Fresh air will be sent to the Phoenix mine through the underground connection drift, and ultimately exhausted to surface through the planned Phoenix ventilation raise.

At Phoenix, fresh air will be distributed along strike through the freeze drifts and the jet bore mining drifts. A dedicated exhaust drift is planned along strike on the south east side of the deposit. It will collect exhaust air from mining areas through crosscuts. The exhaust drift will be connected to the Phoenix exhaust raise by a dedicated exhaust crosscut. No mining activity is planned in the exhaust drift or exhaust crosscut, effectively separating workers from the exhaust air streams.

Surface ventilation fans and mine air heaters are described in Section 18.9.

### 16.8.5 Underground Mine Dewatering

The maximum sustained water inflow rate is estimated at 1,500 $m^3/h$. The risk of this occurring is greatest during the Phoenix mining phase with development freeze drifts located below the unconformity and jet bore cavities protected by the tent freeze wall. The mine dewatering system is designed for 1.5 times this rate at 2,250 $m^3/h$.

The main sumps and pumps to surface will be located at the planned Gryphon mine. Phoenix mine water will be transferred to the Gryphon main sumps through two parallel 400 mm (16 inch) HDPE pipe lines installed in the -4% gradient, 2,808 m long connection drift.

The Gryphon main sumps will have sufficient retention time to settle out most of the coarse suspended solids. The main pumps will be centrifugal dirty water pumps arranged in series to meet the duty requirements. Two steel dewatering columns of 400 mm (16 inch) diameter will be installed in the Gryphon shaft, feeding the surface water management ponds. Local area sumps are planned throughout the two planned mines. Drainage holes and submersible dirty water electric pumps will be used to transfer water to the main collection sumps.

### 16.8.6 Underground Power Distribution

Electrical power is required for underground development and ongoing production. Underground power distribution will consist of several main feeders sourced from a main 13.8 kV substation located on surface near the Gryphon shaft.

One feeder will provide power to the main Gryphon surface ventilation fans via a short overhead power line. Variable speed drives will provide the ability to throttle these fans as required to optimize power consumption.

Two additional 15 kV TECK feeder cables from the Gryphon substation will be installed in the Gryphon shaft as primary feeds to the underground mine. Power feeds will be advanced down the Gryphon internal ramp as it is developed, and down the ventilation raise that services the mining sublevels. One of the Gryphon power feeds will be carried by the development crew that drives the connection drift from Gryphon to Phoenix.

A second overhead power line will be constructed from the Gryphon substation to the Phoenix surface exhaust fans, freeze plant, and high grade slurry load out facility. Variable speed drives will be provided on the ventilation fans. A suspended TECK cable will be installed in the Phoenix exhaust raise. It will tie the Phoenix power line into the underground feed installed in the connection drift to provide redundancy to the Phoenix underground power supply. This will provide reliability and switching capability for maintenance and construction.

To facilitate the development activities, each development crew will be provided with independent electrical distribution equipment, which will allow them to proceed at their own pace. A typical crew setup will consist of the following:

- Medium voltage (13.8 kV) junction box, portable power cables and GF protected starters
- Mine portable substation (1 Mva, 13.8 kV/600 V)

The development crews will tie their equipment into the permanent 13.8 kV infrastructure and advance as per schedule. Utilizing 13.8 kV provides the capability to develop much further before needing to establish a shorter tie.

### 16.8.7 Underground Maintenance Shops

Underground mobile equipment will be lowered down the Gryphon shaft. No ramp access to surface is planned. A fully serviced multi-bay underground maintenance shop will be constructed within walking distance of the Gryphon shaft, and a second underground shop will be constructed at the Phoenix mine.

### 16.8.8 Emergency Escapeway

A construction type man hoist will serve as a secondary escape route from underground to surface. It will first be installed in the Gryphon ventilation raise and then moved to the Phoenix ventilation raise as soon as that raise is commissioned (Figure 16-32).


**Figure 16-32: Alimakhek Scando 650 Construction Man Hoist**

### 16.8.9 Refuge Stations

Five permanent refuge stations are planned, and three portable units that can be moved with development crews. One of the permanent stations will be incorporated into the underground shop.

# 17 Recovery Methods

*This entire report section has been reproduced from "Denison Mines Limited, Wheeler River Preliminary Economic Assessment, Process Aspects,", Amec Foster Wheeler Americas Limited, January 21, 2016, an internal company supporting document commissioned by Denison.*

This PEA study is based on the assumption that the Wheeler River project mill feed will be trucked to an existing uranium mill in northern Saskatchewan for processing under a custom milling agreement. There are currently three such facilities in northern Saskatchewan.

The closest existing uranium mill is the Key Lake operation, owned by Wheeler River joint venture partners Cameco (83%) and AREVA (17%). It is about 30 km from the Wheeler River site by road. At present, the Key Lake mill is fully utilized for processing of McArthur River ores and is assumed to be not available for processing Wheeler River deposits.

The furthest existing mill from Wheeler River is at the Rabbit Lake operation, owned by Wheeler River joint venture partner Cameco (100%). It has capacity to process low grade feed such as Gryphon, but is nearing the end of its existing tailings storage capacity. It is uncertain if new capacity will be made available for future potential feed sources.

The third facility, the McClean Lake site's JEB mill, is owned by AREVA 70%, 22.5% Denison, and 7.5% OURD Canada. Upon completion of upgrades in progress during 2015/2016, the JEB mill will have available capacity in both processing and tailings storage, making it the preferred facility.

This section first describes the handling of the coarse mineralization from the Gryphon site, and then the Phoenix slurry process, corresponding with the planned order of production. This section then outlines the McClean Lake milling process, including a description of how this process will handle the Phoenix and Gryphon mill feeds and description of production constraints to be debottlenecked at McClean Lake mill for each feed.

## 17.1 Wheeler River Site Processing

### 17.1.1 Gryphon Deposit Mineralization Handling at Wheeler River

The design for handling Gryphon deposit production is similar to current practice at Cameco's Eagle Point mine. Coarse muck will be skipped to surface for storage on a plant feed pad. For the few stopes that could be exceptionally high grade, underground blending would likely be performed prior to loading the skip.

On surface, the plant feed will be stockpiled and blended as required, and loaded onto tractor-trailer trucks for transport to the McClean Lake mill. The trailer design will match those used currently for hauling "special waste" (low grade material) from McArthur River to Key Lake mill. The regulatory limit for this style of shipping is approximately 2.4% $U_3O_8$. With the average run-of-mine grade from Gryphon predicted to be 1.90%, a moderate amount of blending of low grade with high grade during truck loading would be suitable to stay within this constraint.

At the McClean Lake mill, the existing mill feed pad used for the grinding circuit feed has sufficient stockpile capacity to allow blending for a consistent feed grade.

### 17.1.2 Phoenix Deposit Processing at Wheeler River

The Phoenix deposit mine production will use a water jet bore system (JBS) mining method as described in report Section 16. It is assumed that the process design for Phoenix will closely follow that installed at Cigar Lake.

The first stage of processing Phoenix takes place underground at the Wheeler River site. The mineralized slurry from the JBS will be pumped to the underground crushing and grinding facilities and the resulting finely ground, high density slurry is pumped to surface storage tanks and thickened. It will then be loaded into truck mounted slurry containers, similar to those currently being used for the McArthur River and Cigar Lake mines. Figure 17-1 provides a simplified overview of the Phoenix process steps at Wheeler River site.

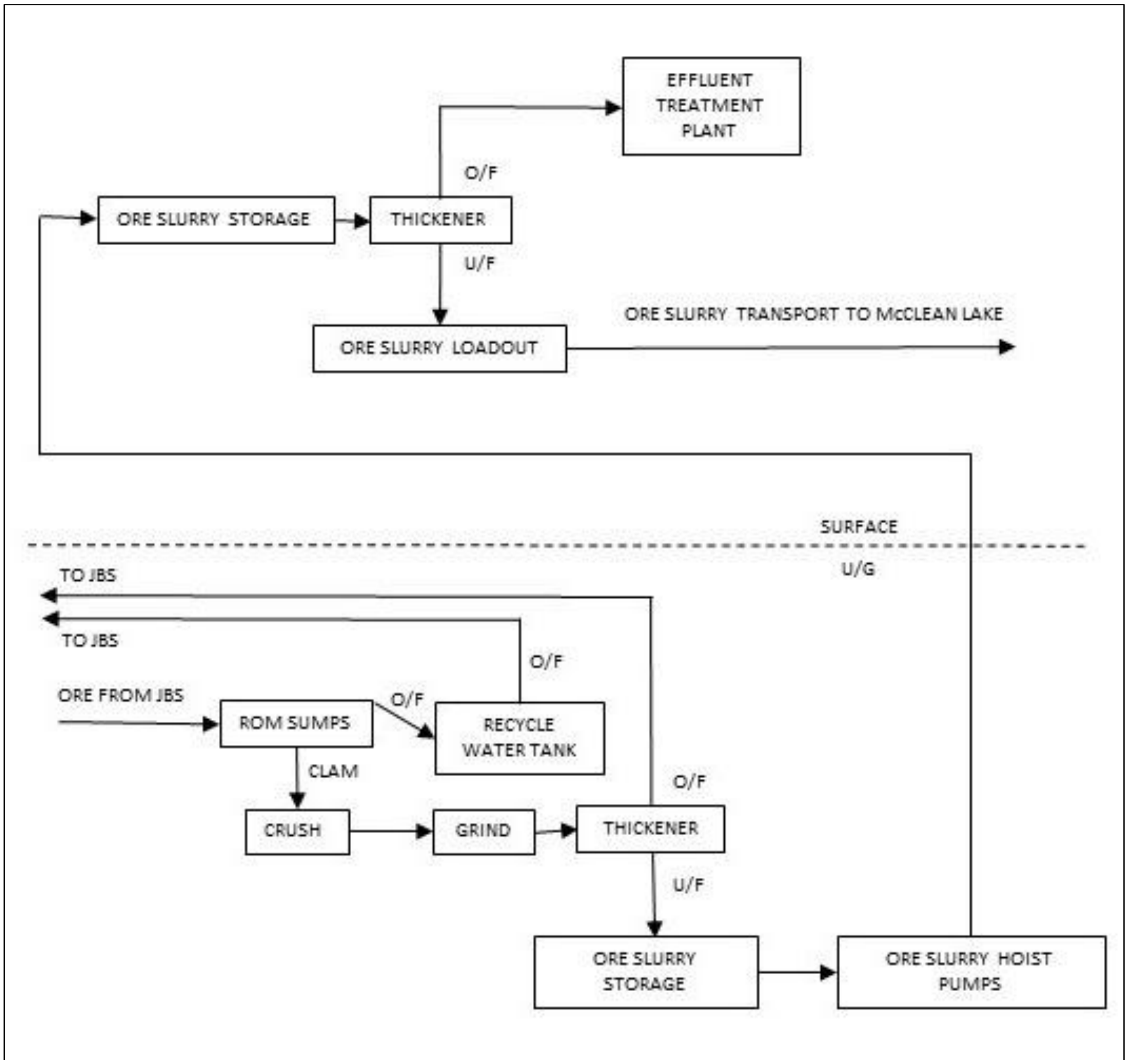



**Figure 17-1: Phoenix Process Overview at Wheeler River**

### 17.1.3 Underground Slurry Handling

The high grade slurry exiting the JBS cavity will flush into the local pump box. The low density slurry will then be pumped to a run-of-mine (ROM) storage sump. Partially dewatered material will be reclaimed from the sumps by an overhead crane mounted clamshell and fed through a water flush cone crusher and on to a ball mill operating in closed circuit with cyclones. Thickened cyclone overflow will be pumped to a high grade slurry storage pachuca tank located underground. From there, the slurry will be pumped up to storage pachucas located on the surface.

As much as reasonably possible, untreated water will be recirculated underground in the process. Overflow water from the ROM storage sump is collected in the recycle water tank for recirculation to the JBS pump box. Overflow water from the underground thickener may be recirculated and filtered for medium and high pressure water supplies to the JBS. Excess water will be pumped to Gryphon and then pumped to the surface water management ponds at Gryphon. Treated water will be utilized in the mining and processing circuits where required.

### 17.1.4 Surface Slurry Handling

Slurry from the surface storage pachucas will be pumped to the mix tank and continuously feed the loadout thickener. The thickener will remove as much water as possible from the slurry prior to container loadout.

The container filling system for Phoenix will replicate the existing systems at Cigar Lake and McArthur River. The thickener underflow will be pumped to the container feed tank. From there, the slurry will be pumped into a loop that will feed the filling stations, located directly above the truck-mounted containers. The slurry containers for transport of Phoenix will be similar to those currently used for the transport from McArthur River to the Key Lake mill and Cigar Lake to the McClean Lake mill.

## 17.2 Transportation

Delivery of the mill feed to the McClean Lake mill will require construction of a 45 km section of haul road between the McArthur River mine and the Cigar Lake mine. The cost for this road has been included in the capital costs estimates. Wheeler River project mill feed will consist of both lower grade coarse dry muck and high grade slurry. Life-of-mine quantities to be trucked are estimated at:

- 975 kt (dry) of low grade coarse dry muck from Gryphon deposit, handled conventionally in covered trucks (Figure 17-2)
- 240 kt (dry) from Phoenix deposit in a high grade slurry (50% solids by weight) in special containers (Figure 17-3)

**Figure 17-2: 40-Tonne Highway Truck – Tarp Covered**
Source: Golder, 2010

**Figure 17-3: B-Train Tractor Trailer with Special Slurry Containers**
Source: Public Document

## 17.3 JEB Mill

### 17.3.1 JEB Mill Process Description

The JEB mill process chemistry is typical of acid leach uranium milling, with design details tailored to high grade Saskatchewan ores. The simplified overview of the mill flowsheet as currently configured is shown in Figure 17-4 below.

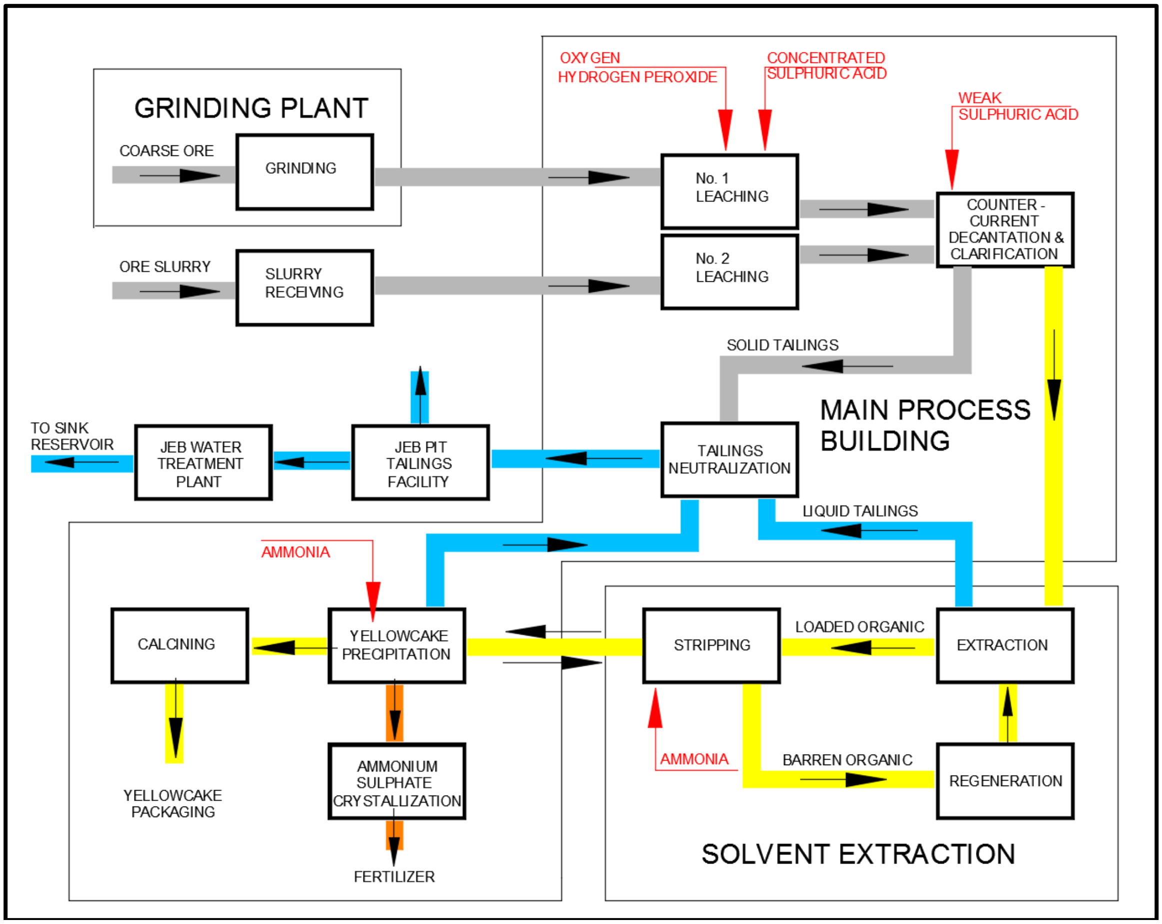



**Figure 17-4: McClean Lake (JEB) Mill Process Overview**

Source: Amec Foster Wheeler, 2015

The grinding circuit for coarse dry muck has a SAG mill and a ball mill in closed circuit with cyclones. The slurry receiving circuit is used for container unloading. The two parallel agitated tank leaching circuits use sulphuric acid from an on-site acid plant, and hydrogen peroxide as the oxidizing agent. An on-site oxygen plant is also installed as an alternate oxidant. A six-stage counter-current decantation (CCD) circuit washes the leach discharge slurry, to minimize soluble losses to tailings. The CCD thickener overflow is clarified and then sent to the solvent extraction (SX) step. The dissolved uranium is purified by extraction into the organic phase using tertiary amine and then concentrated in the ammonium sulfate stripping solution. Anhydrous ammonia is added to precipitate the uranium, which is washed and calcined to yield a high purity $U_3O_8$ product. Ancillary circuits include an ammonium sulfate by-product crystallization plant, tailings neutralization, and effluent treatment. Tailings are placed in the tailings management facility (TMF) that was originally the JEB open pit mine, adjacent to the mill.

## 17.3.2 JEB Mill History

Numerous expansion plans have been completed or contemplated for McClean Lake mill over the years, so a brief outline of the mill's history is presented.

## Phase 1 – Mill Start-up

The JEB mill commenced operations in June 1999 (Badea and Schwartz, 2000; Remple, 2000), with feeds from the JEB and Sue pits grading up to 4% $U_3O_8$. The coarse dry ROM muck was fed to the grinding circuit, with a capacity of over 30 tonnes/hour (t/h). Four 80 $m^3$ slurry storage pachucas were used for surge capacity to accommodate part-time grinding operation, while continuously feeding up to 12 dry t/h design feed rate to leaching.

The original leaching circuit had a two-stage configuration, with three 60 $m^3$ primary leach tanks processing fresh leach feed diluted with CCD overflow solution. The primary leach discharge went to the primary thickener where its overflow advanced to clarification and SX, while the thickened underflow fed the seven 24 $m^3$ secondary leach tanks. The secondary leach discharged to CCD for washing. With operating conditions of 12 t/h feed, at least 40% CCD underflow density (due to low clay content feed) and wash ratio of 4.5 $m^3$ wash water per tonne dry feed, soluble losses were 0.4%.

The Phase 1 design production capacity was 6 M lb/yr $U_3O_8$ from “McClean site source ores” (JEB, Sue, McClean). However, there were design considerations for potential mill expansion to 18-24 M lb/yr, knowing Cigar Lake would be developed over the life of JEB mill.

## JEB Mill Modifications for receiving Cigar Lake Slurry

As of 2010, the following new equipment was installed in anticipation of receiving Cigar Lake slurry:

- Slurry receiving building, four new high grade slurry storage pachucas, and a new neutral thickener
- Oxygen plant to supply oxidant for leaching
- Counter current cyclone (CCC) circuit to supplement CCD for leach residue washing
- Expand from four to six carbon columns for molybdenum removal
- Expansion of existing ammonium sulphate crystallization plant

## JEB Mill Modifications for 100% Cigar Lake Production

Effective November 30 2011, the JEB Toll Milling Agreement was amended by a Memorandum of Agreement to process 100% of Cigar Lake Phase 1 feed (18 M lb/yr $U_3O_8$) to yellowcake at McClean Lake. An additional 4 M lb/yr of “McClean Lake source” or other feed could notionally be co-milled with Cigar Lake, for an intended total production capacity of 22 M lb/yr (Cameco, 2012).

The following expansion construction work is in progress, and assumed to be completed by the time Cigar Lake Phase 1 reaches full production:

- Use one of the four original grinding discharge storage pachucas for Cigar Lake pre-leach.
- Reconfiguration to two parallel low pressure leach circuits to allow Cigar Lake leaching separately from other feed sources. The original seven secondary leach tanks are configured as #2 leach circuit for Cigar Lake ore. The original three primary leach tanks are configured as #1 leach circuit for other feed sources.
- With recent design changes to deal with hydrogen evolution from leaching Cigar Lake ore, it is assumed that hydrogen peroxide will be used as the oxidant in both leach circuits rather than oxygen.
- Cigar Lake (#2 leach circuit) discharge to the primary thickener, with its overflow advancing to clarification and underflow to be combined with potential #1 leach circuit residues for washing

in the CCD circuit. Additional wash capacity in the CCC circuit "can be implemented if required" (Cameco, 2012).
- Additional clarification and storage capacity will be provided for pregnant leach solution.
- A new 17 M lb/yr $U_3O_8$ SX circuit to operate in parallel with the existing SX circuit.
- Expansion from six to eight carbon columns for molybdenum removal.
- Additional precipitation circuit residence time and improved barren strip clarification.
- A new centrifuge for yellowcake dewatering.
- Add a third ammonia reagent supply tank.
- A new ammonium sulphate crystallization plant similar in size to the existing plant.
- Expanded acid plant capacity (to 250-300 tonnes/day).
- New tailings neutralization circuit to provide required retention times.
- Extra capacity for ferric sulphate and barium chloride reagents.

The current focus at McClean Lake operation is on achieving 18M lb/yr production from Cigar Lake Phase 1 feed. It is assumed that by the time Wheeler River feeds would be delivered to McClean Lake, the mill will have proven its capacity for 18 M lb/yr $U_3O_8$ from Cigar Lake production.

## 17.4 McClean Lake Co-milling

It is assumed for this study that Wheeler River feeds will be co-milled at McClean Lake (JEB) mill with ongoing production from the Cigar Lake mine. The toll milling battery limits are from receipt of mineralized coarse dry muck and/or slurry to feed the McClean Lake mill, through to production of yellowcake in drums and tailings to be sent to a tailings management facility (TMF).

### 17.4.1 Mill Feed Rates

The basis of mine production plan feed sources to the McClean Lake mill was established from:

- Wheeler River's Gryphon and Phoenix zones mining production plan, as shown in Table 16-13 of this report
- Cameco's Cigar Lake NI-43-101 technical report, dated February 24, 2012 (Cameco, 2012)

The Cigar Lake mine is currently ramping up shipments to McClean Lake, and yellowcake production began in late 2014. Cigar Lake's production of Phase 1 reserves is approximately one year behind the schedule laid out in Table 16-3 of the technical report (Cameco, 2012), and is offset accordingly for this study. It is assumed for this study that the high grade resources identified in Cigar Lake Phase 2 (Cameco, 2012; Table 14-2) will be brought into production as Phase 1 production ramps down, and lower grade resources will not enter production.

In this scenario, it is assumed that there is no competition for the use of JEB mill's #1 leach circuit in the Wheeler River production time frame. The mine production plans for JEB mill feed are summarized as follows:

- Cigar Lake Phase 1 high grade slurry feed from present to 2028, peaking at 50 kt/yr containing 18 M lb/yr $U_3O_8$ in feed
- Gryphon deposit low grade mineralization feed from 2025 to 2032, peaking at 144 kt/yr containing 6.0 M lb/yr $U_3O_8$ in feed
- Cigar Lake Phase 2 high grade slurry feed from 2028 to 2040, peaking at 25 kt/yr containing 10.8 M lb/yr $U_3O_8$ feed
- Phoenix deposit high grade slurry feed from 2032 to 2041, peaking at 26 kt/yr containing 7.0 M lb/yr $U_3O_8$ in feed

The peak combined mill feed rate is 24 M lb/yr $U_3O_8$, matching the intended license capacity of McClean Lake upon completion of the expansion work in progress as discussed in report Section 17.4.2. The detailed year-by-year production scenario is shown in Table 17-1below.

It is assumed that mill availability is 24 hours per day, 325 days (89%) of the year. The average feed tonnage rates for Gryphon and Phoenix are therefore 18.4 t/h and 3.4 t/h, respectively.

## Table 17-1: Co-Milling Production Scenario

| Wheeler River PEA Production Schedule | | | | Pre-production → | | | | | | Production → | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Scheduled | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 | 2037 | 2038 | 2039 | 2040 | 2041 |
| **MINE PRODUCTION** | | | | | | | | | | | | | | | | | | | | | | | | | |
| **Gryphon** | Kilotonnes | | **975** | | | | | | 61.8 | 144 | 144 | 144 | 144 | 144 | 144 | 51.6 | | | | | | | | | |
| | % U3O8 | | **1.90** | | | | | | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | | | | | | | | | |
| | Mlbs U3O8 | | **40.7** | | | | | | 2.58 | 6.00 | 6.00 | 6.00 | 6.00 | 6.00 | 6.00 | 2.16 | | | | | | | | | |
| **Phoenix** | Kilotonnes | | **240** | | | | | | | | | | | | | 16.3 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 26.2 | 13.9 |
| | % U3O8 | | **12.1** | | | | | | | | | | | | | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 |
| | Mlbs U3O8 | | **64.1** | | | | | | | | | | | | | 4.34 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 3.71 |
| **Wheeler** | Kilotonnes | | **1,215** | | | | | | 62 | 144 | 144 | 144 | 144 | 144 | 144 | 68 | 26 | 26 | 26 | 26 | 26 | 26 | 26 | 26 | 14 |
| **Combined** | % U3O8 | | **3.91** | | | | | | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 4.35 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 |
| | Mlbs U3O8 | | **105** | | | | | | 2.6 | 6.0 | 6.0 | 6.0 | 6.0 | 6.0 | 6.0 | 6.5 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 3.7 |
| **Cigar Ph. 1** | Kilotonnes | | **377** | 41.7 | 39 | 39.4 | 44.8 | 48.9 | 51.6 | 50 | 40.6 | 21.3 | | | | | | | | | | | | | |
| Reserves | % U3O8 | | **18.1** | 20.0 | 21.3 | 20.5 | 18.6 | 13.9 | 14.1 | 16.6 | 20.4 | 21.1 | | | | | | | | | | | | | |
| in mine plan | Mlbs U3O8 | | **150** | 18.4 | 18.3 | 17.8 | 18.4 | 15.0 | 16.0 | 18.3 | 18.3 | 9.9 | | | | | | | | | | | | | |
| * Basis is Cigar 43-101 2012 Table 16-3, offset 1 year | | | | | | | | | | | | | | | | | | | | | | | | | |
| **Cigar Ph. 2** | Kilotonnes | | **317** | | | | | | | | | 17.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | |
| Resources | % U3O8 | | **16.9** | | | | | | | | | 16.9 | 17.0 | 19.6 | 19.0 | 18.5 | 18.0 | 17.5 | 17.0 | 16.5 | 16.0 | 15.5 | 14.3 | 14.0 | |
| Inferred | Mlbs U3O8 | | **118** | | | | | | | | | 6.3 | 9.4 | 10.8 | 10.5 | 10.2 | 9.9 | 9.6 | 9.4 | 9.1 | 8.8 | 8.5 | 7.9 | 7.7 | |
| * Basis is Cigar 43-101 2012 Table 14-2, high grade only assumed after Phase 1 | | | | | | | | | | | | | | | | | | | | | | | | | |
| **MILL FEED** | | | | | | | | | | | | | | | | | | | | | | | | | |
| Cigar | Kilotonnes | | **694** | 41.7 | 39 | 39.4 | 44.8 | 48.9 | 51.6 | 50 | 40.6 | 38.3 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 0 |
| | % U3O8 | | **17.5** | 20.0 | 21.3 | 20.5 | 18.6 | 13.9 | 14.1 | 16.6 | 20.4 | 19.2 | 17.0 | 19.6 | 19.0 | 18.5 | 18.0 | 17.5 | 17.0 | 16.5 | 16.0 | 15.5 | 14.3 | 14.0 | |
| | Mlbs U3O8 | | **269** | 18.4 | 18.3 | 17.8 | 18.4 | 15.0 | 16.0 | 18.3 | 18.3 | 16.2 | 9.4 | 10.8 | 10.5 | 10.2 | 9.9 | 9.6 | 9.4 | 9.1 | 8.8 | 8.5 | 7.9 | 7.7 | 0.0 |
| Wheeler | Kilotonnes | | **1,215** | | | | | | 62 | 144 | 144 | 144 | 144 | 144 | 144 | 68 | 26 | 26 | 26 | 26 | 26 | 26 | 26 | 26 | 14 |
| | % U3O8 | | **3.91** | | | | | | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 4.35 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 | 12.1 |
| | Mlbs U3O8 | | **105** | | | | | | 2.6 | 6.0 | 6.0 | 6.0 | 6.0 | 6.0 | 6.0 | 6.5 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 3.7 |
| **Combined** | **Mlbs U3O8** | | **373** | **18.4** | **18.3** | **17.8** | **18.4** | **15.0** | **18.6** | **24.3** | **24.3** | **22.2** | **15.4** | **16.8** | **16.5** | **16.7** | **16.9** | **16.6** | **16.4** | **16.1** | **15.8** | **15.5** | **14.9** | **14.7** | **3.7** |
| **MILL PRODUCTION** | | | | | | | | | | | | | | | | | | | | | | | | | |
| | Recoveries | | | | | | | | | | | | | | | | | | | | | | | | |
| **Cigar** | 98.5% | Mlbs U3O8 | **265** | 18.1 | 18.0 | 17.5 | 18.1 | 14.8 | 15.8 | 18.0 | 18.0 | 16.0 | 9.2 | 10.6 | 10.3 | 10.0 | 9.8 | 9.5 | 9.2 | 9.0 | 8.7 | 8.4 | 7.8 | 7.6 | 0.0 |
| **Phoenix** | 98.1% | Mlbs U3O8 | **62.8** | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 4.3 | 6.9 | 6.9 | 6.9 | 6.9 | 6.9 | 6.9 | 6.9 | 6.9 | 3.6 |
| **Gryphon** | 97.0% | Mlbs U3O8 | **39.5** | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 2.5 | 5.8 | 5.8 | 5.8 | 5.8 | 5.8 | 5.8 | 2.1 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| Subtotal Wheeler River | | Mlbs U3O8 | **102** | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 2.5 | 5.8 | 5.8 | 5.8 | 5.8 | 5.8 | 5.8 | 6.4 | 6.9 | 6.9 | 6.9 | 6.9 | 6.9 | 6.9 | 6.9 | 6.9 | 3.6 |
| **Total Mill Production** | | **Mlbs U3O8** | **367** | **18.1** | **18.0** | **17.5** | **18.1** | **14.8** | **18.3** | **23.8** | **23.8** | **21.8** | **15.0** | **16.5** | **16.1** | **16.4** | **16.6** | **16.4** | **16.1** | **15.8** | **15.6** | **15.3** | **14.6** | **14.5** | **3.6** |

### 17.4.2 Mill Operating Scenario for Wheeler River Feeds

The current McClean Lake construction permit is for a 24M lb/yr license limit capacity as per the License Conditions Handbook (CNSC, 2014), and an operating permit for this rate is anticipated to be received in 2016. However, the 2011 JEB toll milling design intent of 22 M lb/yr total stated above allows for only 4 M lb/yr through the #1 leach circuit. The potential bottlenecks to increasing from 22 to 24 M lb/yr are most likely to impact production plans for low grade feed from the Gryphon deposit, as discussed in report Section 17.5.2.

The co-milling scenario for Wheeler River largely aligns with the November 2011 configuration described above, with two parallel leach circuits to be upgraded as required. In all likelihood there will be different ownership structures between Wheeler River and Cigar Lake (or any other feed source), so metallurgical accounting is a critical aspect of a co-milling operation. At a minimum, the requirements for accounting are completely separate circuits for feed receiving through leaching for each feed source. It is expected that each mill feed type has slightly different leach recovery, so accurate individual measurements are required.

The #1 and #2 leach circuits' discharges will be washed in the CCD circuit, also upgraded as required, with the CCC circuit assumed to be offline. The combined primary thickener overflow and CCD #1 overflow will be fed to clarification. From there, the pregnant solution will be split between the original SX circuit and the new SX circuit under construction.

The two SX pregnant strip streams will be fed through two parallel sets of carbon columns for molybdenum removal. The streams will be combined to feed the expanded precipitation circuit, and the dewatered ammonium diuranate precipitate feeds the existing calciner and packaging system.

The barren strip bleed stream is fed to two parallel crystallization trains, with the new unit similar in size to the existing train.

Supporting assumptions are:

- A commercial agreement will be established for toll milling Wheeler River feed. A precedent was established by the Cigar Lake joint venture (CLJV) toll milling agreement. As part of this agreement, a metallurgical accounting/reconciliation method will need to be arranged.
- Regulatory approval is assumed to be in place for up to 24 M lb/yr production. According to the current "McClean Lake Operation License Conditions Handbook" (CNSC, 2014), the following activities are licensed up to June 30, 2017:
  - Up to 13 M lb/yr $U_3O_8$
  - Construction activities to modify the mill to increase capacity to 24 M lb/yr $U_3O_8$
- Sufficient tailings capacity is assumed to be made available.
- Sufficient power, water and process supplies are assumed to be available.

## 17.5 Gryphon Deposit Milling

Gryphon feed is assumed to be trucked as coarse dry muck to McClean Lake, and fed through the grinding circuit to the #1 leach circuit. It will be co-milled with Cigar Lake Phase 1 slurry, which feeds the #2 leach circuit.

### 17.5.1 Gryphon Deposit Equipment Design Details

Equipment design details assumed for Gryphon are:

- Grinding - use existing circuit to produce 100% passing 300 microns ($P_{100}$=300 µm).
- Leaching - 48% solids feed, residence time nine hours using peroxide and ferric sulphate addition, minimum four tanks in series. Pre-leach time (acid addition without oxidant) is not included in residence time available, as this is subject to future optimization test work.
- Solid-liquid separation using the existing CCD circuit, to a maximum of 13 t/h - six stages, 40% solids underflow, wash ratio 4 $m^3$ wash solution/dry tonne solids. Supplement CCD with new filter press capacity to handle full co-milling tonnage, to meet soluble loss criteria.
- Assume clarification and downstream circuits are suitable for co-milling upon completion of current mill expansion

### 17.5.2 Gryphon Deposit Production Constraints

#### Grinding

Assuming annual operating availability as per report Section 17.4.1, there is ample grinding circuit capacity at McClean Lake for this rate. However, with two of the four original slurry pachucas re-deployed as pre-leach tanks, there is little grinding discharge surge capacity remaining for #1 leach circuit feed. Grinding would need to be started and shut down every few hours. While this is an operational nuisance, it does not constrain production.

#### Leaching

The low grade circuit as currently configured has a nominal volume of 180 $m^3$ in the three primary leach tanks with peroxide addition. A standard leach circuit design has at least one redundant tank to compensate when one is taken off-line for maintenance.

The Gryphon zone requires a high feed rate, so the residence time of the existing three leach tanks would be less than seven hours. One additional tank of 76 $m^3$ leach volume (or two tanks of 38 $m^3$ each) is required to give nine hours residence time.

#### Solid/Liquid Separation

The JEB mill was designed primarily for a low tonnage of high grade feed, its CCD circuit has the smallest diameter thickeners of the three mills in the Athabasca Basin. This imposes a design constraint on tonnage throughput to avoid substantial soluble recovery loss.

The combined solid/liquid separation circuit solids will receive residues from 18.4 t/h Gryphon leach feed plus 6.4 t/h Cigar Lake Phase 1 leach feed, for a total of 24.8 t/h. This is approximately double the mill start up throughput in 1999 of 12 t/h for McClean Lake's leaching and CCD circuits. It is anticipated that unacceptable CCD recovery loss would occur if it were to be fed at this higher tonnage.

### Clarification and Downstream Circuits

After the pregnant solution is separated from the leached solids residue, the downstream circuits (clarification, SX, carbon columns, precipitation, calcining, packaging, crystallization) can be characterized by volumetric flow rate and yellowcake production rate constraints. However, Amec Foster Wheeler cannot validate the design criteria used for these circuits to calculate their capacities at this time.

Based on the stated new SX plant capacity of 17 M lb/yr to operate in parallel with the existing estimated 10 M lb/yr SX plant, it is a reasonable assumption that the clarification and SX circuits will be able to handle 24 M lb/yr peak production during Gryphon/Cigar Lake Phase 1 co-milling.

Downstream from SX (carbon columns, precipitation, calcining, packaging, crystallization), the scope of expansion according to the November 2011 Toll Milling Agreement appears to support a production rate somewhere in the range of 22-24 M lb/yr. However, milling solely Cigar Lake Phase 1 up to the year 2024 means that these circuits will have only proven their capacity up to 18 M lb/yr. It is assumed for the purpose of this study that all these circuits will be capable of 24 M lb/yr. However, should commercial negotiations proceed, it is recommended that Denison requests AREVA to validate design capacities, and identify any equipment that may require upgrades.

## 17.6 Phoenix Deposit Milling

A B-train tractor-trailer holding four IP-2 certified slurry containers will transport the material to the McClean Lake mill. It will be offloaded in the existing slurry receiving building built for Cigar Lake feed.

### 17.6.1 Phoenix Deposit Equipment Design Details

Cigar Lake Phase 2 slurry will be leached in #2 leach circuit used for Cigar Phase 1, while Phoenix slurry will be leached in the #1 circuit.

Equipment design details assumed for Phoenix are:

- Leaching - 48% solids feed slurry, residence time minimum 12 hours using hydrogen peroxide oxidant and ferric sulphate addition
- CCD - six stages, 30% solids underflow, wash ratio 6.7 $m^3$ wash solution/dry tonne solids OR filter press with cake wash
- Assume clarification and downstream circuits are suitable for co-milling upon completion of current mill expansion

### 17.6.2 Phoenix Deposit Production Constraints

The production demands of Phoenix/Cigar Phase 2 co-milling phase are much lower than the preceding Gryphon/Cigar Phase 1 co-milling phase. Combined peaks are reduced from 24.8 to 6.6 t/h leach feed and 23.8 to 16.6 M lb/yr $U_3O_8$ production. As such there are no constraints anticipated for Phoenix due to prior debottlenecking for Gryphon. Instead, some piping and pump flow rates will require reconfiguration and downsizing.

## Slurry Receiving and Storage

Some small modifications are anticipated with the shared use of slurry container receiving facilities for accounting purposes.

The combined tonnage rate of Phoenix and Cigar Lake Phase 2 is assumed to be very close to Cigar Lake Phase 1, which has four slurry offloading pachucas. It is reasonable to expect that dedicating two of the four high grade slurry pachucas each for Phoenix and Cigar Lake Phase 2 is sufficient capacity, such that no additional storage tankage is required. Rather, some piping modifications would be required to offload trucks to and feed slurry from dedicated storage pachucas, to the two parallel leach circuits. For production accounting, a physical method of assuring that high grade slurry is not offloaded into the wrong pachucas will be required.

## Leaching

It is assumed that the #1 circuit used for Gryphon would also be used for Phoenix, as no other feed is anticipated in this scenario. The #2 circuit would continue to be used for Cigar Lake. This permits separate collection of leach discharge samples to measure leach efficiencies and contained uranium content in solution.

As per report Section 17.4.1, it is assumed the Phoenix feed rate average is 3.4 t/h, with a nearly equal tonnage of Cigar Lake Phase 2 feed assumed to be co-milled. The #1 leach circuit will have been modified for a much higher Gryphon feed rate prior to receiving Phoenix feed. Thus, the actual residence time available for Phoenix will be over 40 hours compared to the 12 hours required. This large excess available capacity could be used to increase Phoenix leach recovery, pending future optimization test work.

## Solid/Liquid Separation

The tonnage for Phoenix/Cigar Lake Phase 2 co-milling is far less than the preceding Gryphon/Cigar Lake Phase 1 feeds, so ample capacity will be in place. The most likely choice is to continue to use the Cigar Lake residue filter press capacity assumed to be installed for Gryphon/Cigar Lake Phase 1 co-milling, to maximize Cigar Lake recovery. For Phoenix, a leach feed rate of 3.4 t/h is low enough to not make CCD a constraint, even with poor settling expected. An alternate option would be to feed Phoenix residue to the second filter press and take CCD offline.

## Clarification and Downstream Circuits

All downstream circuits will have a proven capacity of 24 M lb/yr, compared to a combined feed rate of approximately 17 M lb/yr for Phoenix/Cigar Lake Phase 2. No constraints are anticipated.

# 18 Surface Infrastructure

## 18.1 Access Road and Site Preparation

Saskatchewan Highway 914 is the main access road in the Wheeler River project area (Figure 18-1). The highway serves as a haul road from the McArthur River mine to the Key Lake processing facility 35 km to the southwest. Access to the Wheeler River project site is provided by gravel roads that branch off from the Key Lake to McArthur River haul road.

These site access roads will be upgraded for the start of construction on the Gryphon site and camp. Later, the selected haulage route from Gryphon to Highway 914 will be upgraded as required to a standard suitable for hauling uranium mineralization.

Site preparation earthworks will first be undertaken at the Gryphon mine site and selected camp site.

## 18.2 Project Site Layout

Figure 18-1 is a plan view of the Wheeler River project area showing the exploration camp area and the Phoenix and Gryphon deposits located within the claim boundaries. Phoenix site is located 3 km to the southeast of the Gryphon site.

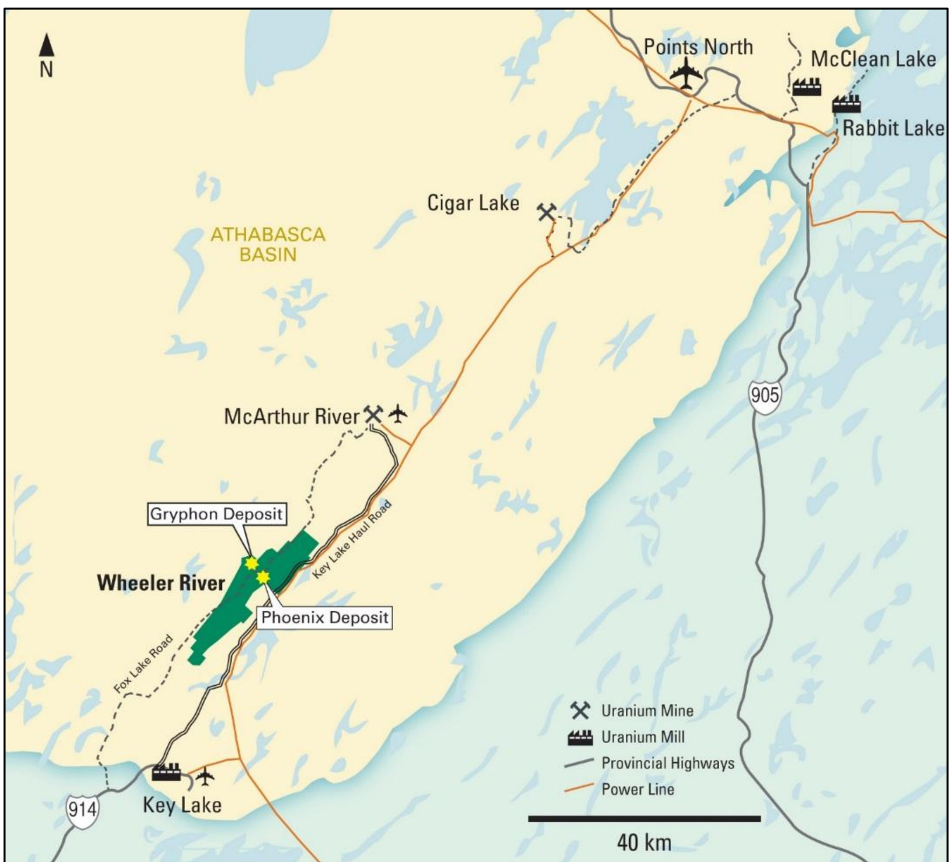



**Figure 18-1: Wheeler River Project Site Showing Phoenix and Gryphon Deposits**

## 18.3 Phoenix Site Layout

Figure 18-2 is a conceptual layout of the Phoenix site showing the relative scale of major infrastructure items including:

- The collar of the 4.5 m (15 feet) diameter ventilation raise which will be equipped with dual exhaust fans
- A 19,000-tonne stockpile of blind bore cuttings created while excavating the ventilation raise (this material could be reclaimed early)
- An emergency man hoist installed in the ventilation raise
- A freeze plant producing chilled brine for the Phoenix tent freeze
- A slurry load out facility for Phoenix high grade slurry pumped to surface
- An overland power line feeding power to the ventilation fans, freeze plant, and slurry load out building

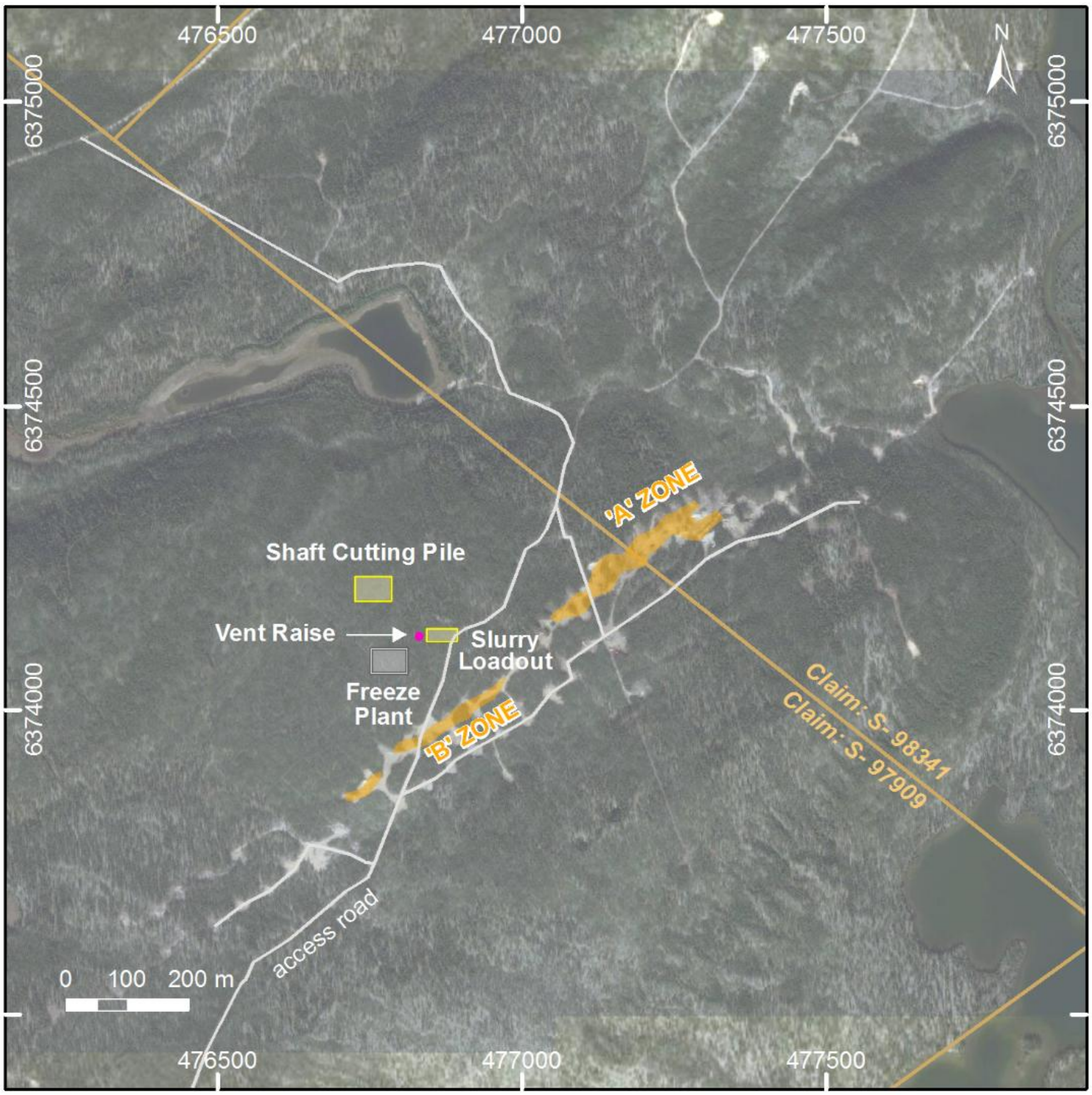



**Figure 18-2: Phoenix Site Conceptual Layout**

## 18.4 Freeze Plant

A surface freeze plant will be constructed at the Phoenix site to service the underground tent freeze layout. The plant will consist of 5 self-contained freeze plants, portable skid mounted units. Each plant is planned to have a power rating of 250 tons of refrigeration (TR) or 880 kW.

300 mm diameter insulated brine circulation piping from surface, approximately 440 m in length will be installed in the Phoenix ventilation raise.

## 18.5 Slurry Load Out Building

A high grade slurry load out facility will be constructed on surface next to the Phoenix ventilation raise. Slurry from the Phoenix jet boring operations will be pumped to surface through steel piping installed in the Phoenix ventilation raise. The emergency man hoist installed in the ventilation raise will be an Alimak type of unit with a work platform suitable for maintenance of piping in the raise.

The load out facility will receive slurry pumped from underground to be stored in tanks until transport trucks carrying special slurry containers enter the building. The slurry transport trucks will be comprised of a tractor, trailer, and four 5.5 $m^3$ Industrial Package Type 2 (IP-2) compliant slurry transport containers. These containers will meet the requirements set out by the International Atomic Energy Agency (IAEA), the NSCA and Regulations, and the Transportation of Dangerous Goods Act and Regulations.

The slurry containers will be transported 160 km to the McClean Lake processing facility.

## 18.6 Gryphon Site Layout

Figure 18-3 is a conceptual layout of the Gryphon site showing the relative scale of major infrastructure items. The waste rock pile is sized to accommodate 300,000 t of rock. The four water management ponds have a combined capacity of 300,000 $m^3$.

The grey access road shown ties into the existing Gryphon exploration road in the lower right corner of the figure. For reference, Kratchowsky Lake is located just south of the area shown in the figure.


**Figure 18-3: Gryphon Site Conceptual Layout**

## 18.7 Infrastructure at Gryphon Mine Site

The following infrastructure is planned and included in the capital cost estimate:

- Headframe and collar house for the 5.50 m (18 feet) diameter production shaft
- Hoisting plant (including air compressors)
- The collar of the 4.50 m (15 feet) diameter ventilation raise which will be equipped with dual ventilation fans
- Mine ventilation fans and propane fired air heaters
- Camp facilities
- Administration office
- Change house
- Maintenance shop

- Warehouse and cold storage
- Emergency facilities building (nursing station, mine rescue facilities, firefighting services, emergency vehicles garage)
- Core logging building
- Laboratories (water/water treatment plant effluent, and environmental services)
- Security gate house
- Truck scales
- Fuel storage and dispensing facility
- Electrical power supply
- Water supply
- Water management facilities including a treatment plant and fire water storage/pumping
- Storage pads for mined mineralization and waste rock
- Backfill plant
- Explosives and detonator magazines

## 18.8 Gryphon Production Shaft

Surface infrastructure at Gryphon will include a production shaft headframe, collar house, and hoist house. The shaft will be set up with a double drum production hoist and a service hoist for men and materials.

The Gryphon shaft will have a nominal hoisting distance of 600 m. It will be equipped with a 1,000 kW (1,300 hp) production hoist and two 5-tonne skips. One skip will be used for hoisting low grade mineralization, and one will be used for hoisting waste rock. This will minimize the cross-contamination of the clean waste rock being hoisted.

The loading pocket will be fed by a transfer conveyor. The conveyor will have two dedicated feeders, one for low grade material and one for waste rock.

The headframe will be equipped with a steel bin and truck loading chute for low grade production, while waste rock hoisted will be deposited on the ground next to the headframe.

## 18.9 Ventilation Fans and Mine Air Heaters

For the first phase of mining at Gryphon, the Gryphon production shaft will be a heated fresh air intake, equipped with a direct fired propane fired mine air heater rated at 59.5 MBTU/h. Exhaust will be pulled from the mine through the Gryphon ventilation raise equipped with dual 2.18 m (86 inch) diameter fans with 336 kW (450 hp) motors.

For the second phase of mining at Phoenix, heated fresh air will be drawn into the mine through both the Gryphon production shaft and the Gryphon ventilation raise. Note that the air flow direction in the Gryphon ventilation raise will be reversed to accomplish this.

Exhaust air for phase two will be through a Phoenix exhaust raise. The overall ventilation system will be a push/pull system with dual axial flow fans installed on both of the Gryphon intakes and the Phoenix exhaust ventilation raise.

For this phase, the Gryphon intake ventilation raise will be equipped with a 17.5 MBTU/h direct fired propane heater and dual 2.18 m (86 inch) diameter fans with 75 kW (100 hp) motors. The Gryphon production shaft will be equipped with dual 1.83 m (72 inch) diameter fans with 150 kW (200 hp) motors. The Phoenix exhaust ventilation raise will be equipped with dual 2.18 m (86 inch) diameter fans with 260 kW (350 hp) motors.

Fan installations will include inlet bells, discharge cones, backdraft dampers, ultrasonic flow monitoring, static pressure sensing, and fan/motor vibration sensors.

## 18.10 Camp

A fully serviced camp is planned for the Gryphon mine area, sized to accommodate a workforce of up to 230 during operations. The camp area will include a dining hall, kitchen, recreation, and exercise and entertainment facilities.

## 18.11 Maintenance Shop

The maintenance shop will be located adjacent to the warehouse facility and will include office space, instrument and electrical shops, and a mechanical shop. The maintenance shop will include service bays capable of accommodating haul trucks, a wash bay, welding shop, lube bay and tool crib.

The Wheeler River project does not include a ramp from surface to underground. Two fully equipped underground maintenance shops (at Gryphon and Phoenix deposits) are planned and included in capital for servicing underground mining equipment.

## 18.12 Warehouse

The warehouse facility will be designed to accommodate the material handling and storage needs for the project. This will include provision for forklift use, any necessary overhead cranes, laydown areas, distribution points, and shelving. The preliminary design of the warehouse facility contains both heated and cold storage facilities.

An outdoor storage area is planned immediately adjacent to the main warehouse. This area will accommodate items not susceptible to the elements or too large to be stored in the warehouse properly. Access to the laydown yard will be via large gate access from the site yard (lockable) and via doors from the main warehouse.

A hazardous materials pad is planned in the same area. This area will receive waste such as used oil, fluorescent bulbs, or other designated substance requiring storage prior to shipping off site. The hazardous materials storage pad will be lined and bermed to contain any spills.

## 18.13 Emergency Facilities

The emergency facility is planned to provide a centralized location for first aid/nursing, fire services, mine rescue, and the associated emergency vehicles (ambulance, fire truck and mine rescue truck). A full set of mine rescue equipment is included in project capital.

Additional space is planned for storage of critical and controlled emergency supplies.

## 18.14 Drill Core Logging Building

The core logging building will be located close to the service shaft collar house to facilitate the handling of core from the mine and to eliminate the potential for contaminating other work areas across the mine site. The core logging building (and laboratories) will be sized to accommodate a forklift. The logging building will be equipped with appropriate logging benches.

The ventilation system for the building will operate on a single pass basis with no recirculation of air to maintain acceptable air quality. There will be a fenced outdoor core storage area for the secure storage of core that will include a designated area for the storage of higher grade core.

## 18.15 Security House and Truck Scales

Access to the property will be controlled by a security gate to be manned 24 hours per day. Truck scales will be installed in the same area as the gate house to permit accounting of stockpiled or shipped materials (mineralized plant feed, waste rock, or “special waste”).

The truck scale installation will be arranged with dual truck scales with an unattended scale monitoring package. Scales will be of sufficient size to accommodate B-train haulage trucks. In addition to the truck scales an automated radiological scanning system will be installed to assist with tracking and monitoring of production shipments.

## 18.16 Fuel Storage and Dispensing

Gasoline and diesel fuel will be stored in approved above-ground containers having sufficient secondary capacity to contain any leaks from the primary tank. Stand alone emergency eyewash facilities will be included. Fuel storage tanks will be sized to ensure that with consideration of draw down rates and freeboard that the tanks can receive refill volume multiples equivalent to full tanker trucks.

Propane tanks for mine air heating will be located near the mine access road a safe distance away from the Gryphon production shaft. It is assumed that the storage tanks and vaporizers will be rented from the propane supplier.

## 18.17 Electrical Power

An existing power line runs along the haul road (Saskatchewan Highway 914) that links the Key Lake and McArthur River sites, but it is essentially fully utilized at present. However, by the end of 2016 a new 230 kV transmission line with about 35 MW of capacity will be operating along this same corridor. This new line will provide sufficient capacity for the Wheeler River project at a provincial industrial rate, expected to be approximately $0.06/kWh.

A branch power line from the existing 230 kV transmission line corridor, following the existing Wheeler River site access roads, would be approximately 11 km in length. The optimal power line route has not been studied.

## 18.18 Back-up Electrical Power

The project capital cost estimate includes 2 MW of back-up power provided by diesel generators. In the event of a power supply interruption, back-up power would be used for maintaining limited mine ventilation, evacuating personnel from underground using the Gryphon service hoist, and for maintaining the underground and surface water management systems.

## 18.19 Water Supply

A fresh water supply source will be established to meet industrial needs. The two underground mines can likely meet most of their needs by recycling a portion of the water collected underground.

A potable water supply system will be provided in compliance with Saskatchewan provincial drinking water standards. The main areas of consumption will be the accommodations camp and the mine change house. The method of water treatment will depend on water quality analysis from the selected source.

## 18.20 Water Management

The underground mine workings will intercept groundwater. Groundwater inflows will vary depending the permeability of the rock and connectivity to more permeable zones. Groundwater information and assessment are presented in report Section 16.1. Routine inflows are likely to range between 10 to 20 $m^3$/h. Episodic non-routine inflows of groundwater to the underground workings could occur when mining encounters structures with connectivity to more permeable zones. The magnitude of these potential inflows is difficult to quantify but could be as great as 1,500 $m^3$/h. The water treatment plant has been sized to treat an influent rate of 1,500 $m^3$/h consistent with the maximum expected non-routine water inflow.

Mine water is expected to have concentrations of metals, uranium, molybdenum, selenium, arsenic and radium that will require treatment to discharge the water. Concentrations of blasting residuals (nitrate and ammonium) may also be elevated and require treatment, but for the purposes of this assessment it is assumed good management practices will limit these constituents to concentrations that do not require treatment prior to discharge.

Underground water inflows to the Gryphon and Phoenix mines will be collected underground and pumped to surface using the Gryphon shaft. Mine water will be discharged into two large holding/surge ponds. The ponds are each planned at 150 m by 150 m and can hold a water depth of 5 m with adequate freeboard to prevent overtopping. The total volume of the surge ponds is 225,000 $m^3$. At the maximum inflow rate of 1,500 $m^3$/h the ponds will fill in a little over six days, if water is not treated and discharged from the system.

The purpose of the holding ponds is to provide surge capacity to dampen variation of the influent rate to the water treatment plant. The operational target is that the ponds should always have 75% of their total volume available in the event of a large inflow to the mine. When 25% of the total volume (56,250 $m^3$ or 117 days of mine inflows of 20 $m^3$/h) is reached, the treatment plant will be started to treat the stored volume. At a nominal treatment rate of 1,000 $m^3$/h this volume can be treated in about a week, including start up and shut down time.

Water from the surge ponds will be pumped to the treatment system. The treatment system has a series of chemical processes to remove the constituents of concern. This conceptual design is based on water treatment at operating uranium mines. The design will be refined as estimates of mine water chemistry and flow are refined in later phases of project development. Figure 18-4 shows a conceptual process flow diagram for the treatment system.

The first process is to precipitate metals by increasing the pH by adding lime. Flocculant will be added to the reactor tank overflow prior to clarification. Sludge from the clarifier will be pumped to a dedicated sludge storage tank before the filter press.

The second process is the addition of ferric sulphate to co-precipitate arsenic, molybdenum, uranium and selenium. The pH will be depressed by the addition of sulphuric acid to less than 5 to enhance the adsorption of these constituents onto ferric hydroxide. Flocculant will be added to the overflow from the reactor prior to clarification. Sludge from the clarifier will be pumped to a dedicated sludge storage tank before the filter press.

The final process is to remove radium. Radium will be removed by co-precipitation with barium sulphate. Barium chloride will be added. The pH will also be adjusted to meet discharge limits in this step. Flocculant will be added to the overflow from the reactor prior to clarification. Sludge from the clarifier will be pumped to a dedicated sludge storage tank before the filter press.

Sludge from the three processes will be filter pressed individually by either a plate and frame or belt filter press. Pressed sludge will be disposed of on site, potentially in completed underground workings. Final disposal options will be evaluated as the project progresses.

Treated water will be discharged to one of two treated water monitoring ponds. The treated water monitoring ponds will be 85 m by 85 m and 5 m deep in addition to adequate freeboard to prevent overtopping. Once a monitoring pond is filled, treatment plant effluent will be discharged to the other monitoring pond and the filled pond will be sampled to ensure water quality compliance objectives are met. The pond will be discharged to the receiving environment once compliance is demonstrated.

The water treatment plant will include a fire water storage tank and fire pump with standby diesel pump. The fire water will be distributed on site through buried HDPE piping with hydrants suitably spaced throughout the site. The fire water reservoir will provide at least thirty minutes of fire protection at maximum flow.

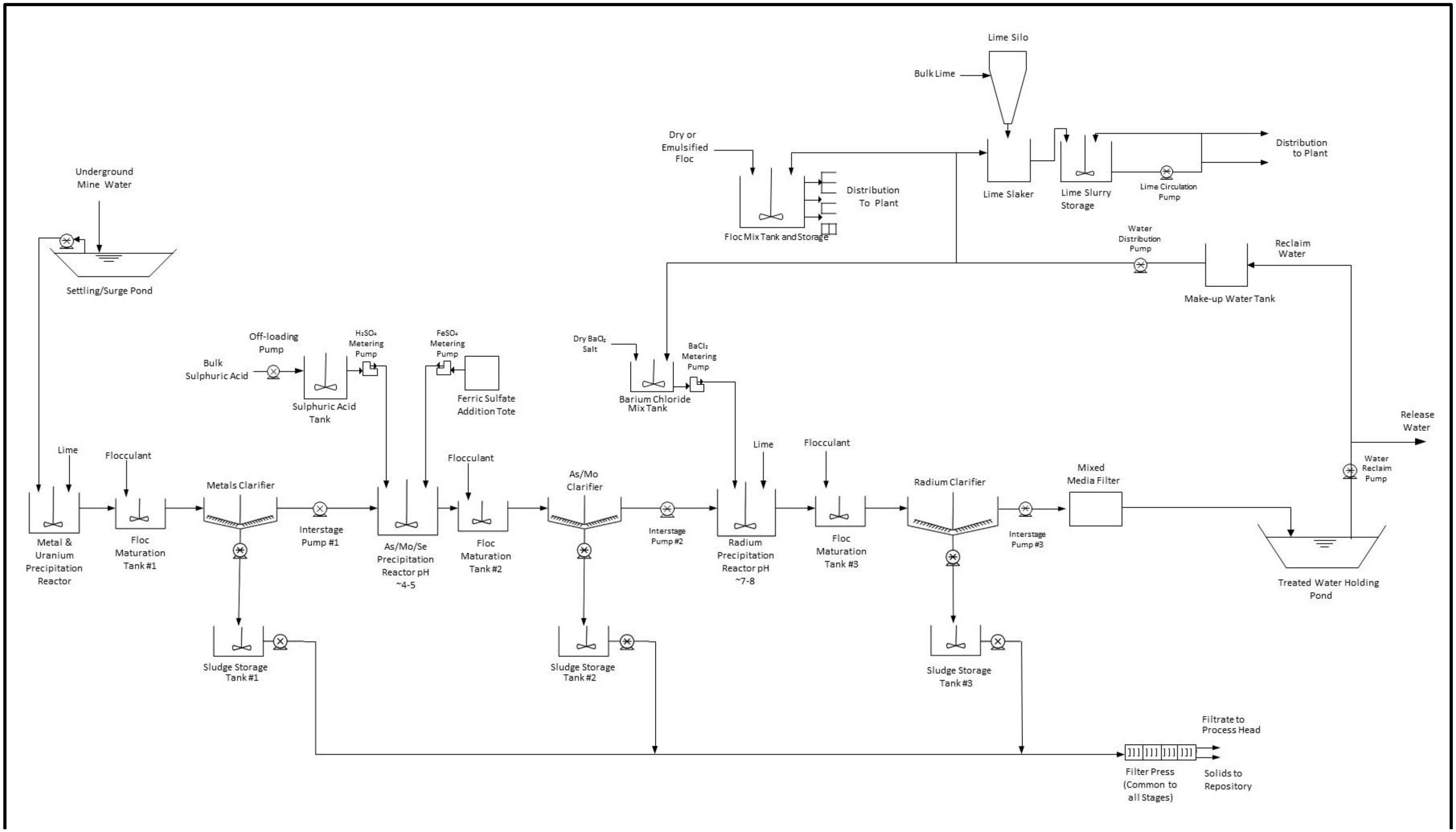



**Figure 18-4: Water Treatment Plant Conceptual Flow Diagram**

## 18.21 Development Waste Rock Management

The ML/ARD potential of the waste rock is described in Section 20.4.3. Waste rock is expected to include a mixture of both clean material and material with potential for ML/ARD. It is also expected that there would be modest amounts of special waste — which is waste rock that contains a uranium content greater than 0.03% $U_3O_8$. There will be rock handling systems for special waste and waste rock to prevent cross contamination, as described in Section 16.8.2.

Denison will implement a development rock monitoring program based on the recommendations provided in the "Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials," MEND Report 1.20.1 (MEND, 2009) to establish procedures and testing to identify, segregate, and properly manage each type of waste rock encountered during mine development.

Waste rock management on surface will be required only at the Gryphon site. The only exception to this will be the one time accumulation of cuttings at Phoenix resulting from the blind boring of the Phoenix ventilation raise. The cuttings will amount to approximately 19,000 tonnes.

It is expected that special waste will be stockpiled and is either blended with the high grade mineralization during processing or used as backfill, and that waste rock with ML/ARD potential would also be stockpiled and used for backfill. In both cases, it is assumed that storage on surface at the Gryphon site, will require separate, double-lined storage pads with integral leak detection and be surrounded by a perimeter ditch designed to contain any runoff from the pad. Drainage from the area will be collected in a sedimentation ponds and treated with the other mine effluent.

Clean waste rock is likely to be used for construction or road surfacing.

## 18.22 Handling Infrastructure for Mined Materials

Gryphon low grade mineralization will be hoisted to a steel bin in the headframe. The bin will feed a truck loading chute. A run-of-mine, lined storage pad will also be constructed next to the Gryphon shaft to provide some blending opportunity and some storage capacity between hoisting and truck loading. Mined low grade mineralization will be trucked to the mill for processing on a regular basis.

## 18.23 Concrete Batch Plant (Backfill)

A small concrete batch plant will be constructed at the Gryphon mine site to prepare backfill for the Phoenix jet bore mining cavities. Figure 18-5 is a schematic view of a concrete batch plant that will be winterized. A ready mix truck will deliver concrete to a slick line installed in the Gryphon shaft

A contractor will prepare aggregates on surface by crushing and screening mine development waste rock.

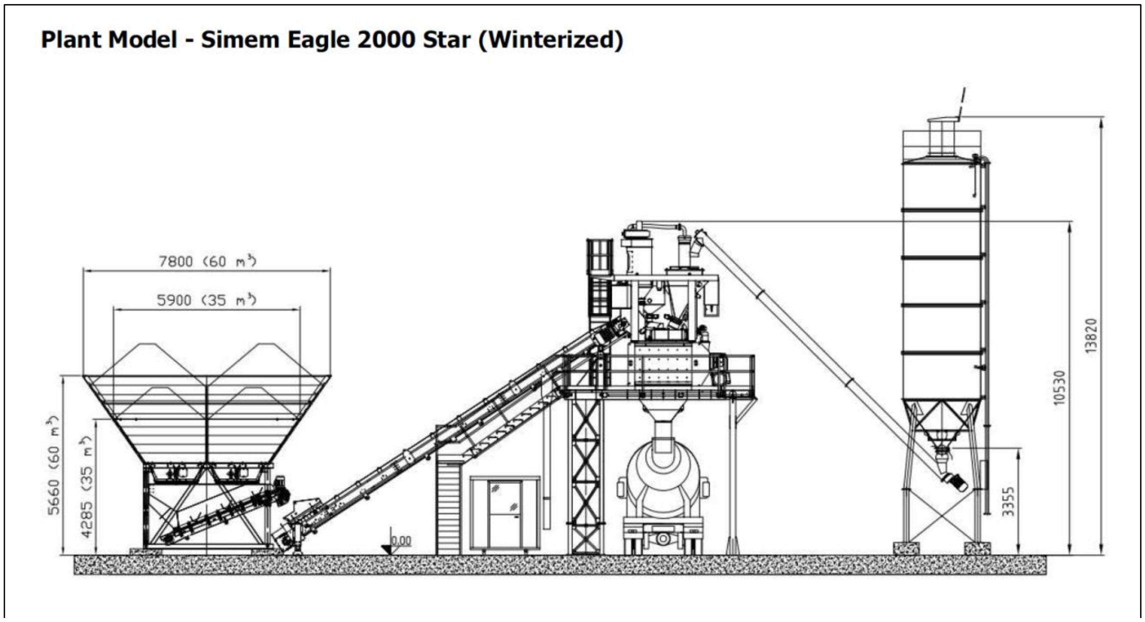



**Figure 18-5: Schematic View of Concrete Batch Plant**

## 18.24 Explosives Magazines

A surface explosives storage facility will be required to support underground development and production activities. The manufacturing of explosives at site will not be required due to the low volume of explosives consumed. This facility would be for storage of materials only. The location of this facility is located away from the surface buildings. The storage facility will be constructed in accordance with federal and provincial regulations and will contain an explosives magazine and a separate magazine for blasting accessories. Access leading to this area will be located within the surface lease, and therefore, will prevent unauthorized access. As the project progresses it is envisioned that this facility could be decommissioned as the explosives can be transported directly to the Gryphon headframe and stored underground in the appropriate magazines.

# 19 Market Studies and Contracts

## 19.1 Marketing

### 19.1.1 The Uranium Industry

In 2015, the focus of the nuclear energy and uranium industries remained on Japan. During this year, however, attention was focused on the number of Japanese nuclear reactors that were brought back on to the grid, as opposed to the 54 reactors that were shut down following the Fukushima Daichii nuclear incident that occurred in March 2011. In June 2015 the Japanese government approved a draft plan for electricity generation to 2030, which calls for nuclear to provide roughly 20-22% of the country's power, and in September 2015, the Japanese nuclear energy industry achieved a significant milestone with the commercial restart of Kyushu Electric Power Company's Sendai Unit 1 reactor. The restart at Sendai Unit 1 was followed by the restart of the Sendai Unit 2 reactor in November 2015 and Kansai Electric Power Company's Takahama Unit 3 reactor in February 2016. These restarts provide significant encouragement for the nuclear energy industry in Japan, which through various companies are in the process of completing modifications and obtaining licences and approvals to bring over 20 additional nuclear power plants online.

With Japan returning to nuclear power generation in 2015, the focus for the industry has started to turn to China, India and Russia, each of which has adopted ambitious plans to increase the use of nuclear power. In China and India, nuclear power is seen as a preferred choice to provide reliable base load power and address an emerging crisis around a lack of clean air and a growing problem with greenhouse gas emissions.

According to the World Nuclear Association ("**WNA**"), as of March 1, 2016, China had 30 operable nuclear reactors capable of producing 26.8 gigawatts of electricity. A further 24 reactors are under construction and an additional 178 reactors are either planned or proposed. Ux Consulting Company, LLC ("**UxC**") estimates that 122 reactors are expected to be operable and capable of producing up to 129 gigawatts of electricity by 2030, representing 5 times as much power capacity as is currently available from nuclear. To achieve this level of production, China's fleet of nuclear reactors will have to increase by approximately 6 or 7 reactors each year for the next 15 years. The WNA is projecting a similar growth profile for India, where 21 reactors were operable as of March 1, 2016, capable of producing 5.3 gigawatts of power. Taken together, 66 reactors are either under construction, planned or proposed in India. UxC estimates that over 22 gigawatts could be operable by 2030, representing over 4 times as much power capacity as is currently available from nuclear. To achieve this level of production, it is estimated that India's fleet of nuclear reactors will have to increase by 19 reactors over the next 15 years – meaning that at least one additional reactor will have to join the fleet each year.

Throughout 2015, the spot price of uranium has sustained itself well above the lows of $28 per pound $U_3O_8$ noted in mid-2014. While the spot price increased during the first quarter of 2015, to near $40 per pound $U_3O_8$, it softened somewhat during the second through fourth quarter of the year, to finish the year at $34.25 per pound $U_3O_8$. The softness in the spot market continues to reflect the fact that the market is currently oversupplied as a result of a combination of factors, including production being sold into higher-priced long term contracts, supply coming from secondary sources, and the impact of a strengthening US dollar. The strengthening of the US dollar provides several producers with the opportunity to sell into the spot market at significantly higher prices, in their local currency, than would have been possible in past years. In Canada, for example, the spot price per pound $U_3O_8$ in Canadian dollars has increased by over 65% to roughly CAD$50 per pound $U_3O_8$ from the low of CAD$30 per pound $U_3O_8$ noted in mid-2014.

Although the uranium market is currently oversupplied, the long term growth projections for the nuclear industry combined with the expected depletion of uranium resources in operation today, continue to suggest that a significant long term supply shortage could emerge, even with new production sources expected to come online. With a sustained period of low commodity prices, the uranium mining industry has been challenged to discover and advance the new production sources necessary to meet the expected increase in demand in future years. Higher prices are expected to be required to justify the construction of new mines and in the absence of a significant price increase in the near term, it is possible that even the most ambitious development plans could leave the market with an unavoidable supply shortage as soon as the early 2020s.

**Uranium Demand**

The WNA reports that there are 440 nuclear reactors operable in 30 countries as of March 1, 2016. These reactors can generate 384 gigawatts of electricity and supply over 11% of the world's electrical requirements. As of March 1, 2016, 65 nuclear reactors are under construction in 14 countries with the principal drivers of this expansion being China (24 reactors under construction), Russia (8), India (6), the United States (5), United Arab Emirates (4) and South Korea (3). Based on the most recent statistics from the WNA, there are a total of 238 reactors that are either under construction or planned around the world, and an additional 337 reactors that are proposed with the potential to be operating by 2030.

According to UxC, in its "Uranium Market Outlook – Q1 2016" (the "**Q1 Outlook**"), global nuclear power capacities are projected to increase by 39%, from 379.4 gigawatts in 2015 to 527.8 gigawatts in 2030. Of the net growth in nuclear generation capacities, China accounts for 70% while India, Korea and Russia collectively make up a further 25%. The Q1 Outlook also estimates that uranium demand, including estimated inventory buildup, could grow by over 30% to as high as 257 million pounds $U_3O_8$ by 2025. This represents an increase of over 50% from estimated demand, excluding inventory buildup, of 168.5 million pounds of $U_3O_8$ in 2015.

**Primary Uranium Supply**

According to the Q1 Outlook, uranium production increased year over year from 145.3 million pounds $U_3O_8$ in 2014 to 158 million pounds $U_3O_8$ in 2015. Factoring out the additional production associated with the ramp up of activities at the Cigar Lake mine, global production remained roughly flat from 2014. Production from Africa, and the United States declined in 2015, while production from Australia, Russia and Kazakhstan remained relatively consistent. Cigar Lake increased production from Canada. Canada remains the second largest producing nation with nearly 22% of the world's production from 2015 coming from within Canada. Kazakhstan continues to be the world's largest producer of uranium, representing nearly 40% of production in 2015.

UxC has estimated in its Q1 Outlook that existing mine production, plus new planned and potential mine production, will increase primary uranium supply from 158 million pounds $U_3O_8$ in 2015 to 165.7 million pounds $U_3O_8$ by 2025. This represents an increase of approximately 4.9%, as compared to the dramatic increases in uranium demand noted 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 Outlook, the main drivers are now limited to the Cigar Lake mine in Canada, which is expected to increase production up to 18 million pounds $U_3O_8$ per year, and the Husab mine in Namibia, which is being built by a Chinese utility as a source of captive supply and continues to be projected to start production in 2016. For other projects to move forward to meet the 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.

### Secondary Uranium Supply

Primary mine production supplies approximately 94% of current demand, excluding inventory buildup. The balance of demand is 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.

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 shut down 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.

Reprocessing of spent fuel is another source of secondary supply but is expected to satisfy roughly 6% of demand. Expansion of this secondary source would require major investments in facilities which could only be supported by a significant increase in long-term uranium prices.

UxC expects that secondary sources of supply will fall from 2015 levels of 44.3 million pounds $U_3O_8$ per year to 30.8 million pounds $U_3O_8$ per year by 2025.

### 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 that uncovered demand is only 3.4 million pounds $U_3O_8$ or 2% of projected demand, including inventory buildup, in 2016. Uncovered demand, however, is projected by UxC to increase significantly over the period of 2016 to 2020, such that up to 72.9 million pounds remains uncovered for 2020, representing roughly 38% of projected demand in that year. Uncovered demand rises rapidly for years after 2020 to 173.6 million pounds for 2025, representing over 80% of projected total demand. At 173.6 million pounds, the uncovered demand in 2025 is estimated to be over 100% of total demand, excluding inventory buildup, from 2015 and approximately 7.9 million pounds $U_3O_8$ greater than the total production expected from new and existing mine production in 2025 – some of which is already committed to the covered portion of the demand projected in 2025. 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, roughly 40 million pounds $U_3O_8$ equivalent were purchased on the spot market per year and approximately 200 million pounds $U_3O_8$ equivalent were contracted in the long term market each year. By comparison, from 2011 to 2015, on average, roughly 48 million pounds $U_3O_8$ equivalent have been purchased on the spot market per year, while less than 100 million pounds $U_3O_8$ equivalent were contracted in the long term market on each year. In 2014 and 2015, long term contracting volumes were roughly 78 million pounds $U_3O_8$ per year. With low contract volumes in recent years and increasing uncovered requirements, we expect that long term contracting activity will have to increase in the 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 2015 at $49.00 per pound $U_3O_8$. On historically low volumes, as noted above, the long-term price declined to $44.00 per pound $U_3O_8$ by the end of the year.

Electric utilities procure their remaining uranium requirements through spot and near-term purchases from uranium producers, traders and other suppliers. Historically, spot prices are more volatile than long-term prices. The spot price began the year at $35.50 per pound $U_3O_8$. It rose to $39.50 per pound $U_3O_8$ during the beginning of the year and then declined to $34.25 per pound $U_3O_8$ by the end of the year and was last quoted at $29.60 per pound $U_3O_8$ on March 21, 2016.

Given the strengthening of the US dollar relative to the currencies of the majority of the uranium producing countries (including Kazakhstan, Canada, and Australia), a relatively flat US dollar denominated spot price for uranium could reflect the fundamental strength of the uranium market. While other commodities have declined significantly in both US dollar terms and foreign currency terms (like oil in particular), uranium has remained relatively flat in US dollar terms and has seen significant increases in foreign currency terms. In Canada, for example, the spot price of uranium in Canadian dollar terms increased by over 15% in 2015. By comparison, the price of oil in Canadian dollar terms (West Texas Intermediate) has decreased by over 17% in 2015. The rising price of uranium in foreign currency terms should encourage spot market sales, which should put downward pressure on prices. Despite this, we saw the spot price for uranium remain relatively flat in 2015.

**Competition**

The uranium industry is small compared to other commodity industries, 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 by four producers accounted for approximately 62% of world production in 2015. In total ten producers represent 88.3% of the world's production. The industry is also geographically concentrated with about 70% of the world's production coming from only three countries: Kazakhstan, Canada and Australia. Kazakhstan is the largest producer, with production of approximately 40% of the total primary production in 2015.

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 and in Africa. 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. In Africa, exploration activity has slowed in recent years as investment has been difficult to come by to fund the relatively low-grade and potentially high-cost operations that are expected to emerge from African uranium deposits.

This PEA study uses a base case long-term contract uranium price of US$44.00 per pound $U_3O_8$ (as of March 28, 2016 and a (CDN:US) exchange rate of 1.35 being consistent with the 10 to 25 year (project production period) forward estimate for the (CDN:US) exchange rate as per Bloomberg on February 1, 2016.

## 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

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.

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 (federal and provincial) and industry have continued to increase their attention to addressing social considerations associated with uranium mining in Saskatchewan.

## 20.1 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.

### 20.1.1 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. The MOE follows the following process to determine which level of assessment will be required.

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.

### 20.1.2 Federal Requirements

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
- 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 "trigging" 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), or 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

## 20.2 Licensing and Permitting

In the event 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.

## 20.3 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, as shown in Figure 16-31. 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 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.

## 20.4 Environmental Considerations

The main environmental considerations associated with this project are centred on the management of its various waste streams. The dominant and/or potentially more problematic of these waste streams are water management and waste rock. Given that this proposed project is currently relying on a toll milling arrangement to process mine production, which is typically a dominant environmental concern associated with high grade uranium mine projects, the management of the tailings and associated effluent will be the responsibility of the chosen toll mill. It is important to note that the owner of the toll mill chosen to process the Wheeler River plant feed will be required to complete an environmental assessment to demonstrate the addition of this material to their tailings management system will not result in unacceptable environmental impacts to their receiving environment.

Based on the existing understanding of the project and its location, there are no environmental fatal flaws identified with this proposed project.

### 20.4.1 Environmental Baseline Studies

In anticipation of advancing the project, Denison completed four studies that will contribute to the baseline information required to support an EA. Two studies completed in 2012 focused on providing baseline information addressing the hydrological and aquatic environments of the project area (Golder, 2013 and 2013a). The third study focused on providing a geotechnical and hydrogeological evaluation of a single diamond borehole completed in the winter of 2014. The fourth study completed in 2015 was a more extensive hydrogeological evaluation which focused on hydraulic testing in 12 diamond drill holes (SRK, 2015).

In the event Denison advances the engineering of this proposed project to a pre-feasibility level, then additional environmental baseline studies and environmental characterization programs should be initiated in parallel.

### 20.4.2 Water Management

Mine water is expected to have concentrations of metals, uranium, molybdenum, selenium, arsenic and radium that will require treatment to discharge the water. Concentrations of blasting residuals (nitrate and ammonium) may also be elevated and require treatment, but for the purposes of this assessment it is assumed good management practices will limit these constituents to concentrations that do not require treatment prior to discharge.

The planned water management infrastructure, including surge/holding ponds, treated water monitoring ponds, and water treatment plant are described in report Section 18.20.

### 20.4.3 Metal Leaching/Acid Rock Drainage (ML/ARD) Potential

To date, there has been no specific efforts to characterize the ML/ARD potential of the tailings or waste rock at either of the deposits. However, there has been extensive sampling and geochemical analyses of mineralization and waste zones in the drill core (Section 10.5.2) that provide a basis for understanding how these materials compare to tailings and waste rock from other deposits in the Athabasca Basin. Summaries of trace element concentrations found in the mineralization and waste rock are provided in Table 20-1 and Table 20-2. A brief discussion on the ML/ARD potential of the mineralization/tailings and waste rock follows.

#### Tailings

Assay results for mineralization (Table 20-1) are based on samples from the vicinity of the Phoenix and Gryphon deposits that had a uranium content of greater than 2000 ppm. Processing is expected to remove the uranium and some of the molybdenum, and will result in some dilution of the trace elements due to the addition of sulphuric acid and lime in the process. However, most of the trace elements are expected to report to the tailings. Concentrations of arsenic, cobalt, and nickel in the mineralization samples were at the low end of the range observed in tailings from other uranium deposits in the Athabasca Basin (confidential data available to the author). In contrast, concentrations of copper and zinc tended to be high in comparison to tailings from other deposits, and concentrations of molybdenum and selenium were at the high end of the range.

Assuming that the tailings are deposited in one of the existing in-pit tailings management facilities where they will remain submerged beneath the water table, acid rock drainage is not expected to be an issue. Additionally, elements that occur as cations in the water (e.g., cobalt, copper, nickel and zinc) are easily controlled by the addition of lime in the neutralization circuit of the process. Experience from other uranium tailings deposits in the region has shown that the most significant “metal” leaching issues are arsenic, radium 226, and uranium. Given the relatively low concentrations of arsenic in the mineralization, and the relatively high iron to arsenic molar ratios in the mineralization (>100 in both deposits) and the acidic raffinate (44 for Phoenix, and 52 for Gryphon), (SRC 2014, 2015) in comparison to tailings from other deposits, arsenic concentrations in the tailings porewater are expected to be at the low end of the range found in other sites in the area. Concentrations of radium-226 and uranium in tailings porewater are dependent on a number of other deposit and processing specific factors. Therefore, testing is typically required to determine the potential range of concentrations that could be present in the tailings supernatant, and tailings porewater over a longer term. Nonetheless, based on the characteristics of the mineralization, it is likely that radium-226 and uranium concentrations will be within the range that is currently managed within existing facilities in the region. Site specific testing is also generally required to estimate concentrations of molybdenum and selenium in the treated effluent, tailings supernatant and tailings porewater. Recommendations for further testing are provided in Section 26.

#### Waste Rock

The assay results for waste rock samples in the vicinity of each of the deposits and in the planned connection drift are provided in Table 20-2. With the exception of nickel, trace element concentrations in the basement rock samples were generally comparable to that of other basement rock hosted deposits such as Eagle Point and McArthur River. Cobalt concentrations in the basement rock were slightly higher than concentrations at other mono-minerallic deposits such as Eagle Point and McArthur River, but not as high as those in polymetallic deposits such as Key Lake and Zone B. Trace element concentrations in the sandstone were generally within the range found in other deposits in the area.

The ARD potential of the waste rock has not been characterized. Geological observations from drill core from the vicinity of the Phoenix and Gryphon deposits indicate that trace concentrations of pyrite occur in approximately 20% of the basement rock samples, and 5% of the sandstone samples, and that more significant concentrations (>1%) occur in approximately 1 to 2% of both sandstone and basement rock samples. These data suggest that there is potential for ARD in the waste rock, indicating that management and control of ARD should be considered in the project planning. However, further characterization is required to quantify the sulphides and neutralization potential of waste rock in the vicinity of the mine workings to quantify the amount of waste rock that will require special management and to determine whether any of the rock will be suitable for construction. Recommendations for further testing are provided in report Section 26.

The metal leaching potential of the waste rock will be largely determined by the pH condition of the waste rock, with higher concentrations expected under acidic conditions. Further laboratory and Summary of Trace Element Concentrations is discussed in report Section 26.7.

**Table 20-1: Summary of Trace Element Concentrations in the Mineralization**

| Zone | Statistic | As ppm | Cd ppm | Cu ppm | Co ppm | Mo ppm | Ni ppm | Pb ppm | Se ppm | Th ppm | U ppm | Zn ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Phoenix U > 2000 ppm** | average | 318 | 20 | 7257 | 193 | 488 | 575 | 9115 | 77 | 365 | 94860 | 2116 |
| | 25th percentile | 63 | 0.5 | 490 | 46 | 26 | 194 | 461 | 0.5 | 24 | 4740 | 45 |
| | median | 160 | 1 | 1520 | 104 | 99 | 336 | 1470 | 11 | 56 | 15400 | 212 |
| | 75th percentile | 366 | 3 | 5305 | 235 | 370 | 614 | 7060 | 61 | 154 | 78450 | 1100 |
| | number of samples | 1059 | 1059 | 1059 | 1059 | 1059 | 1059 | 1059 | 1059 | 1059 | 1059 | 1059 |
| **Gryphon U > 2000 ppm** | average | 26 | - | 337 | 64 | 1132 | 102 | 3197 | 32 | 58 | 31812 | 21 |
| | 25th percentile | 4 | - | 16 | 13 | 160 | 52 | 264 | 4 | 23 | 3410 | 0.5 |
| | median | 10 | - | 55 | 33 | 507 | 77 | 695 | 10 | 40 | 8310 | 1 |
| | 75th percentile | 28 | - | 208 | 68 | 1270 | 128 | 2620 | 30 | 66 | 27900 | 4 |
| | number of samples | 377 | - | 377 | 377 | 377 | 377 | 377 | 377 | 377 | 377 | 377 |

**Table 20-2: Summary of Trace Element Concentrations in the Waste Rock**

| Zone | Statistic | As ppm | Cd ppm | Cu ppm | Co ppm | Mo ppm | Ni ppm | Pb ppm | Se ppm | Th ppm | U ppm | Zn ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Phoenix Basement** | average | 4 | 0.2 | 62 | 16 | 9 | 50 | 12 | 0.49 | 22 | 37 | 24 |
| | 25th percentile | 0.31 | 0.1 | 3 | 6.5 | 0.3 | 22 | 3.1 | 0.1 | 14 | 3.1 | 7 |
| | median | 0.7 | 0.1 | 10 | 13 | 0.7 | 43 | 5.5 | 0.1 | 21 | 5.5 | 15 |
| | 75th percentile | 2.3 | 0.2 | 32 | 20 | 4.1 | 64 | 11 | 0.5 | 27 | 12 | 29 |
| | number of samples | 3975 | 3975 | 3975 | 3975 | 3975 | 3975 | 3975 | 3975 | 3975 | 3975 | 3975 |
| **Gryphon Basement** | average | 1.6 | 0.07 | 46 | 11 | 9 | 44 | 19 | 0.37 | 23 | 24 | 28 |
| | 25th percentile | 0.24 | 0.01 | 7 | 4 | 1 | 20 | 6.5 | 0.1 | 15 | 6.2 | 12 |
| | median | 0.48 | 0.02 | 17 | 10 | 3 | 45 | 11 | 0.1 | 21 | 8.4 | 19 |
| | 75th percentile | 1.1 | 0.1 | 40 | 16 | 6 | 61 | 24 | 0.4 | 27 | 15 | 35 |
| | number of samples | 3644 | 3644 | 3644 | 3644 | 3644 | 3644 | 3644 | 3644 | 3644 | 3644 | 3644 |
| **Connection drift Basement** | average | 0.88 | 0.19 | 19 | 11 | 1.5 | 31 | 8.9 | 0.58 | 29 | 5.2 | 25 |
| | 25th percentile | 0.22 | 0.1 | 1.6 | 3 | 0.2 | 9.4 | 3.2 | 0.1 | 14 | 2.4 | 8 |
| | median | 0.4 | 0.1 | 5.2 | 7.9 | 0.5 | 25 | 5.6 | 0.2 | 22 | 4 | 17 |
| | 75th percentile | 0.73 | 0.2 | 21 | 16 | 1.3 | 41 | 11 | 0.9 | 29 | 5.7 | 31 |
| | number of samples | 480 | 480 | 480 | 480 | 480 | 480 | 480 | 480 | 480 | 480 | 480 |
| **Phoenix Sandstone** | average | 0.82 | 0.12 | 5.6 | 0.93 | 0.36 | 8.6 | 4.9 | 0.12 | 16 | 4.3 | 3.4 |
| | 25th percentile | 0.3 | 0.1 | 1.4 | 0.35 | 0.12 | 3.8 | 2.5 | 0.1 | 6.1 | 1.3 | 1 |
| | median | 0.5 | 0.1 | 2.8 | 0.51 | 0.18 | 5.8 | 3.1 | 0.1 | 11 | 1.9 | 2 |
| | 75th percentile | 0.76 | 0.1 | 5.3 | 0.92 | 0.29 | 9.4 | 4.1 | 0.1 | 20 | 3.2 | 4 |
| | number of samples | 9379 | 9379 | 9379 | 9379 | 9379 | 9379 | 9379 | 9379 | 9379 | 9379 | 9379 |
| **Gryphon Sandstone** | average | 1 | 0.15 | 2.5 | 0.47 | 0.19 | 2.8 | 4.7 | 0.14 | 23 | 1.8 | 4.2 |
| | 25th percentile | 0.23 | 0.1 | 0.6 | 0.12 | 0.11 | 1 | 2.8 | 0.1 | 8 | 1 | 3 |
| | median | 0.38 | 0.1 | 1 | 0.22 | 0.16 | 1.4 | 3.6 | 0.1 | 15 | 1.3 | 4 |
| | 75th percentile | 0.63 | 0.2 | 2 | 0.39 | 0.22 | 2.4 | 4.9 | 0.1 | 29 | 1.9 | 4 |
| | number of samples | 3931 | 3931 | 3931 | 3931 | 3931 | 3931 | 3931 | 3931 | 3931 | 3931 | 3931 |

## 20.5 Social Considerations

Significant efforts have been expended by the Saskatchewan government and the uranium mining industry since the early 1990s to solicit and incorporate the traditional knowledge, concerns, and desires of northern Saskatchewan residents (both aboriginal and non-aboriginal) into the environmental assessment process. There are a number of well-established forums and committees in existence, in Saskatchewan, mandated to facilitate consultation between the proponents of proposed uranium developments and stakeholder groups. In addition, the CEAA 2012 EA process recommends the project description for any proposed project be vetted with the project's stakeholder groups prior to its submission to the responsible authority for screening.

The primary public and First Nations stakeholder groups the Wheeler River project will be required to interact with throughout all phases of the project life (environmental assessment, operations, closure and post closure) will be:

- English River First Nation
- Community of Pinehouse
- Community of Patuanak

In addition, as the project advances, the stakeholder group will undoubted be expanded to include additional Athabasca Basin communities and First Nations to address consultation requirements associated with milling the plant feed and subsequently managing the tailings and effluent at the McClean Lake project.

Previous assessments involving the above stakeholder groups have shown the fundamental areas of concern involve the development and implementation of robust environmental management plans throughout operations, coupled with a closure plan that ensures very low risk of long term environmental impacts.

From a socio-economic perspective, many if not all of these communities and political entities have interests in limited partnerships and other business ventures established to take advantage of the economic opportunities associated with northern Saskatchewan’s mining industry. These stakeholder groups would be looking for opportunities to enter into contractual arrangements to maximize the involvement of these businesses with the project in the event the project gains environmental assessment approvals to proceed.

# 21 Capital and Operating Costs

## 21.1 Basis of Cost Estimates

Capital and operating cost estimates for the Gryphon and Phoenix underground mines, supporting site infrastructure and McClean Lake mill modifications are described in this section. All capital and operating costs are expressed in 2015 Canadian dollars. Capital costs are estimated to a bottom line accuracy of +/- 40% with no escalation applied.

Capital and operating costs for the project have been estimated by:

- Amec Foster Wheeler, responsible for the areas of Phoenix underground mineral process infrastructure, Phoenix high grade slurry load out facility, and co-milling of Wheeler River mill feed at the McClean Lake mill
- SRK, responsible for all other cost estimates, assisted by Denison on certain items

### 21.1.1 Amec Foster Wheeler Cost Estimation Approach

In May 2015, Denison engaged Amec Foster Wheeler to undertake a cost study of toll milling uranium feed from the Phoenix and Gryphon zones on the Wheeler River property, at the McClean Lake mill (Amec, 2015). To establish capital and operating costs, Amec Foster Wheeler developed design criteria, characterized production constraints, and determined the scope of equipment modifications for each circuit. The Phoenix and Gryphon zones have limited metallurgical test data available, so conservative design criteria as established in report Sections 13 and 17 are used pending future test results.

An equipment-factored capital cost approach was taken by Amec Foster Wheeler, as typically used for an AACEI Class 5 estimate. The mechanical equipment costs are based primarily on in-house data. Other direct and indirect project costs are based on installed mechanical equipment cost, applying typical factors. No risk or escalation factors were applied. For underground areas, excavation costs are covered separately from process equipment in report Section 16.

### 21.1.2 SRK Cost Estimation Approach

To support cost estimation, SRK created 3D mine models for Gryphon and Phoenix and prepared a LOM schedule of key development and production activities including shaft and ventilation raise sinking, and underground lateral and vertical development. Ventilation modelling was done and annual equipment operating hours were estimated.

SRK cost estimates are based on:

- Comparisons to the Cigar Lake mine publically stated costs and JBS productivities
- Comparisons to other northern Saskatchewan uranium mines
- Shaft sinking contractor budgetary quotations
- Equipment and supplies budgetary quotations from suppliers, and on-line pricing information
- Technical and cost information on ground freezing from Newmans Geotechnique Inc.
- Factored costs from other recent feasibility level and other studies
- SRK’s in house cost database

## 21.2 Capital Costs

The Wheeler River project is at an advanced exploration stage. Additional exploration drilling, resource modelling, technical studies, and permitting activities are anticipated as part of the project's development. The PEA project schedule (Figure 16-31) includes estimated timelines for technical studies and permitting. The project could be fully permitted by the end of 2020.

Project development costs for technical and environmental studies, field work programs, regulatory applications have been incorporated into the project schedule between January 1, 2016 and December 31, 2020. An allowance of $25M has been allocated for these activities which are considered as excluded capital costs for the purpose of economic modelling.

Initial capital costs are based on the five-year period from January 1, 2021 through to December 31, 2025. Sustaining capital costs are for the period from January 1, 2026 through to end of 2045.

## 21.3 Capital Cost Summary

The Wheeler River project total capital cost estimate is estimated at $1,103 million, comprised of $560 million initial capital and $543 million sustaining capital (Table 21-1). Initial capital includes a 30% contingency while sustaining capital has a 23% contingency.

**Table 21-1: Wheeler River Project Capital Cost Summary**

| Capital Costs Area | Initial $M | Sustaining $M | Total $M | Direct $M | Indirect $M |
|---|---|---|---|---|---|
| Owners Costs | $25 | | **$25** | | $25 |
| Site Infrastructure | $122 | $5 | **$127** | $98 | $29 |
| Haul Road to McClean Lake Mill | $38 | | **$38** | $31 | $7 |
| Surface Mobile Equipment | $7 | $2 | **$9** | $9 | |
| Underground Mine Development | $19 | $144 | **$163** | $122 | $41 |
| Production Shaft and Ventilation Raises | $148 | $24 | **$173** | $155 | $18 |
| Underground Mining Equipment | $18 | $74 | **$92** | $86 | $6 |
| Underground Mining Infrastructure | $33 | $20 | **$53** | $43 | $10 |
| Phoenix Freeze Infrastructure | | $73 | **$73** | $63 | $10 |
| On Site Processing Facilities | | $58 | **$58** | $39 | $19 |
| McClean Mill Modifications | $19 | $2 | **$20** | $12 | $8 |
| Decommissioning | | $40 | **$40** | $32 | $8 |
| **Subtotal** | **$429** | **$442** | **$871** | **$690** | **$181** |
| Contingency | $131 | $101 | **$232** | $178 | $54 |
| **Total Capital ($M)** | **$560** | **$543** | **$1,103** | **$868** | **$235** |

## 21.4 Capital Cost Details

The following report sections provide details on the line items shown in Table 21-1. The capital costs shown below are before contingency.

### 21.4.1 Owners Costs

Owner's costs consist of owner's employee costs, insurance, regulatory and licensing fees, taxes and other similar costs incurred during construction.

### 21.4.2 Site Infrastructure

Site infrastructure capital cost details are shown in Table 21-2.

**Table 21-2: Site Infrastructure Capital Costs**

| Site Infrastructure | Cost $M | Direct $M | Indirect $M |
|---|---|---|---|
| Roads & Site Preparation | **$10** | $8.3 | $1.7 |
| Camp & Buildings | **$17** | $14 | $3.0 |
| Power Supply & Distribution | **$9.3** | $7.8 | $1.5 |
| Fuel Storage & Dispensing | **$0.6** | $0.5 | $0.1 |
| Main Vent Fans & Heaters | **$5.4** | $4.0 | $1.4 |
| Water Management Ponds | **$5.9** | $4.4 | $1.5 |
| Water Treatment Plant | **$73** | $54 | $19 |
| Backfill Plant | **$0.8** | $0.7 | $0.1 |
| Feed & Waste Rock Storage Pads | **$5.0** | $4.3 | $0.7 |
| **Total Capital ($M)** | **$127** | **$98** | **$29** |

Camp & Buildings includes:

- A 250 room camp and change house
- Administration office
- Maintenance shop, warehouse and cold storage
- Fuel storage
- Laboratories
- Emergency facilities building

The capital cost for the 1,500 $m^3$/hr capacity water treatment plant was estimated based on costs for similar operating water treatment facilities at uranium mines in northern Saskatchewan. A Lang factor for chemical plants processing primarily liquids was used to scale using the ratio of flow rates. Capital costs include major process equipment, process control, instrumentation, piping, buildings, associated geotechnical work for foundations and electrical engineering.

Further details on site infrastructure are included in report Section 18.

### 21.4.3 Haul Road to McClean Lake

A new 45 km section of plant feed haul road is needed between the McArthur River mine and the Cigar Lake mine. This will allow plant feed haul trucks to take the shortest route to the McClean Lake mill.

### 21.4.4 Surface Mobile Equipment

The major units included in surface equipment are:

- Grader, front end loaders (2), dump truck, track dozer, flatbed truck and backhoe
- Ambulance, fire truck and mine rescue truck
- Concrete ready mix trucks (2)
- Forklifts (2)
- Plant feed haulage trucks (8), including high grade slurry containers
- Crew vans (6)

### 21.4.5 Underground Mine Development

This capital item includes:

- Definition drilling at Gryphon

- Mine general (indirect) costs during the pre-production period
- 16 km of capitalized lateral development shown in (Table 16-7)

Lateral development unit rates per meter for owner capitalized development are estimated at $8,600 for Gryphon, $13,000 for the connection drift and $12,000 for Phoenix.

### 21.4.6 Production Shaft and Ventilation Raises

This capital item covers the excavation and concrete lining of the 5.50 m diameter, 583 m deep Gryphon production shaft, the 4.50 m diameter, 550 m deep Gryphon ventilation raise, and the 4.50 m diameter, 440 m deep Phoenix ventilation raise. Additional costs included cover the Gryphon headframe and hoist house, underground rock breaker and bins, loading pocket and shaft furnishings.

### 21.4.7 Underground Mining Equipment

This capital item includes:

- Jet boring units (2)
- Gryphon and Phoenix mobile mining fleets
- Replacement equipment

Selected units include 2-boom jumbos, mechanical bolters, 10-tonne and 6.7-tonne capacity load-haul-dump (LHDs), transmixer and shotcrete unit, 20-tonne capacity mine trucks, and 63mm (2.5 inch) diameter longhole drills.

### 21.4.8 Underground Mining Infrastructure

This capital item includes:

- Underground dewatering system with a design capacity of 2,250 $m^3$/hr with major sumps and pump stations at Gryphon and Phoenix
- Underground high pressure jet pump room
- Underground electrical distribution
- Other underground construction items, including:
  - Maintenance shops (2)
  - Fuel and lube storage facilities
  - Backfill slurry plant
  - Refuge stations and escape way to surface
  - Ventilation doors and regulators

### 21.4.9 Phoenix Freeze Infrastructure

This capital item includes five self-contained surface freeze plants, steel columns for brine transfer in the Phoenix ventilation raise, underground heat exchangers, underground brine circulation system and 24,000 m of freeze hole installation.

### 21.4.10 Phoenix On-Site Processing Facilities

This capital item, estimated by Amec Wheeler Foster, includes the facilities for underground processing Phoenix high grade mineralization and for slurry load out on surface at the Phoenix site:

- Underground ROM sump and JBS water recycle
- Underground crushing, grinding and thickening of jet bore production
- Underground slurry hoist pumps
- Surface slurry load out facility at Phoenix

### Phoenix Underground Processing Equipment

For underground processing of Phoenix deposit high grade slurry from JBS mining, the majority of equipment is assumed to be of similar design as that operating at Cigar Lake. Grinding and slurry hoisting aspects for Phoenix are also similar to McArthur River mine. Phoenix underground processing can be considered as three areas: JBS recirculation, grinding, and slurry hoisting.

With Phoenix production tonnage rate planned to be approximately half of Cigar Lake Phase 1, the JBS recirculation equipment can be reduced in size accordingly. The JBS recirculation area has one ROM sump, a recycle water tank, a process water tank, sand filters, and pumps to supply recycle water and medium pressure water services to the JBS.

Grinding and slurry hoisting are planned to be in operation part-time, as practiced at McArthur River and Cigar Lake mines. The equipment for Phoenix is similar in size to these operations, with a grinding production rate of at least 10 t/h. The Phoenix grinding area has a waterflush cone crusher, ball mill, hydrocyclones, and a thickener with flocculant system and associated pumps. The hoisting area has a thickener underflow pachuca, charge pumps and two hoist pumps (one operating, one standby) to surface. The underground process equipment cost is summarized in Table 21-3.

**Table 21-3: Phoenix Underground Process Equipment Capital Cost Estimate**

| Summary CAPEX | Factor | Cost $M |
|---|---|---|
| Total Direct Costs | 100% | $23.6 |
| Construction Indirects | 33% | $7.79 |
| EPCM Indirects | 22% | $5.20 |
| Owner Cost | 12% | $2.83 |
| Tax | | $0.36 |
| **Total Project Cost** | | **$39.8** |

### Phoenix Surface Slurry Loadout Capital Cost Estimate

The capital cost for the surface slurry load out facility at Phoenix is estimated at $18.6 M.

## 21.4.11 McClean Lake Mill Modifications

The JEB mill at McClean Lake has the majority of equipment already in place to process Phoenix high grade slurry and Gryphon coarse muck. There are a number of relatively minor debottlenecking modifications required in the slurry offloading, leaching and solid-liquid separation areas to co-mill Wheeler River feed sources with Cigar Lake ore.

An important consideration in leach circuit design is potential for hydrogen evolution, and how its risks are mitigated. This is of particular concern for the JEB mill due to the existing tanks being of closed top, pressurized design. It is recommended that hydrogen evolution from the Gryphon and Phoenix deposits be measured in future test work. If high rates of hydrogen evolution are observed, then its management may require alteration of the leach circuit design chosen for this study.

For process capital cost estimation, the Gryphon zone/Cigar Lake Phase 1 co-milling and Phoenix zone/Cigar Lake Phase 2 co-milling scenarios are shown separately, as each has unique capital and operating costs.

An equipment-factored capital cost approach was taken, as typically used for an AACEI Class 5 estimate. The mechanical equipment costs are based on in-house data as well as budgetary vendor quotes. Placeholder allowances were included for un-scoped brownfield demolition of existing equipment to make way for new installation. Other direct and indirect project costs are based on installed mechanical equipment cost, applying typical factors. No risk or escalation factors were applied.

## Process Capital Cost Estimate – Gryphon

### *Leaching Scope*

One additional leach tank of 76 $m^3$ volume (or two tanks of 38 $m^3$ each) is required to give nine hours residence time in the #1 leach circuit. A likely location is the old tailings neutralization tanks area adjacent to the leaching circuit, which will be out of service. To mitigate risks from potential hydrogen generation, the leach tank design chosen is open to atmospheric pressure such that ventilation rate is unconstrained by the tank(s). It is recommended that the new tank(s) be configured at the front end of the #1 leach circuit tank series, where maximum hydrogen generation is likely to occur.

A ferric sulphate reagent dosing pump (one operating, one standby) and a dedicated line to the front end of #1 leach circuit is required.

### *Solid/Liquid Separation Scope*

As indicated in Section 17, the CCD circuit is the most important bottleneck, and it is improbable that CCC would add capacity. A conventional approach to wash poorly settling solids is pressure filtration. Pressure filters are an efficient and compact option as opposed to expanding the CCD circuit.

For the base case to reach full Cigar Lake Phase 1/Gryphon co-milling capacity within the recovery rate criteria, two sets of new pressure filters are proposed to supplement CCD:

- Cigar Lake leach residue slurry from the primary thickener underflow feeds to a new dedicated pressure filter. The washed cake is sent directly to tailings neutralization.
- Gryphon slurry is split into coarse and fine fractions using an existing CCC hydrocyclone, and then:
  - The coarse fraction is sent to the existing CCD instead of CCC wash cyclones. This way, CCD tonnage is reduced and settling performance is improved at the same time.
  - The fines fraction is sent to a new pressure filter instead of CCC. The washed cake is sent directly to tailings neutralization.

### *Clarification and Downstream Circuits Scope*

No capital cost has been allocated downstream of CCD under the assumption that all these circuits will be capable of 24 M lb $U_3O_8$/yr.

The scope of modifications for Gryphon milling are depicted in Figure 21-1 below.

The capital cost buildup for the new equipment is shown in Table 21-4.

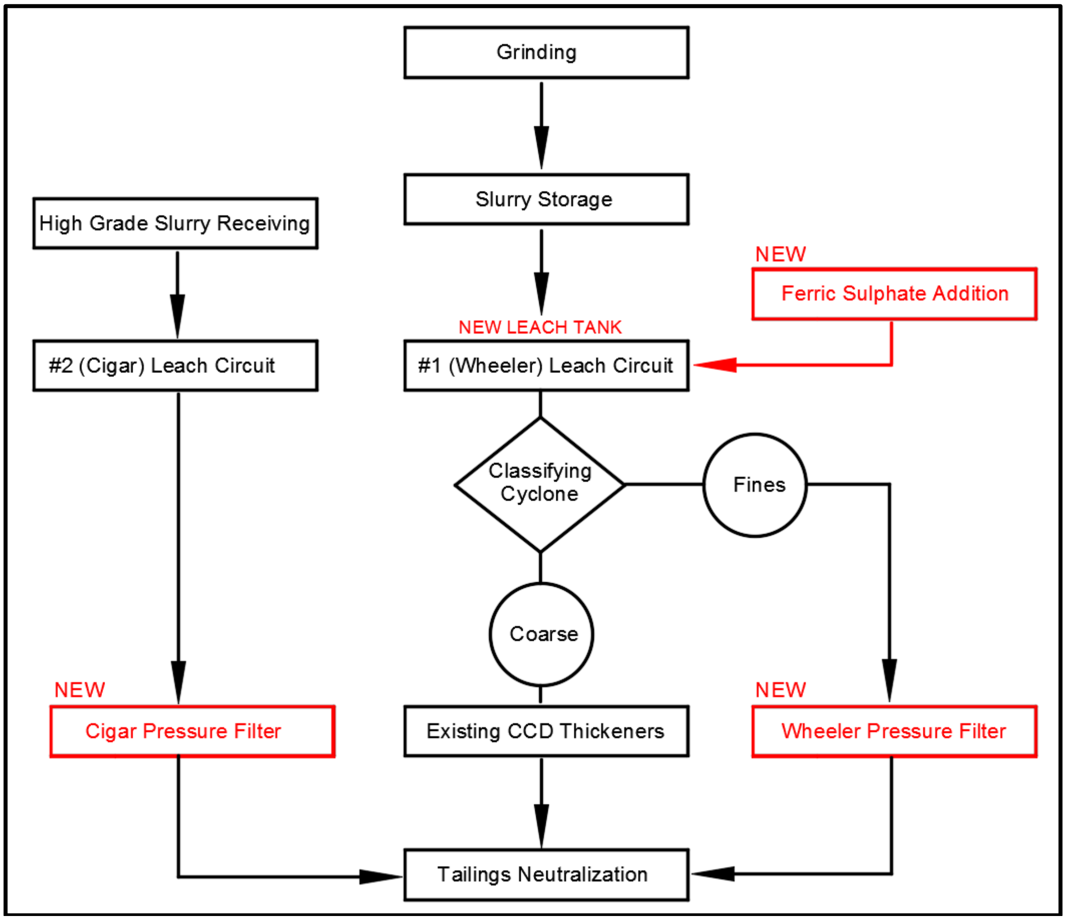



**Figure 21-1: Scope of Modifications for Co-Milling Cigar Phase 1 and Gryphon Deposit**
Source: Amec Foster Wheeler, 2016

**Table 21-4: Complete Project Cost Estimate for Gryphon Zone Mill Modifications**

| Summary CAPEX | Factor | Cost $M |
|---|---|---|
| Total Direct Costs | 100% | $10.98 |
| Construction Indirects | 33% | $3.62 |
| EPCM Indirects | 22% | $2.42 |
| Owner cost | 12% | $1.32 |
| Tax | | $0.18 |
| **Total Project Cost** | | **$18.51** |

## Process Capital Cost Estimate – Phoenix

### *Scope of Modifications*

In slurry receiving and storage, a new set of dedicated pumps and pipes to transfer ore slurry from the Phoenix high grade slurry pachucas to #1 circuit leach feed pachucas is likely required.

In leaching, the #1 leach circuit is expected to have a greatly decreased feed rate for Phoenix compared to Gryphon. The pumps will likely require lower capacity impellers and operate at lower speed, likely by changing out sheaves. New smaller diameter piping is anticipated to maintain flow velocity. With the high grade of feed, the exothermic reactions that occur due to acid addition and uranium dissolution may produce excessive heat. This means that some form of cooling to control the leach operating temperature may be required. This could be mitigated by diluting and cooling the concentrated sulphuric acid prior to leach tank addition. Due to the large excess residence time capacity of the circuit at the low Phoenix feed tonnage, diluting the slurry to run at a much lower density than typical 40-50% solids is also an option.

In solid/liquid separation, some flow re-routing, pump capacity adjustments and use of smaller diameter leach discharge piping is anticipated.

No capital cost has been allocated downstream of CCD under the assumption that all these circuits will be capable of operating at 24 M lb $U_3O_8$/yr.

The scope of modifications for Phoenix milling are depicted in Figure 21-2.

The capital cost buildup for the new pumping and piping configuration are shown in Table 21-5.

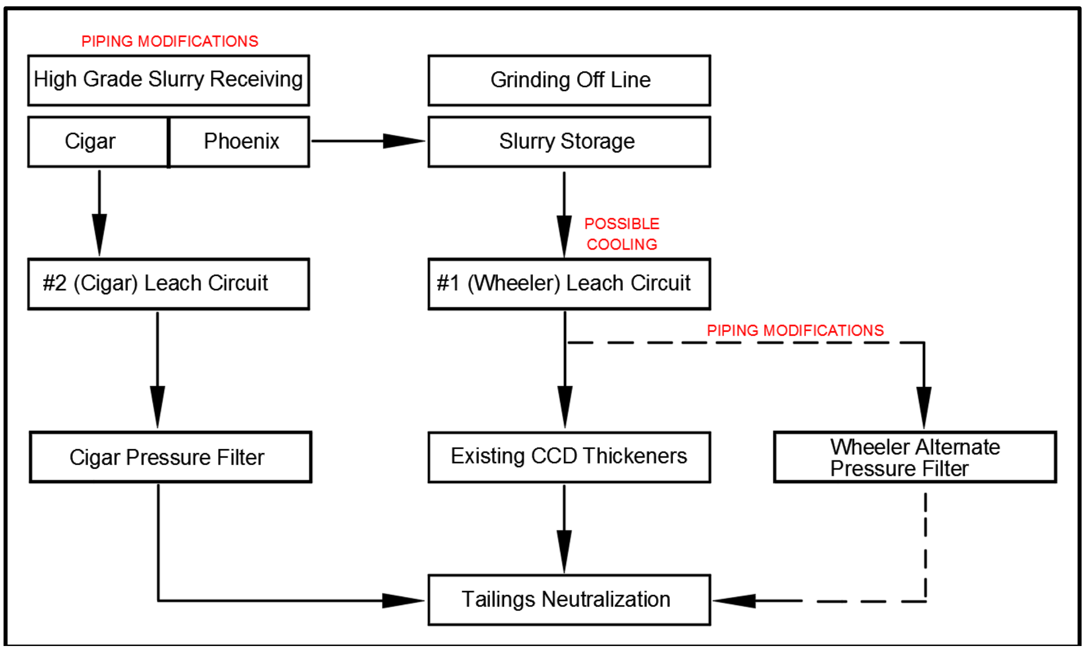



**Figure 21-2: Scope of Modifications for Co-Milling Cigar Lake and Phoenix Deposit**
Source: Amec Foster Wheeler, 2016

**Table 21-5: Complete Project Cost Estimate for Phoenix Deposit Mill Modifications**

| **Summary CAPEX** | **Factor** | **Cost $M** |
|---|---|---|
| Total Direct Costs | 100% | $1.11 |
| Construction Indirects | 33% | $0.37 |
| EPCM Indirects | 22% | $0.24 |
| Owner cost | 12% | $0.13 |
| Tax | | $0.01 |
| **Total Project Cost** | | **$1.86** |

### 21.4.12 Decommissioning

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.

Decommissioning will largely occur at the end of the project life over several years. Table 21-6 identifies Saskatchewan uranium properties and the current financial assurance packages for decommissioning.

**Table 21-6: Saskatchewan Uranium Decommissioning Financial Assurance Packages**

| Facility | CDN Dollar Amount | Notes |
|---|---|---|
| Cigar Lake Project | ~$49M | Includes u/g mine, 2 shafts and large site footprint |
| McArthur River Operation | ~$48M | Includes u/g mine, 3 shafts and large site footprint |
| Rabbit Lake Operation | ~$202M | Includes mill, TMF and historic open pit operations |
| Key Lake Operation | ~$225M | Includes mill, TMF and historic open pit operations |
| McClean Lake Operation | ~$107M | Includes mill, TMF and historic open pit operations |

These comparisons provide a benchmark for potential decommissioning costs for Wheeler River. The development of Wheeler River will be carried out using modern technology, minimizing surface disturbance and incorporate eventual reclamation into the initial design phases. As such, a cost estimate of $40M has been included in the cash flow model for decommissioning costs at the Wheeler River project.

## 21.5 Capital Cost Expenditure Schedule

Table 21-7 shows the capital cost expenditure schedule which is an input to the cash flow model.

Table 21-7: Wheeler River Project Capital Cost Schedule

| Wheeler River Project Capital Cost Schedule | $ Million | Pre-production 2021 | 2022 | 2023 | 2024 | 2025 | Production 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 | 2037 | 2038 | 2039 | 2040 | 2041 | 2042 to 2045 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Surface** | | | | | | | | | | | | | | | | | | | | | | | |
| Owners Costs | $25.0 | $2.0 | $2.0 | $8.0 | $8.0 | $5.0 | | | | | | | | | | | | | | | | | |
| Roads & Site Preparation | $10.0 | $5.0 | | $2.5 | $2.5 | | | | | | | | | | | | | | | | | | |
| Camp/Buildings | $16.6 | $1.00 | $2.0 | $5.0 | $5.6 | $3.0 | | | | | | | | | | | | | | | | | |
| Power Supply/Distribution | $9.3 | $3.1 | $1.9 | $1.0 | $1.8 | | | | | | $1.5 | | | | | | | | | | | | |
| Fuel Storage & Dispensing | $0.6 | $0.3 | | $0.3 | | | | | | | | | | | | | | | | | | | |
| Main Vent Fans & Heaters | $5.4 | | | | $2.9 | | | | | $2.5 | | | | | | | | | | | | | |
| Water Management Ponds | $5.9 | $2.0 | | $3.9 | | | | | | | | | | | | | | | | | | | |
| Water Treatment Plant | $73.1 | | | $73.1 | | | | | | | | | | | | | | | | | | | |
| Surface Mobile Equipment | $9.4 | | $2.84 | $2.84 | | $1.39 | | | | | | $2.3 | | | | | | | | | | | |
| Haulage Road to McLean Lake | $37.5 | | | | $25.0 | $12.5 | | | | | | | | | | | | | | | | | |
| Backfill Plant | $0.8 | | | | | | | | | | | $0.4 | $0.4 | | | | | | | | | | |
| Feed and Waste Storage Pads (lined) | $5.0 | | | $2.0 | $2.0 | $1.0 | | | | | | | | | | | | | | | | | |
| Phoenix Slurry Loadout | $18.6 | | | | | | | | | | | $13.6 | $5.0 | | | | | | | | | | |
| Mill Upgrade Gryphon | $18.5 | | | | $9.26 | $9.26 | | | | | | | | | | | | | | | | | |
| Mill Upgrade Phoenix | $1.9 | | | | | | | | | | | | $1.9 | | | | | | | | | | |
| **Underground** | | | | | | | | | | | | | | | | | | | | | | | |
| Gryphon Definition Drilling | $2.7 | | | | | $0.9 | $0.89 | $0.9 | | | | | | | | | | | | | | | |
| Gryphon Production Shaft | $119 | $34.0 | $34.0 | $34.0 | $17.0 | | | | | | | | | | | | | | | | | | |
| Gryphon Ventilation Raise | $28.8 | | | $19.2 | $9.6 | | | | | | | | | | | | | | | | | | |
| Emergency Man Hoist | $1.9 | | | | | $1.7 | | | | | $0.3 | | | | | | | | | | | | |
| Phoenix Tent Freeze Infrastructure | $65.3 | | | | | | | | | | $32.7 | | | | $16.3 | | | $16.3 | | | | | |
| Phoenix Vent Raise | $23.6 | | | | | | | | $7.9 | $15.7 | | | | | | | | | | | | | |
| Mine General | $13.0 | | | | $1.5 | $1.5 | $1.5 | $1.5 | $1.0 | | $3.0 | $3.0 | | | | | | | | | | | |
| Connection Drift Gryphon to Phoenix | $42.7 | | | | | | $6.1 | $12.2 | $12.2 | $12.2 | | | | | | | | | | | | | |
| Lateral/Raise Dev't (Gryphon) | $41.3 | | | | $5.2 | $10.3 | $10.3 | $10.3 | $5.2 | | | | | | | | | | | | | | |
| Lateral/Raise Dev't (Phoenix) | $63.2 | | | | | | | | | | $15.8 | $15.8 | | $15.8 | | $15.8 | | | | | | | |
| UG Dewatering System | $32.6 | | | | $13.1 | $13.1 | | | | | | $6.5 | | | | | | | | | | | |
| Underground Electrical Distribution | $7.0 | | | | $0.7 | $0.7 | $0.7 | $0.7 | $0.7 | $0.7 | $0.7 | $0.7 | $0.7 | $0.7 | | | | | | | | | |
| Underground Construction | $13.1 | | | | $2.6 | $2.6 | $2.6 | | | | | $1.7 | $1.7 | $1.7 | | | | | | | | | |
| Mobile Equipment Initial | $17.7 | | | | $8.9 | $8.9 | | | | | | | | | | | | | | | | | |
| Replacement Equipment Phoenix | $14.1 | | | | | | | | | | $7.0 | $7.0 | | | | | | | | | | | |
| Jet Boring Units (2) | $60.0 | | | | | | | | | | | | $30.0 | $30.0 | | | | | | | | | |
| Crushing, Grinding, Slurry Pumping | $39.8 | | | | | | | | | | | $19.9 | $19.9 | | | | | | | | | | |
| General Sustaining | $7.5 | | | | | | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | $0.5 | | |
| Decommissioning | $40.0 | | | | | | | | | | | | | | | | | | | | $10.0 | $8.0 | $22.0 |
| **Sub-total Project Capital** | **$871** | **$47.4** | **$42.8** | **$152** | **$116** | **$71.8** | **$22.6** | **$26.1** | **$27.4** | **$31.6** | **$61.5** | **$71.6** | **$60.1** | **$48.8** | **$16.8** | **$16.3** | **$0.5** | **$16.8** | **$0.5** | **$0.5** | **$10.5** | **$8.0** | **$22.0** |
| Contingency | $232 | $17.0 | $15.6 | $48.5 | $31.6 | $18.0 | $7.0 | $7.8 | $8.9 | $10.9 | $13.5 | $19.2 | $12.4 | $11.7 | $2.5 | $4.7 | $0.0 | $2.5 | $0.0 | $0.0 | $0.0 | $0.0 | $0.0 |
| **Total Project Capital** | **$1,103** | **$64.5** | **$58.4** | **$200** | **$147** | **$89.8** | **$29.6** | **$33.9** | **$36.3** | **$42.6** | **$75.0** | **$90.8** | **$72.5** | **$60.4** | **$19.3** | **$21.0** | **$0.5** | **$19.3** | **$0.5** | **$0.5** | **$10.5** | **$8.0** | **$22.0** |

## 21.6 Operating Costs

Operating costs are estimated for the 16-year production period from January 1, 2026 through to end of 2041. Gryphon production is scheduled for the first seven years, followed by Phoenix production for nine years. The underground mine operating cost estimates are quite different for the two deposits due to the different mining methods selected.

## 21.7 Operating Cost Summary

Table 21-8 presents the Wheeler River project scoping level operating cost estimate.

**Table 21-8: Wheeler River Project Operating Cost Estimate**

| Operating Costs | $/lb $U_3O_8$ | |
|---|---|---|
| **Area** | **Gryphon** | **Phoenix** |
| Mining | $3.45 | $17.45 |
| Surface Transportation | $1.63 | $0.85 |
| Processing | $8.03 | $6.03 |
| Toll Milling Fee | $2.00 | $2.00 |
| General & Administration | $4.17 | $3.57 |
| **Total** | **$19.28** | **$29.90** |

## 21.8 Operating Cost Details

### 21.8.1 Mine Operating Cost Estimate

Conventional longhole mining planned at Gryphon is much less expensive than the specialized jet bore mining planned for Phoenix, on the basis of tonnes or pounds $U_3O_8$ mined.

A mining cost of $144 per tonne was estimated for Gryphon based on a comparison to a similar project. The reference project involved longhole mining of steeply dipping veins of similar thickness at a production rate of 390 t/d. The reference project was based on decline access and truck haulage. An extra 30% was added to the reference mine cost estimate to cover extra costs expected for uranium mining, yielding an equivalent operating cost of $3.45 per pound $U_3O_8$.

A mining cost of $17.45 per pound $U_3O_8$ was estimated for the Phoenix JBS mining based on a comparison to the publicly stated December 31, 2011 mining cost of $10.21 per pound $U_3O_8$ for the Cigar Lake mine. The main adjustment factor applied was due to Phoenix having a lower mining grade than Cigar Lake. Additional adjustment factors considered inflation and economy of scale.

### 21.8.2 Plant Feed Transportation Operating Cost Estimate

Surface transportation costs are based on hauling 160 km (one-way distance) from the Wheeler River site to the McClean Lake mill. Both low grade broken uranium mineralization in covered trucks and high grade slurry in special containers (by truck) will be transported. The estimate is based on:

- Cost information provided by Denison on northern Saskatchewan truck haulage costs for uranium plant feed
- Estimated truck cycle times
- A cost reference guide on truck transportation costs

### 21.8.3 Process Operating Cost Estimates

To establish process operating costs, Amec Foster Wheeler prepared an in-house cost model for each of the Gryphon and Phoenix deposits. The operating cost model includes all on-site operating costs, including sustaining capital, and off-site burdens including general administrative (G&A), and head office costs. Operating costs shared between the two mill feeds (employee costs and building heating for example) are split according to the respective share of uranium production. Operating costs specific to each mill feed (grinding ball consumption and leach acid consumption for example) are calculated separately for each mill feed.

In addition to the toll milling charges for Wheeler River's share of McClean Lake's expenses presented in this section, a toll milling fee similar to that set out in Cigar Lake's JEB Toll Milling Agreement may be applicable. The cash flow model includes an additional $2.00/lb provisional toll milling fee. Potential toll milling fees and revenues are excluded from the operating cost estimates below, and are discussed in Section 22.4, Toll Milling Revenue.

#### Gryphon Operating Cost Estimate

The operating cost estimate for Gryphon zone co-milling of \$8.03/lb $U_3O_8$ is shown in Table 21-9.

**Table 21-9: Gryphon Milling Operating Cost Estimate**

| Operating Cost Area | $000 per year | \$/lb $U_3O_8$ |
|---|---|---|
| Employee Costs | $5,539 | $0.92 |
| Milling Reagents and Consumables | $14,236 | $2.37 |
| Waste/Effluent Treatment and Tailings Management | $7,110 | $1.19 |
| Ammonium Sulphate Sales | -$41 | -$0.01 |
| Electric Power, Maintenance and Building Services | $7,863 | $1.31 |
| Site Administration | $7,349 | $1.22 |
| Sustaining Capital | $1,936 | $0.32 |
| G&A, Head Office | $4,161 | $0.69 |
| **Total** | **$48,153** | **$8.03** |

#### Phoenix Operating Cost Estimate

The operating cost estimate for Phoenix zone co-milling of \$6.03/lb $U_3O_8$ is shown in Table 21-10.

**Table 21-10: Phoenix Milling Operating Cost Estimate**

| Operating Cost Area | $000 per year | $/lb $U_3O_8$ |
|---|---|---|
| Employee Costs | $8,714 | $1.24 |
| Milling Reagents and Consumables | $7,384 | $1.05 |
| Waste/Effluent Treatment and Tailings Management | $1,624 | $0.23 |
| Ammonium Sulphate Sales | -$48 | -$0.01 |
| Electric Power, Maintenance and Building Services | $6,650 | $0.95 |
| Site Administration | $8,573 | $1.22 |
| Sustaining Capital | $3,045 | $0.44 |
| G&A, Head Office | $6,269 | $0.90 |
| **Total** | **$42,211** | **$6.03** |

### 21.8.4 G & A Operating Cost Estimate

An average general and administration (G&A) operating cost of $4.06/lb $U_3O_8$ is based on an estimated annual fixed cost of $25 million to operate a remote mine site in northern Saskatchewan. The G&A costs include flights, accommodations and catering, administration, general management, procurement, insurance, communications, etc. Costs are based on an average of actual costs for two northern mine sites in Saskatchewan.

# 22 Economic Analysis

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 (see section 22.5) 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 facts and circumstances; and
- A post-tax discounted cash flow model, specific to Denison (see section 22.6), which shows the economics of the project on a 60% basis (Denison's current interest in the Wheeler 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 apply to Denison.

## 22.1 Input and Assumptions

Inputs to both the pre-tax and post-tax cash flow models include:

- A five-year pre-production period from January 2021 to December 2025;
- A project production period of 16 years from January 2026 to the end of 2041;
- LOM production of 1.22 Mt of plant feed at an average grade of 3.91% $U_3O_8$ containing 105 Mlbs of $U_3O_{8;}$
- A production rate of 6.0 Mlbs $U_3O_8$ per year from Gryphon transitioning to Phoenix production at a rate of 7.0 Mlbs $U_3O_8$ per year;
- Estimated metallurgical process uranium recoveries of 97.0% and 98.1% for Gryphon and Phoenix plant feeds, respectively;
- A Base case uranium price of US$44.00 per pound translated to CAD using an exchange rate of 1.35 CAD/USD;
- Project operating costs as shown in Table 21-7. Included in operating costs is an allowance for a toll-milling processing fee of $2.00/lb $U_3O_{8.}$ Processing is assumed to occur at the McClean Lake mill which significantly reduces the capital needed for the project (as a new stand-alone processing facility is not required);
- Saskatchewan revenue-based royalties and surcharges applicable to uranium production, as follows: a) a basic royalty of 5.0% of gross uranium revenue; b) a resource credit of 0.75% of gross uranium revenue (which partially offsets the basic royalty); and c) a resource surcharge of 3.0% of the value of uranium sales;
- Wheeler River project total capital cost estimates of $1,103 million, comprised of $560 million initial capital and $543 million sustaining capital as shown in Table 21-3; and
- Operating costs as shown in Table 21-8

No inflation or escalation of revenue or costs has been incorporated.

## 22.2 Canadian Royalties Applicable to the Project

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"). Significant revisions to the uranium royalty regime in Saskatchewan became effective in 2013. The new royalty system is effective retroactive to January 1, 2013 and has three components:

i. Basic Royalty: Computed as 5% of gross revenues derived from uranium extracted from ore bodies in the province;
ii. Saskatchewan 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 new 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.

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.

Gross revenue, for the Basic Royalty, is determined in accordance with the Regulations and allows for reductions based on specified allowances. Net profit, for the Profit Royalty, is calculated based on the recognition of the full dollar value of a royalty payer's exploration, capital, production, decommissioning and reclamation costs, in most cases, incurred after January 1, 2013. Net profits will be 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.

## 22.3 Canadian Income and Other Taxes Applicable to the Project

In 2015 the taxable income of Canadian resource companies was subject to federal taxes at a rate of 15%, and provincial taxes in Saskatchewan, and other Provinces and Territories at rates varying between 10% and 16%. Taxable income for each entity is allocated between provinces and territories based on a two point average of the proportion of salaries and revenues attributable to each province or territory. Resource corporations in Saskatchewan are also subject to a resource surcharge equal to 3% of the value of resource sales from production in Saskatchewan, if any, during the year.

## 22.4 McClean Lake Toll Milling Revenue

Denison's wholly owned subsidiary, Denison Mines Inc., holds a 22.5% interest in the McClean Lake joint venture project (MLJV). The MLJV is a joint venture between AREVA Resources Canada Inc. (70%), Denison Mines Corp. (22.5%), and OURD (Canada) Ltd. (7.5%). AREVA is the joint venture operator.

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.

## 22.5 Pre-Tax Economic Analysis

### 22.5.1 Pre-Tax Cash Flow Model

#### Basis of the Model

The pre-tax 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% 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
- Discounting at 8% has been selected for NPV calculations (refer to following section)

Table 22-1 shows the Wheeler River project pre-tax cash flow model.

Note: For presentation purposes decommissioning costs (as per capital schedule) have not all been shown in the table (beyond year 2041), but are included in the financial results.

#### Basis of Discount Rate

A discount rate of 8% was selected for assessing the time value of money in project economics. While the standard industry discount is 10%, a lower rate was selected based on the following rationale:

- Current interest/lending rates are at all time historic lows
- Project country risks (political, taxation changes, corruption, civil unrest) are considered low in Canada and in Saskatchewan
- Minimal unknown risks for operating in northern Saskatchewan due to the significant existing regional infrastructure and current mining/milling operations
- Well established and proven mill operation with limited capital modifications required for the Wheeler River project
- Project production will be sourced first from the conventional mining methods planned for the Gryphon deposit while the more challenging Phoenix deposit production is phased in after Gryphon mining
- The project is relatively small with a short capital pre-production period and payback period, reducing inflation exposure
- Project specific risks (geotechnical, hydrogeological) have been assessed through specific risk, schedule and contingency analysis and do not impact discount rate

## Table 22-1: Wheeler River Project Pre-tax Cash Flow Model

| Wheeler River Pre-Tax Economic Model | | | | Pre-production | | | | | Production | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Units | Scheduled | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 | 2037 | 2038 | 2039 | 2040 | 2041 |
| **MINE PRODUCTION** | | | | | | | | | | | | | | | | | | | | | | | | |
| | Gryphon | Tonnes | 974,758 | | | | | 61,797 | 143,563 | 143,563 | 143,563 | 143,563 | 143,563 | 143,563 | 51,582 | | | | | | | | | |
| | | % U3O8 | | | | | | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | | | | | | | | | |
| | | Mlbs U3O8 | 40.739 | | | | | 2.58 | 6.00 | 6.00 | 6.00 | 6.00 | 6.00 | 6.00 | 2.16 | | | | | | | | | |
| | Phoenix | Tonnes | 239,852 | | | | | | | | | | | | 16,266 | 26,210 | 26,210 | 26,210 | 26,210 | 26,210 | 26,210 | 26,210 | 26,210 | 13,904 |
| | | % U3O8 | | | | | | | | | | | | | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 |
| | | Mlbs U3O8 | 64.06 | | | | | | | | | | | | 4.34 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 7.00 | 3.71 |
| | **Plant Feed** | Tonnes | 1,214,610 | | | | | 61,797 | 143,563 | 143,563 | 143,563 | 143,563 | 143,563 | 143,563 | 67,849 | 26,210 | 26,210 | 26,210 | 26,210 | 26,210 | 26,210 | 26,210 | 26,210 | 13,904 |
| 100% | | % U3O8 | 3.91 | | | | | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 1.90 | 4.35 | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 | 12.11 |
| | | Mlbs U3O8 | **105** | | | | | 2.6 | 6.0 | 6.0 | 6.0 | 6.0 | 6.0 | 6.0 | 6.5 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 7.0 | 3.7 |
| **GROSS REVENUE** | | | | | | | | | | | | | | | | | | | | | | | | |
| | Process Recovery | | | | | | | 97.0 | 97.0 | 97.0 | 97.0 | 97.0 | 97.0 | 97.0 | 97.0 | | | | | | | | | |
| | Gryphon | % | 97.0 | | | | | | | | | | | | | | | | | | | | | |
| | Phoenix | % | 98.1 | | | | | | | | | | | | 98.1 | 98.1 | 98.1 | 98.1 | 98.1 | 98.1 | 98.1 | 98.1 | 98.1 | 98.1 |
| | Recovered U3O8 | Mlbs U3O8 | **102.4** | | | | | 2.51 | 5.82 | 5.820 | 5.82 | 5.82 | 5.82 | 5.82 | 6.35 | 6.87 | 6.87 | 6.87 | 6.87 | 6.87 | 6.87 | 6.87 | 6.87 | 3.64 |
| | Price U3O8 | USD/lb | $44.00 | | | | | | | | | | | | | | | | | | | | | |
| | | exchange | 1.35 | | | | | | | | | | | | | | | | | | | | | |
| 100% | | CAD/lb | **$59.40** | | | | | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 | 59.40 |
| | Gross Revenue | $M | **$6,080** | | | | | 149 | 346 | 346 | 346 | 346 | 346 | 346 | 377 | 408 | 408 | 408 | 408 | 408 | 408 | 408 | 408 | 216 |
| **NET REVENUE** | | | | | | | | | | | | | | | | | | | | | | | | |
| | Royalties | | | | | | | | | | | | | | | | | | | | | | | |
| 3.00% | Resource Surcharge | $M | $182 | | | | | 4.5 | 10.4 | 10.4 | 10.4 | 10.4 | 10.4 | 10.4 | 11.3 | 12.2 | 12.2 | 12.2 | 12.2 | 12.2 | 12.2 | 12.2 | 12.2 | 6.5 |
| 5.00% | Basic Royalty | $M | $304 | | | | | 7.4 | 17.3 | 17.3 | 17.3 | 17.3 | 17.3 | 17.3 | 18.9 | 20.4 | 20.4 | 20.4 | 20.4 | 20.4 | 20.4 | 20.4 | 20.4 | 10.8 |
| -0.75% | Sask. Resource Credit | $M | -$46 | | | | | -1.1 | -2.6 | -2.6 | -2.6 | -2.6 | -2.6 | -2.6 | -2.8 | -3.1 | -3.1 | -3.1 | -3.1 | -3.1 | -3.1 | -3.1 | -3.1 | -1.6 |
| 7.25% | Total Royalies | $M | $441 | | | | | 10.8 | 25.1 | 25.1 | 25.1 | 25.1 | 25.1 | 25.1 | 27.4 | 29.6 | 29.6 | 29.6 | 29.6 | 29.6 | 29.6 | 29.6 | 29.6 | 15.7 |
| | Net Revenue | $M | **$5,639** | | | | | 138 | 321 | 321 | 321 | 321 | 321 | 321 | 350 | 378 | 378 | 378 | 378 | 378 | 378 | 378 | 378 | 201 |
| **OPERATING COSTS** | | | | | | | | | | | | | | | | | | | | | | | | |
| | | Gryphon | Phoenix | | | | | | | | | | | | | | | | | | | | | |
| | Mine | $3.45 | $17.45 | | | | | $3.45 | $3.45 | $3.45 | $3.45 | $3.45 | $3.45 | $3.45 | $6.81 | $17.45 | $17.45 | $17.45 | $17.45 | $17.45 | $17.45 | $17.45 | $17.45 | $17.45 |
| | Surface transport | $1.63 | $0.85 | | | | | $1.63 | $1.63 | $1.63 | $1.63 | $1.63 | $1.63 | $1.63 | $1.45 | $0.85 | $0.85 | $0.85 | $0.85 | $0.85 | $0.85 | $0.85 | $0.85 | $0.85 |
| | Process + Toll Charge | $10.03 | $8.03 | | | | | $10.03 | $10.03 | $10.03 | $10.03 | $10.03 | $10.03 | $10.03 | $8.69 | $8.03 | $8.03 | $8.03 | $8.03 | $8.03 | $8.03 | $8.03 | $8.03 | $8.03 |
| | G&A | $4.17 | $3.57 | | | | | $9.68 | $4.17 | $4.17 | $4.17 | $4.17 | $4.17 | $4.17 | $3.85 | $3.57 | $3.57 | $3.57 | $3.57 | $3.57 | $3.57 | $3.57 | $3.57 | $6.73 |
| Total Unit Cost ($/lb) | | **$19.28** | **$29.90** | | | | | $24.79 | $19.28 | $19.28 | $19.28 | $19.28 | $19.28 | $19.28 | $20.79 | $29.90 | $29.90 | $29.90 | $29.90 | $29.90 | $29.90 | $29.90 | $29.90 | $33.06 |
| | | | | | | | | | | | | | | | | | | | | | | | | |
| 100% | Operating Cost | $M | **$2,690** | | | | | 64 | 116 | 116 | 116 | 116 | 116 | 116 | 135 | 209 | 209 | 209 | 209 | 209 | 209 | 209 | 209 | 123 |
| **OPERATING CASH FLOW** | | | | | | | | | | | | | | | | | | | | | | | | |
| | Operating Cash Flow | $M | **$2,949** | | | | | 74 | 205 | 205 | 205 | 205 | 205 | 205 | 215 | 169 | 169 | 169 | 169 | 169 | 169 | 169 | 169 | 78 |
| **CAPITAL COSTS** | | | | | | | | | | | | | | | | | | | | | | | | |
| 100% | Capital Costs | $M | **$1,103** | 64 | 58 | 200 | 147 | 90 | 30 | 34 | 36 | 43 | 75 | 91 | 72 | 60 | 19 | 21 | 1 | 19 | 1 | 1 | 11 | 8 |
| **NET CONTRIBUTION** | | | | | | | | | | | | | | | | | | | | | | | | |
| before profit royalty | | | | | | | | | | | | | | | | | | | | | | | | |
| | Net Contribution | $M | **$1,846** | -64 | -58 | -200 | -147 | -16 | 175 | 171 | 169 | 162 | 130 | 114 | 142 | 109 | 150 | 148 | 169 | 150 | 169 | 169 | 159 | 70 |
| | Cumulative | $M | | -64 | -123 | -323 | -471 | -486 | -311 | -140 | 29 | 191 | 321 | 435 | 578 | 686 | 836 | 984 | 1153 | 1303 | 1471 | 1640 | 1798 | 1868 |
| **Indicative Pre-tax Economic Measures** | | | | | | | | | | | | | | | | | | | | | | | | |
| | **IRR** | % | **20%** | | | | | | | | | | | | | | | | | | | | | |
| | **NPV (8%)** | $M | **$513** | | | | | | | | | | | | | | | | | | | | | |

### 22.5.2 Pre-tax Indicative Economic Results

The Wheeler River project pre-tax indicative base case economic results include:

- A net present value (NPV) of $513 million when discounting project cash flows back to 2021 using a discount rate of 8%;
- An internal rate of return (IRR) of 20.4%;
- A payback period of ~3 years from the start of production

Note that this analysis considers all project development costs prior to the start of pre-production capital (January 2021) as excluded costs and as such they do not factor into the above calculation.

SRK notes that this PEA mining study is preliminary in nature. The mineral resources within the PEA design plan disclosed in the mine plans include a portion of Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

### 22.5.3 Pre-tax Sensitivities

#### Basic Sensitivities

The base case results are based on (all in Canadian dollars):

| | |
|---|---|
| Uranium price | $59.40 per pound (US$44.00/lb and 1.35 CAD/USD) |
| Average plant feed grade | 3.91% $U_3O_8$ |
| Average site operating cost | $25.67 per pound |
| Total project capital cost | $1,103 million |

SRK prepared a sensitivity analysis by varying these four inputs. Table 22-2 and Table 22-3 show the impact of varying input values on the base case pre-tax economic indicators – NPV8% in millions of dollars and IRR.

Figure 22-1 and Figure 22-2 present these sensitivities in graphical format.

**Table 22-2: Sensitivity of NPV (8%)**

| | NPV(8%) Sensitivity ($M) | | | | | | |
|---|---|---|---|---|---|---|---|
| Variable | -30% | -20% | -10% | 0% | 10% | 20% | 30% |
| Capex | 714 | 647 | 580 | 513 | 446 | 380 | 313 |
| Opex | 795 | 701 | 607 | 513 | 419 | 326 | 232 |
| Uranium Price | (130) | 84 | 299 | 513 | 728 | 942 | 1157 |
| $U_3O_8$ Grade | 110 | 244 | 379 | 513 | 648 | 782 | 917 |

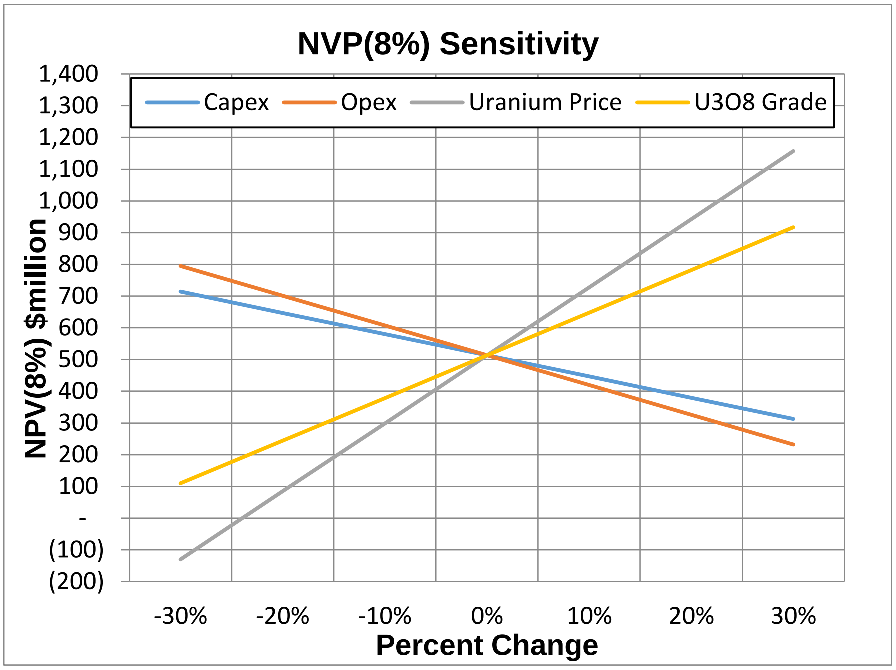



**Figure 22-1: NPV(8%) Sensitivity Graph**

**Table 22-3: Sensitivity of IRR%**

| | IRR Sensitivity (%) | | | | | | |
|---|---|---|---|---|---|---|---|
| **Variable** | **-30%** | **-20%** | **-10%** | **0%** | **10%** | **20%** | **30%** |
| Capex | 29% | 26% | 23% | 20% | 18% | 16% | 14% |
| Opex | 25% | 23% | 22% | 20% | 19% | 17% | 15% |
| Uranium Price | 3% | 11% | 16% | 20% | 24% | 28% | 31% |
| $U_3O_8$ Grade | 11% | 14% | 18% | 20% | 23% | 25% | 28% |

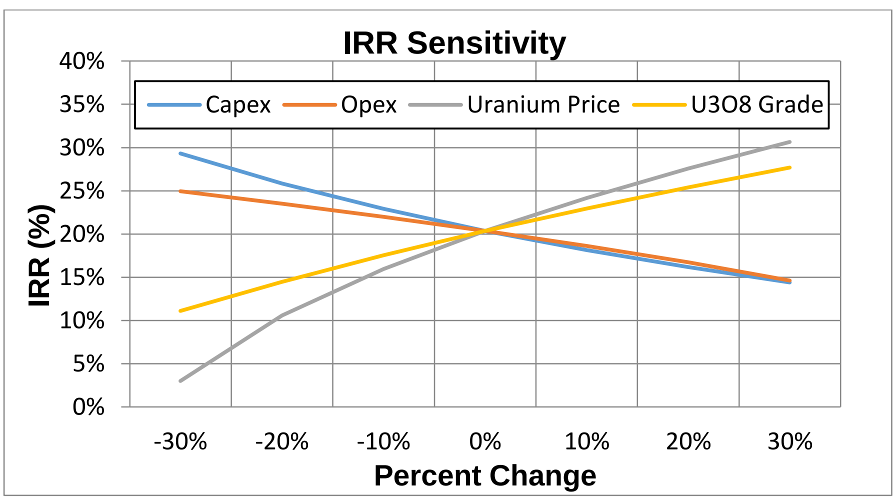



**Figure 22-2: IRR% Sensitivity Graph**

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.

### 22.5.4 Production Case Price Sensitivity

The project economic results are quite sensitive to the price of uranium. The PEA considers two pricing scenarios because of the long lead time to commercial production (2026) and the current uranium market:

(1) A Base case scenario using the long term contract price of US$44.00/lb as quoted by UxC as of March 28, 2016
(2) A Production case price sensitivity using the mid-case projected long-term contract price for the year 2026 per UxC's Uranium Market Outlook for Q1'2016. The production case considers a uranium price estimate of US$62.60/lb for the year 2026 when the project commercial production period is expected to begin.

Using the year 2026 estimated uranium price of US$62.60/lb, with all other variables held constant, the project's pre-tax NPV at 8% discounting increases to $1,420 million, the IRR increases to 34.1%, and the payback period decreases to ~18 months.

## 22.6 Post-tax Economic Analysis

### 22.6.1 Post-tax Cash Flow Model

The post-tax cash flow model is specific to Denison's 60% interest in the Wheeler River project and Denison's specific facts and circumstances. The model is based on the inputs noted in Section 22.1 and the following additional items:

- The impact on the project economics of Denison's 22.5% share of the MLJV's toll milling fees and production credits is included;
- Denison's share of project development costs (refer to Section 21.2) is included in the Company's estimated tax pools;
- The impact of the Saskatchewan Profit Royalty is included;
- Denison's expected provincial and federal income taxes payable are included; and
- Discounting at 8% has been selected for NPV calculations (see Section 22.5.1).

The following assumptions were used in computing the Federal and Provincial income tax, as well as Saskatchewan Profit Royalty amounts owing by Denison in the model:

- All applicable tax deductions currently available in Denison and its applicable Canadian subsidiaries at December 31, 2015, 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;
- The currently enacted tax laws and the proposed tax law amendments at the time of this PEA 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 threshold between the 10% and 15% net profit taxation tiers has been indexed up to Q4-2015 and then held constant at that amount through-out the life of the Wheeler 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:

**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 |
|---|---|---|
| Project Percentage | 100.0% | 60.0% |
| Gross Uranium Revenue | 6,080 | 3,648 |
| Toll Milling Fees | Excl. | 19 |
| Operating Costs | (2,690) | (1,614) |
| Operating Costs – Toll Milling Credits | Excl. | 28 |
| Saskatchewan Revenue Royalties, Surcharges | (441) | (264) |
| Operating Cash Flow | 2,949 | 1,817 |
| Capital Costs | (1,103) | (662) |
| Capital Costs – Project Development | Excl. | (15) |
| Net Contribution before Taxes | 1,846 | 1,140 |
| Saskatchewan Profit Royalties | Excl. | (175) |
| Canadian Federal / Provincial Income Taxes | Excl. | (197) |
| Net Contribution After Taxes | | 768 |
| NPV (8%) | $513M | $206M |

## 22.6.2 Post-tax Indicative Economic Results

The post-tax indicative base case economic results are shown below:

- A net present value (NPV) of $206 million when discounting project cash flows back to 2021 using a discount rate of 8%;
- An internal rate of return (IRR) of 17.8%

Note that this analysis considers all project development costs prior to the start of pre-production capital (January 2021) as excluded costs and as such they do not factor into the above calculation.

SRK notes that this PEA mining study is preliminary in nature. The mineral resources within the PEA design plan disclosed in the mine plans include a portion of Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

### 22.6.3 Post-tax Production Case Sensitivity

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.

Using the year 2026 estimated uranium price of US$62.60/lb, with all other variables held constant, the project's post-tax NPV to Denison, at 8% discounting, increases to $548 million and the IRR increases to 29.2%.

Table 22-5 contrasts the results of the Wheeler River project production case pre-tax cash flow model and the post-tax cash flow model as it applies to Denison:

**Table 22-5: Production Case Cash Flow Model: Pre-tax vs Post-tax Comparison**

| Item Description | Prod'n Case Pre-Tax Summary | Prod'n Case Post Tax Summary |
|---|---|---|
| Project Percentage | 100.0% | 60.0% |
| Gross Uranium Revenue | 8,650 | 5,190 |
| Toll Milling Fees | Excl. | 19 |
| Operating Costs | (2,690) | (1,614) |
| Operating Costs – Toll Milling Credits | Excl. | 28 |
| Saskatchewan Revenue Royalties, Surcharges | (627) | (376) |
| Operating Cash Flow | 5,333 | 3,247 |
| Capital Costs | (1,103) | (662) |
| Capital Costs – Project Development | Excl. | (15) |
| Net Contribution before Taxes | 4,230 | 2,570 |
| Saskatchewan Profit Royalties | Excl. | (403) |
| Canadian Federal / Provincial Income Taxes | Excl. | (522) |
| Net Contribution After Taxes | | 1,645 |

# 23 Adjacent Properties

There are no adjacent properties that are relevant the Wheeler River PEA.

# 24 Other Relevant Data and Information

## 24.1 Project Risks

### 24.1.1 Mineral Resources within PEA Design Plan

The mineral resources within PEA design plan (MR within PEA) are based partly on Inferred mineral resources. Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is a risk that the plant feed will not be fully realized in quantity or uranium grade, negatively impacting the project economics.

### 24.1.2 Ground Water Inflow

Both the McArthur River and Cigar Lake mines have been significantly impacted by schedule delays and increased costs due to unexpected ground water inflows.

For the Wheeler River project, the risk of an unexpected ground water inflow is greatest at the Phoenix deposit due to its location at the unconformity. Fortunately, production will be sourced first from the Gryphon zone where the groundwater risk is much lower, due to the deposit being well into basement rocks.

The risk at the Phoenix deposit will be mitigated by freeze wall protection, and the design will benefit from past operating experience at other basin uranium mines.

### 24.1.3 Geotechnical

The most significant geotechnical risks are potential instability of the Phoenix hangingwall when the JBS cavities are excavated, and also the risks related to the mining access drift development in the basement rock at Phoenix. The planned access drifts will encounter areas of weak rock and may also intersect water bearing fractures. There is a risk of development and production delays and increased costs.

### 24.1.4 Jet Bore Mining Method

The jet bore mining system (JBS) has been developed specifically for the Cigar Lake deposit over a period of many years. Commercial production based on this mining method was achieved at the Cigar Lake mine in late 2014.

This study assumes that JBS mining will be successful in the Phoenix deposit achieving similar results. There is a risk the JBS method will not perform as well as at Phoenix due to different rock mass characteristics.

This risk is partly mitigated by the continual improvement process that is occurring with JBS mining at the Cigar Lake mine. According to the project schedule, it will be about 15 years before jet bore mining will start at Phoenix.

### 24.1.5 Radiation Exposure

The risk related to employee radiation exposure is very low. The Wheeler River project will benefit from years of mining industry experience in northern Saskatchewan and elsewhere. Radiation exposure pathways and health risks are well understood and effective safeguards have been used in the industry. Public reports show that underground uranium mine workers receive only a fraction of the allowable safe limit of 50 mSv/y.

### 24.1.6 Schedule for Permitting and Approvals

The project development schedule (Figure 16-31) is based on SRK's understanding of typical provincial and federal permitting timelines. There is a risk that it may take longer to obtain full project approval, delaying the start of the construction on the site.

There is also a risk that further permitting and approvals will be required at the site selected for custom milling.

### 24.1.7 Metallurgy and Process

This study is based on custom milling the Wheeler River plant feed at an existing uranium mill in northern Saskatchewan. There is a risk that Denison may not be able to reach an agreement with mill owners under favourable terms. There is also a risk that sufficient plant capacity or tailings capacity may not be available for the Wheeler River feed, delaying the project or requiring additional capital to fund further modifications to the existing plant or construction of new milling facilities.

The composite samples used for metallurgical testing of each of Gryphon and Phoenix deposits do not reflect potential variability of mill feed, and that uranium milling recoveries of 97.0% for Gryphon and 98.1% for Phoenix may not be consistently achieved.

### 24.1.8 Capital and Operating Costs

Capital and operating cost estimates developed as part of this study are preliminary in nature, and this has been partly mitigated by the use of contingencies.

There is a risk that actual costs could be higher than those estimated for this study. Capital costs could be significantly impacted if the project experiences an unexpected major water inflow during the development period.

There is a risk that the provisional toll milling charge of $2.00/lb $U_3O_8$ could be higher.

## 24.2 Project Opportunities

### 24.2.1 Mineral Resource Expansion

Denison has announced their exploration plan for 2016 as follows:

> *Exploration activities at Wheeler River during 2016 are expected to focus on numerous unconformity and basement targets in the vicinity of the Gryphon deposit. Recent exploration results have continued to return mineralization in the area surrounding the Gryphon deposit and along the K-North trend, which hosts the Gryphon deposit. The results*

*in this area continue to suggest the potential for the discovery of additional zones of significant uranium mineralization.*

*A total of 47,000 metres of exploration drilling is planned at Wheeler River between the 2016 winter and summer drill programs, along with geophysical surveys at a total cost of $10.0 million (Denison's share, $6.0 million).*

It is possible that Denison's exploration work will discover additional mineralization that could become part of the Wheeler River project mining plan. During the winter 2016 program, drill testing within 200 metres north and northwest of the Gryphon deposit returned numerous high-grade intersections which have been reported in the Company's press releases. These results are not included in the current resource estimate or PEA.

### 24.2.2 Haulage Road Funding

This PEA includes the full estimated cost of a new 45 km section of haul road between the McArthur River mine and the Cigar Lake mine. It is apparent that this road link will benefit other parties.

There may be an opportunity to share the cost of this road section with other interested parties or the provincial government.

### 24.2.3 High Grade Uranium Mining and Handling

It is likely that there will be future improvement in the approved methods for underground mining and handling high grade uranium mineralization. The jet boring system used at Cigar Lake mine for example has been developed and improved over an approximate 20 year period, and continuing improvement is likely.

With mining at Gryphon planned first, and Phoenix mining planned well into the future, it is likely that continuous improvements made by currently operating high grade uranium mines will benefit the Wheeler River project.

### 24.2.4 Increase Phoenix Mining Rate

It may be possible to increase the Phoenix mining rate (Mlbs U3O8 per year) to utilize apparent excess capacity in the McClean mill during co-milling with Cigar Lake Phase 2. A detailed mine production schedule and justification are required at pre-feasibility study level to assess this potential opportunity.

### 24.2.5 Processing

The low Phoenix feed tonnage through the McClean Lake #1 leach circuit means that a large excess residence time capacity is potentially available. By increasing leaching residence time beyond the 12 hour design criteria tested to date, there is an opportunity to increase Phoenix leach recovery.

# 25 Interpretation and Conclusions

## 25.1 General Conclusions

In September 2015, SRK was mandated to work with Denison, and other consultants commissioned by Denison, on the preparation of a preliminary economic assessment (PEA) for the development of the Wheeler River project, targeting the mineral resources of the Phoenix and Gryphon deposits.

This technical report provides a summary of the results and findings from assessments of geological modelling, hydrogeology, rock mechanics, mineral resource estimation, underground mine design, processing options, infrastructure conceptual design, environmental management and permitting, capital and operating costs, and economic analysis. The level of investigation for each of these areas is considered to be consistent with that normally expected with scoping studies for resource development projects.

The results of the PEA indicate that the Wheeler River project has a positive economic return at the base case assumptions considered (Section 22). The results are considered sufficiently reliable to guide Denison's management in a decision to further develop the project. This would typically involve the preparation of a preliminary feasibility study.

The following sections summarize the conclusions for each area of study.

## 25.2 Geology and Mineral Resources (RPA)

*This report sub-section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada," Roscoe Postle Associates, November 25, 2015.*

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-kilometre-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% $U_3O_8$. Indicated Resources total 166,400 t at 19.14% $U_3O_8$ containing 70.2 million lb $U_3O_8$. Inferred Resources total 8,600 t at 5.80% $U_3O_8$ containing 1.1 million lb $U_3O_8$.

Mineralization at the Gryphon deposit, located 3 km northwest of Phoenix, occurs in basement rocks from 580 m to 850 m metres below surface and approximately 220 metres below the sub-Athabasca unconformity. The current resource estimate for the Gryphon deposit includes the A, B and C series lenses - a set of parallel, stacked, mineralized lenses that are broadly conformable with the basement geology, and are associated with a significant fault zone (G-Fault) that separates a thin unit of quartzite (Quartz-Pegmatite) from an overlying graphitic pelite (Upper Graphite). The lenses dip moderately to the southeast and plunge moderately to the northeast. The Gryphon deposit is approximately 450 metres along plunge, 80 metres across plunge and varies in thickness, between 2 and 20 metres, depending on the number of mineralized lenses present.

Current mineral resources for Gryphon, based on 55 diamond drill holes totalling 40,041 m, were estimated by RPA at a cut-off grade of 0.2% $U_3O_8$. Inferred mineral resources total 834,000 t at 2.31% $U_3O_8$ containing 43.0 million lb $U_3O_8$. In RPA's opinion, additional infill drilling on 25 m profile spacing or wedging off of current drill holes would be needed to bring the Inferred mineral resource into Indicated status.

## 25.3 Geotechnical

- At Phoenix, four conceptual geotechnical domains have been proposed:
  - Sandstone domain encompassing the fair to good rock mass conditions outside the influence of mineralization.
  - Broken Zone domain which encompasses the very poor rock mass conditions present in the immediate HW to the mineralized zone, above the unconformity.
  - Unconformity domain encompassing the variable fair to poor rock mass conditions within and 20 m to 30 m around the mineralized zone.
  - Basement domain that includes the generally fair to good conditions distal to mineralization and fault structures.
- Ground freezing at Phoenix is required to allow extraction of mineralization through mitigation of high water pressures and volumes, and improving the very poor HW rock mass conditions.
- At Gryphon, two conceptual geotechnical domains have been proposed:
  - Sandstone domain encompassing the fair to good rock mass conditions above the unconformity.
  - Basement domain that includes the generally fair to good conditions within and around the mineralized zone.
- Localized zones of poor quality rock mass are present at Gryphon that have been attributed to major fault structures. Enhanced ground support will be required through these zones, especially where they intersect planned stoping.
- The geotechnical conditions at Gryphon indicate that conventional mining methods are applicable for mineralization extraction.

## 25.4 Hydrogeology

- Cameco's McArthur River and Cigar Lake operations have both experienced significant groundwater inflow events (two at McArthur River and three at Cigar Lake) associated with mine workings in Athabasca sandstone.
- Athabasca sandstone hydraulic conductivity at Wheeler River is largely in the moderate permeability range, moderately variable, with localized high permeability sections. There is little matrix permeability in these rocks, and highly fractured sandstone and conglomeratic sections tend to be highly permeable.
- Crystalline basement rocks are highly variable in terms of bulk hydraulic conductivity, with a geometric mean hydraulic conductivity in the low permeability range. Significant sub-vertical structures may locally be highly permeable over widths of 10 m or more, as indicated by 2 of 25 tests conducted in the granite.
- No assessment of groundwater inflows has been conducted for Wheeler River to date, however, it is considered likely that maximum sustained routine inflows are likely to be less than 300 $m^3$/h, provided that diligent groundwater control measures, such as ground freezing in development areas within Athabasca sandstone, grouting of all boreholes from the bottom up and cover drilling and grouting are applied in areas of new development within the crystalline basement. These measures will also lessen the potential for sudden, large volume inflow events such as have occurred at McArthur River and Cigar Lake.

## 25.5 Mining

- Phoenix high-grade mineralization requires special mining methods to minimize radiation exposure to underground workers.
- Jet bore system mining is suitable for the Phoenix deposit. Freeze wall protection is required, and a tent configuration will work with the deposit geometry.
- The tent freeze holes can be positioned in the Phoenix HW in such a way that freeze wall development will strengthen the weak ground, helping to minimize excessive mining over break.
- A blind raise boring mining method could be applied at Phoenix, but it was considered less favourable than the jet boring method.
- Conventional longhole open stoping with backfill is suitable for the Gryphon zone. No freeze wall protection is needed.
- Underground production will begin with the Gryphon zone, and a production shaft and ventilation raise are required.
- Blind shaft boring with pre-grouting from surface and concrete shaft lining was selected as the best excavation method for vertical development based on competitive cost and schedule, and minimizing water inflow risk.
- An underground connection drift between the two deposits appears to be practical and offers synergies and capital cost savings as outlined in this report.
- The project schedule indicates that underground production ramp up at Gryphon can begin in 2025.
- Reasonable production rates are 6.0 Mlbs $U_3O_8$ per year for Gryphon over a seven-year mine life and 7.0 Mlbs per year for Phoenix over an additional nine-year mine life.

## 25.6 Metallurgy and Process

*This report sub-section has been reproduced from "Denison Mines Limited, Wheeler River Preliminary Economic Assessment, Process Aspects," Amec Foster Wheeler, January 21, 2016.*

### Processing at Wheeler River

Many process aspects for the Wheeler River project are based on designs that have been proven and are being successfully used at the McArthur River and Cigar Lake mines. Incorporation of these designs and practices should significantly reduce the risk in numerous areas of the Wheeler River project.

### Milling at McClean Lake

The metallurgical test results indicate that the Gryphon and Phoenix deposits are suitable for processing through the McClean Lake mill. Overall process recovery has been estimated at 97.0% $U_3O_8$ for Gryphon (due to lower grade), while Phoenix recovery is estimated at 98.1% $U_3O_8$.

The mine plan for the Gryphon and Phoenix deposits aligns well with making use of available capacity at the McClean Lake mill, while co-milling anticipated feeds from the Cigar Lake mine. The peak production rate of approximately 24 M lb/yr $U_3O_8$ occurs while co-milling Cigar Lake Phase 1 high grade and Gryphon low grade feeds, matching the intended total assessed license capacity of the mill.

The current scope of the McClean Lake mill modifications approved for construction is focused on enabling the full 18 M lb/yr $U_3O_8$ high grade milling of Cigar Lake Phase 1 feed, while a notional 4

M lb/yr $U_3O_8$ low grade co-milling capacity exists for a total of 22 M lb/yr $U_3O_8$. In the expected mill operating scenario, there is no constraint to production of 18 M lb/yr $U_3O_8$ of high grade feed, whereas production capacity constraints are identified for low grade feed due to tonnage restrictions.

To co-mill 144 kt/yr of Gryphon feed with Cigar Lake Phase 1/Phase 2 feed, expansion of the low grade leaching and solid/liquid separation circuits' capacities is required. The capital cost of this expansion is estimated at $19 million excluding contingency. Operating cost is estimated at $8.03/lb $U_3O_8$ for Gryphon co-milling.

To co-mill 26 kt/yr of Phoenix feed with Cigar Lake Phase 2 feed, some minor re-configurations of the slurry receiving, leaching, and solid/liquid separation circuits are required. Capital cost is estimated at $2 million excluding contingency. The operating cost is estimated at $6.03/lb $U_3O_8$ for Phoenix co-milling.

The downstream circuits at the McClean Lake mill (Clarification, SX, carbon columns, precipitation, calcining, packaging, crystallization) are assumed from stated expansion plans to be capable of 24 M lb/yr $U_3O_8$.

## 25.7 Waste Management

Although specific efforts to characterize the ML/ARD potential of the tailings or waste rock have not yet been completed, geological observations and results from the extensive assay database indicate:

- Concentrations of arsenic in the mineralization are at the low end of the range observed in tailings from other uranium deposits in the Athabasca Basin, indicating that arsenic concentrations in the tailings and tailings porewater are likely to be at the low end of the range found in all of the in-pit tailings management facilities in the Athabasca Basin. Concentrations of other trace elements are also within the range of other tailings in the area. The characteristics of the tailings suggest that mine production is likely to be an attractive option for custom milling operations because they are unlikely to have negative effects on the geochemical characteristics of the tailings or tailings porewater.
- Geological observations in drill core from the vicinity of the Phoenix and Gryphon deposits indicate that trace concentrations of pyrite occur in approximately 20% of the basement rock samples, and 5% of the sandstone samples, and that more significant concentrations (>1%) occur in approximately 1 to 2% of both sandstone and basement rock samples. These data indicate that there is potential for ARD and, therefore, metal leaching in a small portion of the waste rock, and that management and control of ML/ARD should be considered in the project evaluation.

Waste rock management plans will include measures to identify and segregate special waste (material with >0.03% $U_3O_8$), and waste rock with ML/ARD potential. Provisions have been made to store both special waste and waste rock with potential for ML/ARD on double lined pads with a leachate collection system to ensure that any runoff in contact with these materials can be collected and directed towards the treatment system. Where possible, these materials will be backfilled into the underground mine.

## 25.8 Water Management

- Mine water is expected to have concentrations of metals, uranium, molybdenum, selenium, arsenic and radium, which will require treatment to discharge the water. Concentrations of blasting residuals (nitrate and ammonium) may also be elevated and require treatment.

- A water treatment plant has been sized to treat an influent rate of 1,500 $m^3$/h consistent with the maximum expected non-routine mine water inflow.
- Water from the surge ponds will be pumped to the treatment system. The treatment system has a series of chemical processes to remove the constituents of concern. This conceptual design is based on water treatment at operating uranium mines.

## 25.9 Environmental and Permitting

- There are no environmental fatal flaws associated with this project. All potential environmental impacts can be successfully mitigated through the implementation of industry best practices.
- The most significant environmental concern associated with the project will be associated with the management of expected and unexpected mine water volumes.
- Based on the assumption the mineralization will be milled at McClean Lake, Denison's JV partner AREVA will be required to demonstrate the potential additional environmental impacts associated with managing the Wheeler River tailings and that the associated effluent at the McClean Lake project can be successfully mitigated.
- The project will be required to complete a federal and provincial environmental assessment. This assessment will be completed as a joint environmental assessment. It is estimated the assessment will require approximately 24 to 36 months to complete following the submission of a detailed project description.

## 25.10 Project Economics

The Wheeler River project total capital cost is estimated at $1,103 million, comprised of $560 million initial capital and $543 million sustaining capital. Total operating costs are estimated at $19.28 per pound $U_3O_8$ for the Gryphon mining phase and $29.90 per pound $U_3O_8$ for the Phoenix mining phase.

The Wheeler River project (100% basis) indicative pre-tax base case economic results include:

- A net present value (NPV) at 8% discounting to 2021 of $513 million
- An internal rate of return (IRR) of 20.4%
- A payback period of ~3 years from the start of production

Denison's 60% ownership interest in the Wheeler River project yields the following indicative post-tax base case economic results:

- An internal rate of return (IRR) of 17.8%
- A net present value (NPV) at 8% discounting of $206 million

This assessment of the potential viability of the mineral resources indicates that the project has a positive economic return and these results provide guidance to Denison to advance the project.

SRK notes that this PEA mining study is preliminary in nature. The MR within PEA estimated in report Section 16.3.3 include a portion of Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

# 26 Recommendations

Assessment of each area of investigation completed as part of this PEA suggests recommendations for further investigations to improve the preliminary designs and to mitigate risks.

The following report section summarizes the key recommendations arising from this study. Each recommendation is not contingent on the results of other recommendations and can be completed independently. Where appropriate, a cost for the recommended work is included, otherwise the cost is considered to be included in the capital and/or operating cost for the project.

## 26.1 Geology and Mineral Resources (RPA)

*This report sub-section has been reproduced from "Technical Report on a Mineral Resource Estimate for the Wheeler River Property, Eastern Athabasca Basin, Northern Saskatchewan, Canada," Roscoe Postle Associates, November 25, 2015.*

Should the project proceed into pre-feasibility, initial work will focus on environmental baseline studies, engineering field programs, and engineering studies.

The Wheeler River joint venture plans to continue exploration on the property in 2016, with emphasis expected to be on the areas to the northeast and southwest of Gryphon, as well as other targets on the property. In addition, an infill drilling program may be undertaken on the Gryphon deposit to bring the Inferred mineral resource into Indicated status if warranted by positive PEA results.

If further drilling is completed at Gryphon, RPA recommends that Denison continue to collect additional bulk density data to increase the confidence of estimated densities of the entire grade range.

## 26.2 Geotechnical

To advance the geotechnical level of detail, the following data collection and evaluation is recommended:

- Targeted geotechnical drilling is required in order to better understand the rock mass conditions at shaft locations and permanent infrastructure areas.
- Update structural models at Phoenix and Gryphon considering the additional drilling completed at each area.
- Undertake field point load testing and laboratory strength testing (uniaxial and tri-axial compressive strength) with stress-strain measurements for each test.
- A thorough geotechnical database review should be completed for quality control purposes, incorporating correlation with Golder (2014) and SRK (2015) geotechnical data. Screening and filtering of the data will allow the compilation of a robust geotechnical data set for use in geotechnical and mine design.
- Estimated cost is $1.4 million for drilling plus $0.5 million for planning, supervision and core logging.

## 26.3 Hydrogeology

- Future hydrogeological investigation should focus on identification and hydraulic testing of permeable structures and zones of proposed shaft and raise development.
- Evaluate drilling and testing requirements for a follow-up hydrogeological program. Coordinate this program with exploration drilling and the geotechnical requirements in order to use single boreholes for multiple investigations. Program field supervision for this work.
- Shallow hydrogeological testing should focus on areas of proposed shafts and raises, and should include construction and hydraulic testing of pumping wells and observation wells.
- Deep hydrogeological testing should include vibrating wire piezometer installation and deep down-hole hydrogeology testing (spinner testing, packer testing, mini-pumping tests).
- Following office-based analysis of hydrogeological drilling and testing data, develop a conceptual groundwater model to be used as the basis for construction of a 3D numerical groundwater model. This model will need to be calibrated on the basis of the data collected, and together with the mine plan, used to conduct groundwater inflow assessments for the various development stages of the mine life. Modelling should include sensitivity and uncertainty analysis.
- The cost of the full program, including compilation of a full hydrogeological report is estimated in the range of $300,000.
- The current practice of grouting exploration drill holes from the bottom up must be continued.

## 26.4 Mining

- A pre-feasibility mining study should be undertaken incorporating results of the 2016 Wheeler River exploration program. It will be necessary to convert the Gryphon deposit Inferred mineral resources to the Measured or Indicated classification for inclusion in the mining study.
- Technical and cost details should be investigated for shaft sinking alternatives as applied to the Wheeler River project shaft and ventilation raise requirements.
- Locations for the production shaft and ventilation raises should be selected based on field investigation and consideration of the geotechnical model.
- Further investigation is needed into the technical aspects of applying the jet boring system at Phoenix.
- Cost estimate is $1.5 million.

## 26.5 Water Management

- The size and number of surface water storage ponds, and the design of the water treatment plant should be refined as estimates of mine water chemistry and flow are refined in later phases of project development.
- A location must be identified and approved for releasing treated water to the environment.
- Review of existing surface hydrology data, including assessing suitability of monitoring network and data collected from it. Develop plans for further monitoring if required.
- Acquisition of long term meteorological data for extreme rainfall analysis, which will be required for storm water management design. Long term data will also be utilized to assess inter-annual variability in rainfall amount.
- Determine a water balance for the two mining sites.
- Cost estimate for this work is $50,000.

## 26.6 Mineral Processing

*This report sub-section has been reproduced from "Denison Mines Limited, Wheeler River Preliminary Economic Assessment, Process Aspects," Amec Foster Wheeler, January 21, 2016.*

The following items are recommended for future work in a pre-feasibility study:

- Perform pre-feasibility level process engineering design and cost estimation for the Wheeler River site's underground and surface plant feed handling facilities, considering precedents set by current Saskatchewan uranium operations such as Cigar Lake.
- Check on the maximum permissible grade of coarse dry muck haulage on public roads. Current understanding is the limit is 2.4% $U_3O_8$, compared to a predicted Gryphon run-of-mine grade of 1.90%. This will influence the degree of grade blending required for truck loading.
- Perform optimization test work on Gryphon and Phoenix deposits for grinding, leaching and CCD circuits' performance. In particular, test extended leaching time for Phoenix to take advantage of anticipated available circuit capacity for potential recovery increase, as well as bench scale filtration testing. Re-confirm production of on-spec yellowcake. Test effluent and tailings treatment. This testwork would require up to 100 kg of sample that can be sourced from exploration drill core.
- Perform test work to investigate potential for hydrogen evolution from the Gryphon and Phoenix deposits. If high rates of hydrogen evolution are observed, then its management may require alteration of the JEB mill #1 leach circuit design. This testwork would require up to 20 kg of sample that can be sourced from exploration drill core.
- Perform pre-feasibility level process engineering design and cost estimation for JEB mill modifications, based upon updated design criteria derived from the recommended test programs.
- Should commercial negotiations proceed, Denison should request AREVA to validate design capacities for each of the downstream circuits (clarification, SX, carbon columns, precipitation, calcining, packaging, crystallization), and identify any equipment that may require upgrades to achieve 24 M lb/yr $U_3O_8$.
- For the scope of Wheeler River and McClean Lake plant feed processing facilities described in Section 17 (excluding transportation from mine to mill), the approximate cost of pre-feasibility level investigations for the next phase is $0.8 million.

## 26.7 Geochemistry and Waste Management

As the project advances, additional characterization of both the tailings and waste rock will be required. This information will be used to support environmental permitting, more detailed planning to determine the requirements for segregation, storage and handling of the waste rock, and for support of custom milling/tailings storage negotiations.

For tailings, characterization will need to be completed in conjunction with further metallurgical testing. These tests will ideally be completed in consultation with processing engineers from the custom milling facility, and should simulate both the metal recovery and tailings preparation steps used at that facility as closely as possible. Samples would ideally include a few composites representing the range of mineralization that will be encountered over the life of mine. The process solutions, final effluent, and the final tailings slurry (solids and liquids) should be analyzed for a complete suite of major and trace elements, and mineralogical characterization should also be completed on the tailings solids. Lastly, the tailings slurry should be subjected to an anoxic aging test to simulate changes that are likely to occur over the short to medium-term following deposition into the tailings management facility.

SRK recommends:

- static testing (acid base accounting tests) on a moderate number of samples from each deposit area (approximately 50).
- Kinetic testing – including both laboratory and field based tests on a representative subset of samples.
- Costs for static testing, including a site visit to examine drill core, sample selection and data interpretation are expected to be on the order of $40,000.

- Kinetic testing, including sample selection, set up and analysis of approximately 8 field and 8 laboratory based tests, and data interpretation is expected to be on the order of $250,000.

## 26.8 Environmental and Social

In conjunction with pre-feasibility level engineering, comprehensive environmental and social baseline studies should be initiated.

These studies should:

- Include a complete characterization of the aquatic and terrestrial environment;
- Heritage and archeological investigations;
- Geochemical, hydrological, and hydrogeological characterization
- Establish a detailed stakeholder engagement plan
- Identify primary stakeholder groups as well as additional groups whose broader based membership is focused on the socio-economic well-being of northern Saskatchewan.
- Costs for the environmental assessment process is estimated at approximately $3 million which includes the engagement plan and baseline studies.

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# CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date March 31, 2016.

I, Ken Reipas, residing at 43 Deverell Street, Whitby, ON do hereby certify that:

1) I am a Principal Consultant (Mining Engineering) with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 1300 - 151 Yonge Street, Toronto, Ontario, Canada;
2) I am a graduate of Queen's University, Kingston, ON in 1981, I obtained a BSc degree. I have practiced my profession continuously since 1981. My experience is in the areas of underground mine engineering, mine production, and consulting;
3) I am a professional engineer registered with the Professional Engineers Ontario - PEO License No.: 100015286;
4) I have personally inspected the subject project on January 29, 2015;
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 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;
6) As a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the main author of this report and responsible for sections 1.1, 1.4, 1.6, 1.8, 1.11, 2, 3, 15, 16.3 to 16.8 except 16.5.2 and 16.8.3, section 18 except 18.4, 18.20 and 18.21, section 19, section 21.1 except 21.1.2, sections 21.2, 21.3, 21.4.2 to 21.4.7, 21.5 to 21.8 except 21.8.3, sections 22, 23, 24 except 24.1.3, 24.1.6, 24.1.7 and 24.2.4, sections 25.1, 25.5, 25.10, 26.4 and 27, and co-authored sections 1.10, 1.12, 1.13, 16.5.2 and 21.4.1 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;
10) SRK Consulting (Canada) Inc. was retained by Denison Mines Corporation to prepare a preliminary economic assessment audit of the Wheeler River uranium project. The preceding report is based on a site visit, a review of project files and discussions with Denison Mines Corporation 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 Corporation; and
12) As of the date of this certificate, 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"]*

Toronto
April 08, 2016

Ken Reipas, PEng
Principal Consultant

# CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date March 31, 2016.

I, William E. Roscoe, Ph.D., P.Eng., do hereby certify that:

1) I am a Principal Geologist with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, Canada 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 (No. 39633011) and designated as a Consulting Engineer in the Province of Ontario. I have worked as a geologist for a total of 46 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Thirty-one years of 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
    - 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 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 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;
5) I have personally inspected the subject project on October 30, 2012, and June 16, 2014;
6) I am responsible for sections 1.2, 1.3, 4 to 12 (except 7.6 which is co-authored), 14, 25.2 and 26.1, and accept professional responsibility for these sections of this technical report;
7) I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
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; and a technical report on a Mineral Resource estimate for the Wheeler River Property dated November 25, 2015;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith;
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 Corporation; and
11) As of the date of this certificate, 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"]*

Toronto
April 08, 2016

William E. Roscoe, Ph.D., P.Eng
Principal Geologist
Roscoe Postle Associates Inc.

# CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date March 31, 2016.

I, K. Todd Hamilton, residing at 526 Anthony Drive, Oakville, ON do hereby certify that:

1) I am a Principal Consultant (Hydrogeology) with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 1300 - 151 Yonge Street, Toronto, Ontario, Canada;
2) I am a graduate of the University of Waterloo, ON, where, in 1985 I obtained a BSc degree in Earth Sciences. I am also a graduate of the University of New South Wales, NSW, Australia, where I obtained a MAppSc degree in Hydrogeology and Groundwater Management. I have practiced my current profession continuously since 1989. My principal experience is in the areas of mine hydrogeology and consulting;
3) I am a professional engineer registered with the Association of Professional Engineers and Geoscientists of British Columbia - PGeo License No.: 23013;
4) I have personally inspected the subject project on June 9-10, 2015;
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 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;
6) I, as a qualified person, I 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 16.1, 25.4 and 26.3 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;
10) SRK Consulting (Canada) Inc. was retained by Denison Mines Corporation to prepare a preliminary economic assessment audit of the Wheeler River uranium project. The preceding report is based on a site visit, a review of project files and discussions with Denison Mines Corporation 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 Corporation; and
12) As of the date of this certificate, 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.

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| | *[“signed and sealed”]* |
| Toronto | K. Todd Hamilton, P.Geo (BC) Lic. # 23013 |
| April 08, 2016 | Principal Consultant |

# CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date March 31, 2016.

I, Bruce Murphy, residing in North Vancouver, British Columbia do hereby certify that:

1) I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 2200 - 1066 West Hastings Street, Vancouver, BC, Canada;
2) I am a graduate of University of the Witwatersrand, Johannesburg, South Africa with a M.Sc. degree in Mining Engineering. I have practiced my profession continuously since graduation (1989) working in the rock engineering field on operating mines till 2002 and then in the consulting field;
3) I am a Fellow of the South African Institute of Mining and Metallurgy (FSAIMM);
4) 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 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;
5) As a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
6) I am a contributing author of this report and responsible for sections 1.5, 16.2, 24.1.3, 25.3 and 26.2, and accept professional responsibility for these sections of this technical report;
7) I have had no prior involvement with the subject property;
8) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith;
9) SRK Consulting (Canada) Inc. was retained by Denison Mines Corporation to prepare a preliminary economic assessment audit of the Wheeler River uranium project. The preceding report is based on a site visit, a review of project files and discussions with Denison Mines Corporation personnel;
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 Corporation; and
11) As of the date of this certificate, 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"]*
Bruce Murphy, FSAIMM
Principal Consultant

Vancouver
April 08, 2016

# CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date March 31, 2016.

I, Greg Newman, residing at 311 Adaskin Cove, Saskatoon, SK do hereby certify that:

1) I am a Principal of the firm of Newmans Geotechnique Inc. (NGI) with an office at 311 Adaskin Cove, Saskatoon, SK, Canada;
2) I am a graduate of University of Saskatchewan having obtained a BA in 1987, a BE (mech.) in 1992 and an M.Sc (geotech.) in 1995. My area of practice is in artificial ground freezing and I worked in this area since 1995.
3) I am a professional engineer registered with the Association of Professional Engineers of Saskatchewan - PEng License No.: 09054;
4) 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 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;
5) I, as a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the author or co-author of sections 16.5.2, 16.8.3, 18.4 and 21.4.8 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 not received, nor do I expect to receive, any interest, directly or indirectly, in the Wheeler River uranium project or securities of Denison Mines Corporation; and
12) That, as of the date of this certificate, 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"]*

Toronto
April 08, 2016

Greg Newman, PEng
Principal, NGI

# CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date March 31, 2016.

I, Mark Liskowich, residing at 118 Stechishin Terrace, Saskatoon, SK do hereby certify that:

1) I am a Principal Consultant (Environmental Management) with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 205 - 2100 Airport Drive, Saskatoon, Saskatchewan, Canada;
2) I am a graduate of the University of Regina, Regina, SK in 1989, I obtained a BSc degree. I have practiced my profession continuously since 1989. My experience is in the areas of environmental management of mining and mineral exploration;
3) I am a professional geoscientist registered with the Professional Engineers & Geoscientists of Saskatchewan - License No.: 10005;
4) I did not personally visit the project area;
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 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;
6) I, as a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am a co-author of this report and responsible for sections 1.9, 20 except 20.4.3, 21.4.11, 24.1.6, 25.9 and 26.8 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;
10) SRK Consulting (Canada) Inc. was retained by Denison Mines Corporation to prepare a preliminary economic assessment audit of the Wheeler River uranium project. The preceding report is based on a site visit, a review of project files and discussions with Denison Mines Corporation 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 Corporation; and
12) That, as of the date of this certificate, 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"]*

Saskatoon
April 08, 2016

Mark Liskowich, PGeo
Principal Consultant

# CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada,* effective date March 31, 2016.

I, Tom Sharp, residing at 662 West Keith Road, North Vancouver, BC do hereby certify that:

1) I am a Principal Consultant (Water Treatment Engineering) with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 2200 – 1066 West Hastings Street, Vancouver, British Columbia;
2) I am a graduate of Montana State University and Montana Tech with a B.Sc. and M.Sc. in Biological Sciences (1988 and 1993), M.Sc. in Environmental Engineering (1996) and a Ph.D. in Civil Engineering (1999);
3) I am a Professional Engineer (P.Eng. #36988) registered with the Association of Professional Engineers, Geologists of British Columbia. I am also a registered Professional Engineer in Northwest Territories, Nunavut and Montana. I am a Member of the Society for Mining, Metallurgy and Exploration;
4) I have not visited 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 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;
6) As a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am responsible for Sections 18.20, 25.8 and 26.5, and co-authored Section 21.4.1 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;
10) SRK Consulting (Canada) Inc. was retained by Denison Mines Corporation to prepare a preliminary economic assessment audit of the Wheeler River uranium project. The preceding report is based on a site visit, a review of project files and discussions with Denison Mines Corporation 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 Corporation; and
12) As of the date of this certificate, 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.

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| | *["signed and sealed"]* |
| Vancouver | Tom Sharp, PEng |
| April 08, 2016 | Principal Consultant |

# CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: *Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada*, effective date March 31, 2016.

I, Kelly Sexsmith, residing at 517 East 10th Street, North Vancouver, BC do hereby certify that:

1) I am a Principal Consultant (Environmental Geochemist) with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 2200 - 1066 West Hastings Street, Vancouver, BC, Canada;
2) I am a graduate of University of British Columbia, Vancouver, BC (BSc in 1991), and the Colorado School of Mines (M.S. in 1996). I have practiced my profession continuously since 1991. My experience is in the areas of mine waste geochemistry and consulting;
3) I am a professional geologist registered with the Association of Professional Engineers and Geoscientists in BC- PGeo License No.: 122254, and the Association of Professional Engineers and Geoscientists in Saskatchewan – PGeo Licence No: 13977;
4) 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 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;
5) I, as a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
6) I am a contributing author of this report and responsible for sections 18.21, 20.4.3, 25.7 and 26.7, and co-authored section 7.6 and accept professional responsibility for these sections of this technical report;
7) I have had no prior involvement with the subject property;
8) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith;
9) SRK Consulting (Canada) Inc. was retained by Denison Mines Corporation to prepare a preliminary economic assessment audit of the Wheeler River uranium project. The preceding report is based on a site visit, a review of project files and discussions with Denison Mines Corporation personnel;
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 Corporation; and
11) That, as of the date of this certificate, 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"]*

Vancouver
April 08, 2016

Kelly Sexsmith, PGeo
Principal Consultant

# CERTIFICATE OF LORNE D. SCHWARTZ

I, Lorne D. Schwartz, P.Eng., do hereby certify that:

1. I am Senior Engineering Specialist, of Amec Foster Wheeler Americas Limited ("**Amec Foster Wheeler**"), a corporation with a business address of 301 – 121 Research Drive, Saskatoon, Saskatchewan, S7N 1K2.
2. I am an author of a technical report entitled "Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada", with an effective date of March 31, 2016 (the "**Technical Report**").
3. I graduated with a degree in B. Sc. Metallurgical Engineering (co-op program) from the University of Alberta in 1994.
4. From 1995 to present I have been actively employed as an engineer in the area of extractive metallurgy. My uranium processing experience consists of employment as a metallurgist at AREVA's McClean (JEB) mill from 1998-2002, metallurgist at Cameco's Rabbit Lake mill from 2002-2007, Cameco corporate office from 2007 to 2011, Hatch Ltd. from 2011 to 2014, and Amec Foster Wheeler from 2014 to present.
5. I am a member, in good standing, of APEGS in the Province of Saskatchewan, member #15328.
6. I have not visited the Wheeler River site.
7. I am responsible for sections 1.7, 13, 17, 21.1.1, 21.4.9, 21.4.10, 21.8.3, 24.1.7, 25.5, 25.6 and 26.6, and co-authored 1.10, 1.12, and 1.13 of the Technical Report.
8. 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**.**
9. As a Senior Engineering Specialist, I have had prior involvement with the Wheeler River property that is the subject of the Technical Report since 2014. The nature of my prior involvement with the Wheeler River property included guidance and interpretation of metallurgical test programs for the Phoenix deposit in 2014 and the Gryphon deposit in 2015. I also authored a report on toll milling Wheeler River feeds at the McClean Lake mill in May 2015.
10. 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.
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. I am independent of the issuer, Denison Mines Corp., applying all of the tests in Section 1.5 of NI 43-101.

Dated this 8th day of April, 2016, in Saskatoon, Saskatchewan.

*["signed and sealed"]*

Lorne D. Schwartz, P. Eng.
Senior Engineering Specialist
Amec Foster Wheeler Americas Limited.

# CERTIFICATE OF CHARLES R. EDWARDS

I, Charles R. Edwards, P.Eng., do hereby certify that:

1. I am Director, Metallurgy, of Amec Foster Wheeler Americas Limited ("Amec Foster Wheeler"), a corporation with a business address of 301 – 121 Research Drive, Saskatoon, Saskatchewan, S7N 1K2.
2. I am an author of a technical report entitled "Preliminary Economic Assessment for the Wheeler River Uranium Project, Saskatchewan, Canada", with an effective date of March 31, 2016 (the "Technical Report").
3. I graduated from Queen's University with a B. Sc. (Engineering Chemistry) in 1965 and an M.Sc. (Chemical Engineering) in 1969.
4. From 1974 to present I have been actively employed as an engineer in the area of extractive metallurgy. My uranium processing experience consists of employment as Research Engineer with Eldorado Nuclear Limited, Ottawa from 1978-1980, as Chief Metallurgist at Eldor Mines' Rabbit Lake mill from 1986-1987, as Senior Metallurgical/Process Engineer with Kilborn Western Limited from 1987-1992, as Regional Director, Mineral Development Agreements, with Energy, Mines and Resources Canada from 1992-1994, as Senior Metallurgist (1994-1996), Chief Metallurgist (1996-2000), Manager, Process Engineering (2000-2002), Director, Engineering & Projects (2002-2007) and Principal Metallurgist (2007-2008) in Cameco's corporate office, and as Director, Metallurgy with Amec Foster Wheeler from 2008 to present.
5. I am a member, in good standing, of APEGS in the Province of Saskatchewan, member #05915.
6. I have not visited the Wheeler River site.
7. I am responsible for sections 1.7, 13, 17, 21.1.1, 21.4.9, 21.4.10, 21.8.3, 24.1.7, 24.2.5, 25.6 and 26.6, and co-authored sections 1.10, 1.12 and 1.13 of the Technical Report.
8. 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.
9. As Director, Metallurgy, I have had prior involvement with the Wheeler River property that is the subject of the Technical Report since 2014. The nature of my prior involvement with the Wheeler River property included guidance and interpretation of metallurgical test programs for the Phoenix deposit in 2014 and the Gryphon deposit in 2015. I also authored a report on toll milling Wheeler River feeds at the McClean Lake mill in May 2015.
10. 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.
11. I have read NI 43-101 and the sections of the Technical Report that I am responsible for, have been prepared in compliance with that Instrument.
12. I am independent of the issuer, Denison Mines Corp., applying all of the tests in Section 1.5 of NI 43-101.

Dated this 8th day of April, 2016, in Saskatoon, Saskatchewan.

*["signed and sealed"]*
Charles R. Edwards
Director, Metallurgy
Amec Foster Wheeler Americas Limited