Arrow Deposit, Rook I Project

Saskatchewan
NI 43-101 Technical Report on Feasibility Study

Prepared For:
NexGen Energy Ltd.

Prepared By:
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Effective Date:
22 February 2021

Project Number:
169520593

IMPORTANT NOTICE

This notice is an integral component of the NexGen Energy Ltd. Arrow Deposit, Rook I Project Technical Report ("Technical Report" or "Report") and should be read in its entirety and must accompany every copy made of the Technical Report. The Technical Report has been prepared in accordance with the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects published by the Canadian Securities Administrators ("NI 43-101").

The Technical Report has been prepared for NexGen Energy Ltd (NexGen) by Stantec Consulting Ltd. (Stantec), Wood Canada Limited (Wood), Rosco Postle Associates (RPA) now a part of SLR Consulting (Canada) Ltd. and Golder Associates Ltd. as the Report Contributors. The Technical Report is based on information and data supplied to Report Contributors by NexGen and other parties. The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in the services of Report Contributors, based on: i) information available at the time of preparation of the Report, and ii) the assumptions, conditions, and qualifications set forth in this Report.

Each portion of the Technical Report is intended for use by NexGen subject to the terms and conditions of its contracts with the Report Contributors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of the Technical Report, by any third party, is at that party's sole risk.

The results of the Technical Report represent forward-looking information. The forward-looking information includes pricing assumptions, sales forecasts, projected capital and operating costs, mine life and production rates, and other assumptions. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially, are presented in the body of this Report under each relevant section.

The Report Authors have used their experience and industry expertise to produce the estimates in the Technical Report. Where these estimates have been made, they are subject to qualifications and assumptions, and it should also be noted that all estimates contained in the Technical Report may be prone to fluctuations with time and changing industry circumstances.

CAUTIONARY NOTE TO U.S. INVESTORS

This Mineral Reserves and Mineral Resources Table contained in the Technical Report includes Mineral Reserves and Mineral Resources classification terms that comply with reporting standards in Canada and the Mineral Reserves and the Mineral Resources estimates are made in accordance with NI 43-101. NI 43-101 is a rule developed by the Canadian Securities Administrators that establishes standards for all public disclosure an issuer makes of scientific and technical information concerning mineral projects. These standards differ from the requirements of the SEC under Industry Guide 7 and set out in SEC's rules that are applicable to domestic United States reporting companies. Consequently, Mineral Reserves and Mineral Resources information included in the Report is not comparable to similar information that would generally be disclosed by domestic U.S. reporting companies subject to the reporting and disclosure requirements of the SEC.

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CERTIFICATE OF QUALIFIED PERSON

1-1760 Regent Street
Sudbury, ON P3E 3Z8
Canada

I, Mark Hatton, P.Eng. am employed as a Mining Engineer by Stantec Consulting Ltd. of 1-1760 Regent St, Sudbury, ON, Canada P3E 3Z8.

This certificate applies to the technical report titled “Arrow Deposit, Rook I Project, Saskatchewan, NI 43-101 Technical Report on Feasibility Study”, dated 22 February 2021 (the “technical report”).

I am registered with the Professional Engineers of Ontario (PEO) as a P.Eng. (No. 90433939). I graduated from Queen’s University in Kingston Sudbury Ontario with a Bachelor of Engineering in Mining Engineering in 1994.

I have practiced my profession continuously since 1994 and have experience in mining operations and consulting. I have worked as a Mining Engineer for 26 years.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101).

I visited the NexGen Rook I Project site on 12 June 2019.

I am the co-author of the Report and am responsible for Sections 1.1, 1.2, 1.12, 1.13, 1.15, 1.17, 1.18, 1.19, 1.20, 1.21, 1.22, 1.23, 1.24, parts of Section 2; parts of Section 3; Section 15; Section 16; Section 18.4, 18.6.3, 18.6.4, 18.7, 18.8, 18.9, 18.10; Section 19; Section 20.3, Section 21; Section 22; Section 24; Sections 25.6, 25.7, 25.9, 25.11, 25.12, 25.13, 25.14, 25.15, 25.16; parts of Section 26.1, 26.2; Parts of Section 27.

I am independent of NexGen Energy Ltd. as independence has been described in section 1.5 of NI 43-101.

I have read NI 43-101 and the sections of the technical report for which I am responsible and have confirm that the technical report has been prepared in compliance with that Instrument.

At the effective date of the Report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Sudbury, ON: 19 February 2021.

“Signed and Sealed”

Mark Hatton, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

301-121 Research Drive,
Saskatoon, SK S7N 1KS
Canada

I, Paul O'Hara, P.Eng., am employed as a Manager, Process with Wood Canada Limited.

This certificate applies to the technical report titled "Arrow Deposit, Rook I Project, Saskatchewan, NI 43-101 Technical Report on Feasibility Study", dated 22 February 2021, (the "technical report").

I am a member of Association of Professional Engineers and Geologists of Saskatchewan (APEGS) member number 11687. I graduated from the University of British Columbia, with a Bachelor of Science degree in Mining and Mineral Process Engineering in 1986.

I have practiced my profession for 34 years. I have been directly involved in the operation of copper, gold, and potash processing plants in Canada. My relevant experience includes process design, surface infrastructure, capital and operating cost estimates cash flow modelling and financial analysis for gold, potash and uranium process plants in Canada, England, Jordan and the Republic of Congo.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Rook I Project on 16 May 2018 and again on 12 June 2019.

I am responsible for Sections 1.10, 1.14, 1.15, 1.16.1, 1.16.3, 1.16.4, 1.16.5, 1.24.2, 1.24.3; Section 2; Section 13; Section 17; Sections 18.1, 18.2, 18.3, 18.5, 18.6.1, 18.6.2, 18.10; Sections 20.2, 20.4, 20.6.2, 20.6.3; Sections 21.1.4, 21.1.6, 21.1.8, 21.1.9, 21.1.10, 21.2.4, 21.2.5 21.2.7, Sections 25.4, 25.8, 25.10.2 to 25.10.5; Section 26.3, and Section 27 of the technical report.

I am independent of NexGen Energy Ltd. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Rook I Project during the preparation of the feasibility study.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated: 22 February 2021

Signed and sealed____________________

Paul O'Hara, P.Eng. (Saskatchewan)

CERTIFICATE OF QUALIFIED PERSON

1568 Cole Blvd, Suite 100
Lakewood, CO 80401

I, Mark B. Mathisen, C.P.G., am employed as Principal Geologist with Roscoe Postle Associates USA Ltd., now part of SLR International Corporation, of 1658 Cole Blvd, Lakewood, CO, USA 80401.

This certificate applies to the technical report titled "Arrow Deposit, Rook I Project, Saskatchewan, NI 43-101 Technical Report on Feasibility Study", dated 22 February 2021 (the "technical report").

I am a Registered Professional Geologist in the State of Wyoming (No. PG-2821) and a Certified Professional Geologist with the American Institute of Professional Geologists (No. CPG-11648). I graduated from Colorado School of Mines in 1984 with a B.Sc. degree in Geophysical Engineering.

I have worked as a geologist for a total of 25 years since my graduation.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101).

I visited the NexGen Rook I Project site on 19–20 June 2016 and 22–25 July 2017.

I am a co-author of the Report and am responsible for Sections 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 1.11; parts of 1.22; 1.24.5; parts of Section 2; parts of Section 3; Section 4; Section 5; Section 6; Section 7; Section 8; Section 9; Section 10; Section 11; Section 12; Section 14; Section 23; Sections 25.1, 25.2, 25.3, 25.5, parts of 25.15, parts of 25.16; parts of Section 26; Parts of Section 27.

I am independent of NexGen Energy Ltd. as independence has been described in section 1.5 of NI 43-101.

I have read NI 43-101 and the sections of the technical report for which I am responsible and have confirm that the technical report has been prepared in compliance with that Instrument.

At the effective date of the Report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated: 22 February 2021.

Signed and Sealed *Mark B. Mathisen*

Mark B. Mathisen, C.P.G.

CERTIFICATE OF QUALIFIED PERSON DAN WALKER

I, Dan Walker, P.Eng., state that:

(a) I am a Principal, Senior Water Resources Engineer at:
Golder Associates Ltd.
Suite 200 - 2920 Virtual Way
Vancouver, BC, V5M 0C4

(b) This certificate applies to the technical report titled Arrow Deposit, Rook I Project, Saskatchewan, NI 43-101 Technical Report on Feasibility Study with an effective date of 22 February, 2021 (the “Technical Report”).

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (the “Instrument”). My qualifications as a qualified person are as follows. I am a graduate of the University of Guelph, University of Stirling and the University of British Columbia with a Bachelor of Science (Engineering) in Environmental Engineering, a Master of Science in Environmental Management and a Doctor of Philosophy (Civil Engineering), respectively. I am a registered professional engineer in British Columbia and in the Northwest Territories and Nunavut. My relevant experience after graduation and over 18 years of practice in the mining industry for the purpose of the Technical Report includes mine water management planning and infrastructure design, environmental assessment and permitting mining projects for environmental compliance.

(d) The requirement for a site visit is not applicable to me.

(e) I am responsible for Sections 1.16.2, 1.16.6, 1.16.7, 20.0, 20.1, 20.2, 20.4.1, 20.5, 20.6.1, 20.7, 20.8, 25.10.1, 25.10.6, 25.10.7, 26.4 and 27 (only for inclusion of references applicable to responsible sections) of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of the Instrument.

(f) My prior involvement with the property that is the subject of the Technical Report includes providing engineering input to environmental assessment and permitting for the Rook I Project with specific focus towards mine water and mine waste management planning.

(g) I have read National Instrument 43-101. The part of the Technical Report for which I am responsible has been prepared in compliance with this Instrument; and

(h) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the part of Technical Report for which I am responsible, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Vancouver, BC this 19 of February, 2021.

"Signed and Sealed"

Dan Walker, P.Eng.

1.0 SUMMARY

NexGen Energy Ltd. (NexGen) retained Stantec Consulting Ltd. (Stantec), Wood Canada Limited (Wood), Roscoe Postle Associates Inc. (RPA) part of SLR Consulting (Canada) Ltd. (SLR), and Golder Associates Ltd. (Golder) to complete a technical report on the results of a Feasibility Study (FS) regarding the Arrow uranium deposit within the Rook I Project (the Project) site in Saskatchewan, Canada.

1.1 Principal Outcomes

The 2021 FS is based on NexGen processing 4,575 kt of uranium grading 2.37% U_3O_8 (probable reserve) at the Arrow Deposit. Processing will take place over an 11-year mine life to produce 233.6 Mlb of recovered yellowcake (YC), with an average metallurgical recovery of 97.5%.

The economic analysis is based on the timing of a final investment decision (FID), and it does not include the pre-commitment early works capital costs, which are costs NexGen intends on expending prior to the FID. The pre-commitment early works scope includes preparing the site, completing initial freeze hole drilling, and building the supporting infrastructure (i.e., concrete batch plant, Phase I camp accommodations, and bulk fuel storage) required for the Project. Costs for the pre-commitment early works will total an estimated C$158 million.

The total capital cost carried in the economic model is C$1,573.9 million, inclusive of C$1,142.0 million in pre-production capital costs, C$362.4.0 million of sustaining capital costs, and C$78.6.0 million of closure / reclamation costs, less $9.1 million in salvage.

Total life of mine (LOM) operating costs are estimated to be C$1,769.8 million.

On a pre-tax basis, the net present value (NPV) at 8% is C$5,577.0 million, the internal rate of return (IRR) is 64.9%, and the assumed payback period is 0.8 years.

On a post-tax basis, the NPV at 8% is C$3,465.0 million, the IRR is 52.4%, and the assumed payback period is 0.9 years.

The payback period is calculated from the start of production.

1.2 Terms of Reference

This report is prepared as an NI 43-101 Technical Report for NexGen by Stantec, Wood, RPA, and Golder, and will be filed with the Canadian Securities Administrators (CSA) on the System for Electronic Document Analysis and Retrieval (SEDAR) filing system. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort based on.

- Information available at the time of preparation
- Data supplied by outside sources
- The assumptions, conditions, and qualifications set forth in this report.

This report is written in Canadian English and uses SI units of measurement, unless otherwise indicated. Monetary units are expressed in Canadian dollars (CAD), except for uranium pricing, which is expressed in United States dollars (USD).

1.3 Project Setting

The Project is located in northwest Saskatchewan, approximately 40 km east of the Alberta–Saskatchewan border, 150 km north of the town of La Loche, and 640 km northwest of the city of Saskatoon. The Project can be accessed via all-weather gravel, Highway 955, which travels north-south approximately 8 km west of the Arrow Deposit. From Highway 955, a 13 km long all-weather, single-lane road provides access to the western portion of the Project, including the Arrow Deposit area.

The Project will take place in a region with a sub-arctic climate typical of mid-latitude continental areas. It is expected that mining activities will be conducted on a year-round basis.

The topography of the project area is variable. Drumlins and lakes / wetlands dominate the northwest and southeast parts of the project area, respectively; and lowland lakes, rivers, and muskegs dominate the central part of the project area. The northwest part of the project area lies over portions of Patterson Lake and Forrest Lake, which are two of the largest waterbodies within 100 km of the Project. Elevations range from 583 metres above sea level (masl) on drumlins, to 480 masl in lowland lakes. The elevation of Patterson Lake is 499 masl.

The Project is covered by boreal forest common to the Canadian Shield. Bedrock outcrops are very rare, but are known to exist in areas of the eastern half of the project area.

1.4 Mineral Tenure, Surface Rights, Water Rights, Royalties, and Agreements

The Property consists of 32 contiguous mineral claims with a total area of 35,065 ha. All claims are 100% owned by NexGen.

Six of the 32 claims are subject to a 2% net smelter return (NSR) royalty payable to Advance Royalty Corporation (ARC), and a 10% production carried interest with Terra Ventures Inc. (Terra). The NSR may be reduced to 1% upon payment of $1.0 million to ARC. The Arrow Deposit is located outside of these six claims.

As of 06 December 2012, mineral dispositions are defined as electronic mineral claims parcels within the Mineral Administration Registry Saskatchewan (MARS) using a

Geographical Information System (GIS). MARS is a web-based, electronic tenure system used for issuing and administrating mineral permits, claims, and leases. Mineral claims are acquired via electronic map staking, and administration of the dispositions is also web-based.

As of the effective date of this report, all 32 mineral claims comprising the Rook I property are in good standing, and are all registered in the name of NexGen.

Surface rights are distinct from subsurface or mineral rights. The Project is located on provincial Crown land; as the owner, the Province of Saskatchewan can grant surface rights under the authority of the Forest Resources Management Act and the Provincial Lands Act. Granting surface rights for the purpose of accessing the land to extract minerals is done by issuing a mineral surface lease subject to the Crown Resource Land Regulations. Mineral surface leases have a 33-year maximum term which may be extended, as necessary.

NexGen does not currently hold surface rights of the project area. Surface rights are obtained after the ministerial review and approval of the Environmental Assessment (EA), and the successful negotiation of a mineral surface lease agreement with the Province of Saskatchewan.

RPA is not aware of any environmental liabilities to which the property is subject. 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 Rook I property.

1.5 Geology and Mineralization

The Rook I property is located along the southwestern rim of the Athabasca Basin, a large Paleoproterozoic-aged, flat-lying, intracontinental, fluvial, redbed sedimentary basin that covers much of northern Saskatchewan and part of northern Alberta. The Athabasca Basin is ovular at surface, with approximate dimensions of 450 km × 200 km. It reaches a maximum thickness of approximately 1,500 m near its centre.

The southwest portion of the Athabasca Basin is overlain by the flat-lying Phanerozoic stratigraphy of the Western Canada Sedimentary Basin, including the carbonate-rich rocks of the Lower to Middle Devonian Elk Point Group, Lower Cretaceous Manville Group sandstones and mudstones, moderately lithified diamictites, and Quaternary unconsolidated sediments.

South of the Athabasca Basin, where Athabasca sandstone cover becomes thin, paleo-valley fill and debris flow sandstones of the Devonian La Loche / Contact Rapids formation (Alberta) or Meadow Lake (Saskatchewan) formation unconformably overlie the basement rocks.

The Paleoproterozoic basement rocks of the Taltson Domain unconformably underlies the Athabasca Basin and the Phanerozoic stratigraphy within the extents of the Rook I property. The crystalline basement rocks comprise a spectrum of variably altered mafic to ultramafic, intermediate, and local alkaline rock types. The most abundant basement lithologies consist of gneissic, metasomatized-feldspar-rich granitoid rocks, and dioritic to quartz dioritic and quartz monzodioritic gneiss, with lesser granodioritic and tonalitic gneiss.

Mineralization occurs at the following seven locations on the property, and is exclusively hosted in basement lithologies below the unconformity that is overlain by the Athabasca Group.

- Arrow Deposit
- South Arrow Discovery
- Harpoon occurrence
- Bow occurrence
- Cannon occurrence
- Camp East occurrence
- Area A occurrence

Of the seven mineralized locations, the Arrow Deposit has undergone the most investigation.

The Arrow Deposit is currently interpreted as being hosted chiefly in variably altered porphyroblastic quartz-flooded quartz-feldspar-garnet-biotite (± graphite) gneiss. Mineralization at the Arrow Deposit is defined by an area comprised of several steeply dipping shears that have been labelled as the A0, A1, A2, A3, A4, and A5 shears. The A0 through A5 shears locally host high-grade (HG) uranium mineralization.

The Arrow Deposit is considered to be an example of a basement-hosted, vein type uranium deposit.

1.6 History

The Geological Survey of Canada in 1961 included the Rook I property as part of a larger area.

From 1968 to 1970, Wainoco Oil and Chemicals Ltd. completed airborne magnetic and radiometric surveys, and geochemical sampling programs. No structures or anomalies of interest were detected.

In 1974, Uranerz Exploration and Mining Ltd. completed geological mapping, prospecting, and lake sediment sampling around the property.

From 1976 to 1982, Canadian Occidental Petroleum Ltd. and other companies (e.g., Saskatchewan Mining and Development Corporation [SMDC, now Cameco]) completed airborne INPUT electromagnetic (EM) surveys. These surveys detected numerous conductors, many of which were subject to ground surveys prior to drilling.

Airborne magnetic-radiometric surveys were also completed and followed up on with prospecting, geological mapping, lake sediment surveys, and some soil and rock geochemical sampling. Few anomalies were found, other than those that were already located during the airborne and ground EM survey.

From 2005 to 2008, Titan Uranium Inc. (Titan) carried out airborne time-domain EM surveys using MEGATEM and Versatile Time Domain Electromagnetic (VTEM) systems, which detected numerous strong EM anomalies. A ground MaxMin II survey conducted in 2008 confirmed the airborne anomalies.

In 2012, pursuant to a mineral property acquisition agreement between Mega Uranium Ltd. (Mega) and Titan dated 01 February 2012, Mega acquired all nine dispositions comprising the Project. A gravity survey was completed over 60% of S-113921 through S-113933, which defined several regional features and some additional local smaller scale features. Simultaneously, Mega sampled organic-rich soils and prospected the same area. No soil geochemical anomalies or radioactive boulders were found.

In 2012, NexGen acquired Mega's interest in the Rook I property.

1.7 Exploration Status

Since acquiring the Rook I property in December 2012, NexGen has carried out exploration activities consisting of the following.

- Ground gravity surveys
- Ground direct current (DC) resistivity and induced polarization surveys
- Airborne magnetic-radiometric- very low frequency (VLF) survey
- Airborne VTEM survey
- Airborne Z-Axis Tipper electromagnetic (ZTEM) survey
- Airborne gravity survey
- Radon-in-water geochemical survey
- Ground radiometric and boulder prospecting program.

NexGen also conducted diamond drilling programs to test several targets on the Rook I property, which resulted in the discovery of the Arrow Deposit in drill hole AR-14-001 (formerly known as RK-14-21) in February 2014.

Mineralization at the Arrow Deposit is defined by an area comprising the A0 through A5 shears, which locally host HG uranium mineralization. The mineralized area is 315 m wide, with an overall strike of 980 m. Mineralization is noted to occur 100 m below surface, and it extends to a depth of 980 m. The individual shear zones vary in thickness from 2 m to 60 m. The Arrow Deposit is open in most directions and at depth.

Regional drilling completed by NexGen from 2015–2019 along the Patterson conductive corridor identified new uranium discoveries at the Harpoon, Bow, Cannon, Camp East, and Area A occurrences, and the South Arrow Discovery.

1.8 Exploration, Drilling, and Analytical Data Collection in Support of Mineral Resource Estimation

As of the effective date of this report, NexGen and its predecessors have drilled 754 holes totalling 380,051 m. From 2013 to the effective date of this report, NexGen has drilled 716 holes totaling 374,917 m.

Three types of drill core samples are collected at site for geochemical analysis and uranium assay.

- One-metre and 0.5-metre samples taken over intervals of elevated radioactivity, and one metre or two metres beyond radioactivity.
- Point samples taken at nominal spacings of five metres or 50 m for infill holes, which is meant to be representative of the interval or of a particular rock unit.
- Composite samples in the Devonian and Athabasca sandstone units where one-centimetre long pieces are taken and spaced throughout sample intervals ranging from one metre to 10 m long.

All samples are analyzed at Saskatchewan Research Council (SRC) Geoanalytical Laboratories by inductively coupled plasma optical emission spectroscopy (ICP-OES) or inductively coupled plasma mass spectroscopy (ICP-MS) for 64 elements, including uranium. Samples with low radioactivity are analyzed using ICP-MS. Samples with anomalous radioactivity are analyzed using ICP-OES.

NexGen personnel perform full core bulk density measurements using standard laboratory techniques. In mineralized zones, average bulk density is measured from samples at 2.5 m intervals, where possible (i.e., approximately 20% of all mineralized samples). In order for density to be correlated with uranium grades across the data set, each density sample directly correlates with a sample sent to SRC for assay.

Samples are also collected for clay mineral identification using infrared spectroscopy in areas of clay alteration. Samples are typically collected at five-metre intervals. and consist of centimetre-long pieces of core selected by a geologist.

Based on the data validation and the results of the standard, blank, and duplicate analyses, RPA believes that the assay and bulk density databases are of sufficient quality for Mineral Resource estimation at the Arrow Deposit.

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

In RPA's opinion, the drilling, core handling, logging, and sampling procedures meet or exceed industry standards, and are adequate for the purpose of Mineral Resource estimation.

1.9 Data Verification

RPA's data verification steps included site visits during which RPA personnel reviewed core handling, logging, sample preparation and analytical protocols, density measurement system, and storage procedures. RPA also reviewed the Leapfrog model parameters and geological interpretation, reviewed how drill hole collar locations are defined, inspected the use of directional drilling methods, observed the data management system, obtained a copy of the master database, and obtained SRC laboratory certificates for all drilling assays.

A review of the database indicated no significant issues. A separate review of the assay table determined minimal errors, and all are most likely due to rounding. Limitations were not placed on RPA's data verification process.

RPA considers the resource database reliable and appropriate to support a Mineral Resource estimate.

1.10 Metallurgical Test Work

NexGen conducted a metallurgical test program in 2018, which included a bench test program, a pilot plant, and paste backfill testing. Test work samples comprised three composite samples, consisting of low grade (LG), medium grade (MG), and high grade (HG) material, and ten samples of localized deposit areas.

Completed bench test work included the following.

- Quantitative evaluation of materials by scanning electron microscopy (QEMSCAN), potential acid generation
- SAGDesign™ and Bond ball mill index
- Batch leach
- Optimization leaching
- Confirmation and variability
- Settling
- Solvent extraction (SX)
- Separating funnel shakeout
- Stripping
- Gypsum precipitation
- YC precipitation
- Preliminary sulfide flotation
- Diagnostic gravity separation

Additionally, two pilot leaching tests were performed in 2018 using two different feed samples.

In 2019, a series of tests were carried out to advance the process design. These tests were carried out at the SRC facilities and included the following.

- Bench-scale testing to recover uranium from gypsum (June 2019).
- Trade-off study / test work of dewatering and washing technologies using belt filters (July 2019).
- Trade-off study / test work of dewatering and washing technologies using centrifuges (August 2019).

An advanced phase of the paste backfill testing program was conducted in 2019 using drill core samples from the pilot plant program. Geotechnical and geochemical evaluations were performed to validate the mine / mill design, and results will be used in for the Project's EA. Test work included investigating the following.

- Particle size distribution
- Whole rock analysis
- Mineralogy
- Static yield stress
- Rheology
- Transportable moisture limit
- Uniaxial compressive strength (UCS)
- Process water analysis
- Tailings and kinetic tests

The 2021 FS assumes a metallurgical steady state uranium recovery of 97.6%. This value was determined based on the results of pilot plant test work, and by compiling the performance of unit operation uranium recoveries. Pilot leach testing results indicated uranium extractions of 99.3%. The washing efficiency in the counter current decantation was greater than 99.6%. All other unit operations in the pilot testing had uranium recoveries of greater than 99.6%.

The QEMSCAN analysis identified that there were no primary molybdenum-bearing minerals present. However, molybdenum did occur in chalcopyrite and galena solid solutions. Similarly, there were no arsenic-bearing minerals identified.

1.11 Mineral Resource Estimation

The Mineral Resource estimate for the Project was based on results from 521 diamond drill holes. It was reported using a $50/lb U_3O_8 price, at a cut-off grade of 0.25% U_3O_8.

- Measured Mineral Resources total 2.18 million tonnes (Mt) at an average grade of 4.35% U_3O_8, for a total of 209.6 million pounds (Mlb) of U_3O_8.

- Indicated Mineral Resources total 1.57 Mt at an average grade of 1.36% U_3O_8, for a total of 47.1 Mlb U_3O_8.
- Inferred Mineral Resources total 4.40 Mt at an average grade of 0.83% U_3O_8, for a total of 80.7 Mlb U_3O_8.

The effective date of the Mineral Resource estimate is 19 July 2019. Estimated block model grades are based on chemical assays only. The Mineral Resources were estimated by NexGen and audited by RPA. Mineral Resources are inclusive of Mineral Reserves. RPA has noted that the deposit is open in many directions.

The Arrow Deposit Mineral Resource estimate is based on the results of surface diamond drilling campaigns conducted from 2014–2019. The Mineral Resources of the Arrow Deposit are classified as Measured, Indicated, and Inferred based on drill hole spacing and apparent continuity of mineralization, as summarized in Table 1-1.

Table 1-1: Mineral Resource Estimate – 19 July 2019

Classification	Zone	Tonnage (t)	Grade (% U_3O_8)	Contained Metal (lb U_3O_8)
Measured	A2-LG	920,000	0.79	16,000,000
	A2-HG	441,000	16.65	161,900,000
	A3-LG	821,000	1.75	31,700,000
Measured Total	–	2,183,000	4.35	209,600,000
Indicated	A2-LG	700,000	0.79	12,200,000
	A2-HG	56,000	9.92	12,300,000
	A3-LG	815,000	1.26	22,700,000
Indicated Total	–	1,572,000	1.36	47,100,000
Measured + Indicated	A2-LG	1,620,000	0.79	28,100,000
	A2-HG	497,000	15.90	174,200,000
	A3-LG	1,637,000	1.51	54,400,000
Measured + Indicated Total	–	3,754,000	3.10	256,700,000
Inferred	A1	1,557,000	0.69	23,700,000
	A2-LG	863,000	0.61	11,500,000
	A2-HG	3,000	10.95	600,000
	A3-LG	1,207,000	1.12	29,800,000
	A4	769,000	0.89	15,000,000
Inferred Total	–	4,399,000	0.83	80,700,000

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported at a cut-off grade of 0.25% U_3O_8.
3. Mineral Resources are estimated using a long-term uranium price of US$50/lb U_3O_8 and estimated mining costs.
4. A minimum thickness of one metre was used.
5. Tonnes are based on bulk density weighting.
6. Mineral Resources are inclusive of Mineral Reserves.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

8. Numbers may not sum due to rounding.
9. HG = High Grade, LG = Low Grade.

RPA has reviewed the geology, structure, and mineralization of the Arrow Deposit based on the results of 566 diamond drill holes. RPA has also audited three-dimensional (3D) wireframe models developed by NexGen, which represent 0.05% U_3O_8 grade envelopes with a minimum thickness of one metre.

Of the 566 holes completed, 45 drill holes were drilled on the South Arrow Discovery and were not used for the purposes of the Mineral Resource estimate. The wireframe models representing the Arrow Deposit mineralized zones are intersected in 418 of 566 drill holes. The updated 2019 Mineral Resource estimate does not account for HG domains within A3, which were accounted for in the previous 2017 Mineral Resource estimates. The A3-HG domains were found to be of relatively LG, with average grades just above the HG modelling threshold of 5% U_3O_8; after the 2019 infill drilling, the variability of grades was better handled with ordinary kriging (OK), where the locally varying mean, in conjunction with the density of data, counters grade smearing.

Based on 5,850 dry bulk density determinations for the Arrow Deposit, NexGen developed a formula that relates bulk density to grade. This formula was used to assign a density value to each assay. Bulk density values were then used to weight the grade estimation and convert volume to tonnage.

High grade values were capped, and their influence was further restricted during the block estimation process. High grade outliers were capped at 1%, 2%, 3%, 4%, 5%, 6%, 8%, 10%, 15%, 25%, and 30% U_3O_8, depending on the domain. This resulted in 428 capped assay values. No outlier assay values were identified in the HG domains. Therefore, no capping was applied to the assays as each HG domain dataset was determined to be stationary and appropriate for interpolation, with the exclusion of the A2-HG8, which was capped at 30% U_3O_8.

Variable density and grade multiplied by density (GxD) were interpolated using OK in the A2-HG domains (excluding A2-HG6 and A2-HG8), the A2-LG domain that envelopes a HG domain, and two large A3-LG domains (301 and 312). Inverse distance squared (ID^2) was used on all remaining mineralized domains. Estimates used a minimum of one to three composites per block estimate, to a maximum of 50 composites per block estimate. The majority of the domains used a maximum of two composites per drill hole.

Sample selection criteria were based on sensitivity testing that compared the estimated block means of each domain to the composited mean. Unsampled intervals and samples below the detection limit within the domains were assigned a grade of zero and considered to be internal dilution. Hard boundaries were used to limit the use of composites between domains. Block grade was derived by dividing the interpolated GxD value by the interpolated density value for each block.

The block model was validated by swath plots, volumetric comparison, visual inspection, and statistical comparison. The average block grade at zero cut-off was compared to the average of the composited assay data to ensure that there was no global bias.

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 other than what has been described in this report.

1.12 Mineral Reserve Estimation

The vertical extent of the Mineral Reserves extends from approximately 320 m below surface to 680 m below surface.

Based on the cut-off grade assessment, an incremental cut-off grade of 0.30% U_3O_8 was applied as the input parameter for designing stopes. This cut-off grade was applied at the level of stoping solids, after inclusion of waste and fill dilution. The Mineral Reserves are limited to the A2 and A3 veins within the Arrow Deposit.

A nominal amount of material between 0.03% U_3O_8 (the regulatory limit between benign waste and mineralized material) and 0.26% U_3O_8 (which is uneconomic to process) has been included in the mine plan, in addition to 88,100 tonnes of waste used to commission the mill and to keep the mill feed grade below 5.0%.

Stantec assumed that both transverse stope and longitudinal retreat stope mining methods would be used. The assumed mining rate is nominally 1,300 tonnes per day (t/d). A total planned dilution of approximately 24% is projected for the longhole stopes. The unplanned or overbreak dilution is estimated at 12% total.

Fill dilution will occur when mining next to fill walls and mucking on fill floors; a 4% fill dilution was applied to secondary transverse stopes only, and a 1% fill dilution was applied to secondary longitudinal stopes. Extraction (mining recovery) is estimated at a combined 95.5% for longhole mining and ore development.

The Mineral Reserve estimate is reported using the 2014 CIM Definition Standards. The effective date of the Mineral Reserve estimate is 21 January 2021. The Qualified Person (QP) for the estimate is Mr. Mark Hatton, P.Eng., an employee of Stantec. Table 1-2 summarizes Mineral Reserves based on a $50/lb uranium price at a cut-off grade of 0.30% U_3O_8.

Factors that may affect the Mineral Reserve estimate include the following.

- Commodity price assumptions.
- Changes in local interpretations of mineralization geometry and continuity of mineralization zones.
- Changes to geotechnical, hydrogeological, and metallurgical recovery assumptions.
- Input factors used to assess stope dilution.

- Assumptions that facilities such as the Underground Tailings Management Facility (UGTMF) can be permitted.
- Assumptions regarding social, permitting, and environmental conditions.
- Additional infill or step out drilling.

Table 1-2: Mineral Reserve Estimate

Classification	Recovered Ore Tonnes (thousands)	U_3O_8 Grade (%)	U_3O_8 lb (millions)
Proven	0	0	0
Probable	4,575	2.37%	239.6
Total	**4,575**	**2.37%**	**239.6**

Notes:
1. CIM definitions were followed for Mineral Reserves.
2. Mineral Reserves are reported with an effective date of 21 January 2021.
3. Mineral Reserves include transverse and longitudinal stopes, ore development, marginal ore, special waste, and a nominal amount of waste required for mill ramp-up and grade control.
4. Stopes were estimated at a cutoff grade of 0.30% U_3O_8.
5. Marginal ore is material between 0.26% U_3O_8 and 0.30% U_3O_8 that must be extracted to access mining areas.
6. Special waste in material between 0.03% and 0.26% U_3O_8 that must be extracted to access mining areas. 0.03% U_3O_8 is the limit for what is considered benign waste and material that must be treated and stockpiled in an engineered facility.
7. Mineral Reserves are estimated using a long-term metal price of US$50/lb U_3O_8, and a 0.75 US$/C$ exchange rate (C$1.00 = US$0.75). The cost to ship the YC product to a refinery is considered to be included in the metal price.
8. A minimum mining width of 3.0 m was applied for all longhole stopes.
9. Mineral Reserves are estimated using a combined underground (UG) mining recovery of 95.5% and total dilution (planned and unplanned) of 33.8%.
10. The density varies according to the U_3O_8 grade in the block model. Waste density is 2.464 t/m^3.
11. Numbers may not add due to rounding.

Stantec is not aware of any environmental, permitting, legal, title, taxation, socio-economic, political, or other relevant factors that could materially affect the Mineral Reserve estimate.

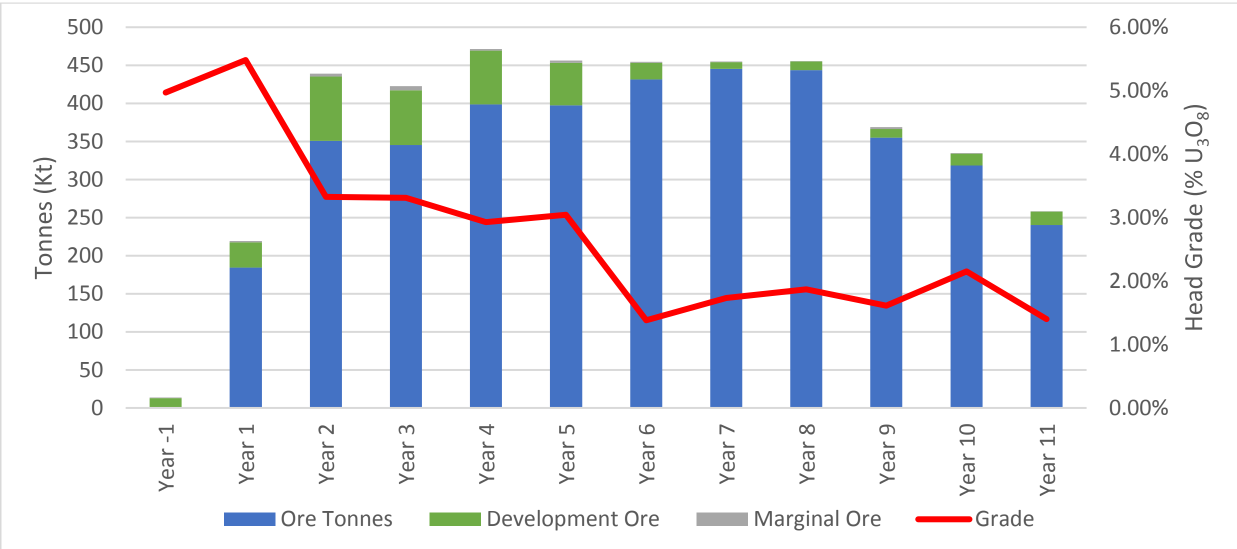
1.13 Mining Methods

Access to the underground (UG) Arrow Deposit will be via two shafts, an 8.0 m diameter Production Shaft (intake air) and a 5.5 m diameter Exhaust Shaft (second egress). Access to the working will be from the Production Shaft with stations on 500 and 590 Levels. Levels will be spaced 30 m apart UG and will be connected via an internal ramp.

Production will be via a conventional longhole mining. The longhole mining methods and mine design discussed in this section were chosen to optimize safety performance, reduce worker exposure to physical hazards and radiation, maximize Mineral Resource extraction, and increase operational flexibility and productivity by achieving simultaneous production from multiple mining fronts.

The estimated mill capacity is targeted at 1,300 tonnes per day (t/d) of ore. To realize this target, the mine plan will include longhole production on four separate mining blocks, with multiple stopes available per block. The estimated production rates of the stopes range from 250 t/d to 300 t/d. This will require approximately five stopes to be active to achieve 1,300 t/d, which will be feasible with that many stopes available. The grades will vary by mining block; this will facilitate the ability to provide a more consistent grade to the process plant with four active blocks. Production profile and head grade from UG are shown in Figure 1-1.

Figure 1-1: Underground Production Profile with Grade (U_3O_8)



The tailings produced by the mill will be returned UG as either cemented paste backfill for the production stopes or as cemented paste tailings into stopes that will be created for this purpose. The UGTMF will be located on the north side of the deposit and will consist of approximately 97 waste stopes and related development.

The mining method will make use of mechanized equipment and conventional processes widely employed in the global mining industry.

Shaft sinking will occur through a variety of stable and unstable strata, including water saturated overburden, Devonian Sandstone, Cretaceous Shales and Athabasca Sandstones, and finally into the basement rocks. These domains consist of poor to very poor-quality rock masses; however, once these have been temporarily artificially frozen for shaft construction, these are not anticipated to be problematic. A 600 mm hydrostatic lining is considered to be the minimum practicable thickness for lining against a freeze wall. As such, a 600 mm liner will be installed to 175 m in the Production Shaft and 217 m in the Exhaust Shaft. To prevent migration of water down the back of the liner and into the shaft, a grout seal will be placed at the base of the hydrostatic pressure resisting liner.

The minimum distance between the shallowest mine excavation and the unconformity is approximately 250 m. This drastically reduces the risks associated with the crown pillar and therefore has not been investigated in detail.

The processing of uranium ore will generate several forms of waste. A portion of the waste will be used for paste backfill. The remainder will be permanently stored in purpose-built excavations / chambers in the footwall (FW) of the deposit, in an area that is interpreted to have relatively minimal alteration or fault or shear structures. The 2021 FS proposes the UGTMF will consist of 97 waste stopes, each approximately 25 m wide by 25 m long by 60 m high. The excavations will be arranged in a regular pattern with a minimum of 15 m pillars between openings. The first waste stopes will be located on the 500 Level and the top of the excavations will be approximately 250 m below the unconformity.

Backfill of mined stopes is planned to use a combination of process waste, cement, potential fillers (such as fly ash), and water. The creation of paste tailings is directly proportional to the amount of material processed through the plant. For each tonne of processed material, 0.82 m^3 of paste tailings will be created, along with 0.32 m^3 of combined waste precipitates. Based on a steady-state production rate, the total fill produced will be nominally 373,100 m^3 per year for paste tailings, and 145,600 m^3 per year for combined precipitates. Tailings not used for paste backfill will be stored in the UGTMF.

The Arrow Deposit is planned to be accessed via two shafts. Both shafts will be located in the FW of the deposit. The first shaft will be used as a Production Shaft, and for transportation of personnel and materials into the mine and will be sunk to a depth of 650 m below surface. The Production Shaft will have divided compartments so that fresh air that comes into contact with ore being skipped to surface will be immediately exhausted within the mine. The Production Shaft will have a permanent headframe and hoisting house. The second shaft will be used as an exhaust ventilation shaft. The Exhaust Shaft will be sunk to a depth of 533 m below surface and will be equipped with a secondary emergency escapeway system.

Thirteen levels, spaced at 30 m intervals sill to sill, are planned for the Arrow Deposit. Lateral development will be concentrated in the first four years to establish the production areas, the UGTMF areas, UG infrastructure and the permanent ventilation system. In addition to the lateral development, there will be an internal ramp system that will connect all mining levels.

Mine dewatering will be completed using a clean water system on the 500 Level. The 500 Level sumps will be capable of collecting and removing all strata and operational process water from the mine infrastructure, ongoing development, operational stopes, shaft inflow, and pastefill seepage. Run-of-mine water will decant through membranes; the clean water will be pumped to surface while the residual solids and water will be collected and placed into the ore handling system.

Transverse stope mining will be used in areas of wider stopes (generally greater than 12 m), while longitudinal retreat stope mining will be used in areas of thinner stope widths. Transverse longhole mining will be completed using primary and secondary stoping sequences to avoid leaving pillars. The order in which stopes are extracted will be largely driven by the head grade, with the overarching goal of processing 30 Mlb of U_3O_8 annually. Primary stopes will be recovered first, followed by primary stopes on two vertical levels above, and then secondary stopes on the original level.

Two separate vertical mining blocks (the Upper Block and Lower Block) will be established, and within each vertical block, the A2 and A3 veins can be mined independently. Mining activities will commence from both the Upper Block and Lower Block, and in the A2 and A3 veins, for a total of four separate production areas. A fifth production block will be created below the 620 Level.

The ore handling system will begin with load-haul-dump (LHD) units loading muck in transverse and longitudinal retreat stopes. The LHDs will tram muck to centrally located ore and waste passes. The bottom of the ore pass will be located on 590 Level, where a control system will direct ore on to a grizzly equipped with a remotely operated rock breaker. The grizzly openings will be 400 mm by 450 mm. The sized ore will be loaded onto a conveyor on the 620 Level and hauled to the shaft for skip loading.

There will be two separate waste handling systems. The waste from the UGTMF will report to a rockbreaker on the 500 Level, near the Production Shaft. The sized waste rock will be loaded onto the 620 Level conveyor and hauled to the shaft for skip loading. The second waste handling system will be located near the ore body and will handle all remaining lateral development. The system will be identical to the ore handing system.

The ventilation system is designed as a predominately negative or “pull” system. Fresh air will be distributed throughout the mine from the 500 and 590 Level shaft stations from the Production Shaft and internal ramp. The auxiliary ventilation system will utilize both flow-through and extraction ventilation to exhaust contaminated air from localized areas to return air drifts and raises.

The Rook I mine will be developed using a high degree of equipment mechanization. Each of the main pieces of equipment will have remote operating capability, and in some cases will be autonomous to reduce radiation exposure. A raisebore machine will be used for development of ore and waste passes, and internal ventilation raises.

The mobile equipment UG will be captive in the mine. The maintenance facility will be equipped to repair and service all captive equipment for the life of the operation.

1.14 Recovery Methods

The process plant design developed by Wood for the Project is based on the metallurgical testing and on the latest unit processes successfully used in uranium process plants across the world, including plants in northern Saskatchewan. The design of tailings preparation has been improved to facilitate a more reliable tailings deposition strategy through the paste plant. The process plant will consist of the following.

- Ore sorting
- Grinding
- Leaching
- Liquid-solid separation via counter current decantation and clarification
- SX
- Gypsum precipitation and washing
- YC precipitation and washing
- YC drying, calcining and packaging
- Tailings preparation and paste tailings plant
- Effluent treatment

Plant throughput will be 1,300 t/d and design production will be 30 Mlb U_3O_8 per annum. It is expected that a 3-month ramp-up period will be required to reach design throughput.

Water from the settling pond and fresh water from Patterson Lake will be fed to the process plant to provide the process requirements. The amount of water recycled from the settling pond has been further optimized to reduce the amount of fresh water required by using settling pond water for counter current decantation (CCD) wash water and using belt filter filtrate for paste process water.

The major reagents required will include sulphur, sulphuric acid, unslaked lime, hydrogen peroxide, flocculant, kerosene, tertiary amine, isodecanol, sodium carbonate, magnesia, barium chloride and ferric sulphate.

The process plant will require approximately 7.4 megawatts (MW) of power to operate at full capacity. The paste plant will require approximately 0.9 MW of power.

1.15 Project Infrastructure

The key infrastructure contemplated for the Project includes the following.

- UG mine with two vertical shafts.
- UG infrastructure, including material handling systems, maintenance facilities, fuel bay, explosives magazine, ventilation, paste backfill and paste tailings distribution system, electrical and communications facilities, UG water supply, dewatering facilities.
- UGTMF.
- Surface support infrastructure for the mine, including headframe and hoist facilities, surface explosives magazine, and ventilation fans.
- Surface support infrastructure for the mill, including process plant, SX plant, effluent treatment plant, and acid plant.
- Site support infrastructure, including accommodation camp, Liquefied Natural Gas (LNG) facilities, LNG power plant, mine and mill dry facilities, analytical and metallurgical laboratory and maintenance, warehouse and security buildings.
- Surface ore storage stockpile facility.

- Waste rock storage facilities for potentially acid generating (PAG), non-potentially acid generating (NPAG) and special waste materials.
- Water management facilities, including: two site water runoff ponds, six contact water process ponds, a PAG stockpile runoff collection pond, and conveyance and diversion structures.
- Domestic / industrial waste management areas.
- Airstrip.
- LNG power plant.

From a study completed during the prefeasibility study (PFS), it was determined that the NexGen Rook I site would be powered by an on-site generation plant due to a lack of existing power infrastructure and a high cost for the installation of a new transmission line. An LNG power plant was progressed during this FS with a power requirement of 26.5 MW based on a nominal demand of 24.1 MW. An N+1 design is planned, with eight generators operating at 3.3 MW and one standby unit. The plant design includes LNG storage and filling facilities with the fuel being trucked to the site.

1.16 Environmental, Permitting and Social Considerations

1.16.1 Ore and Special Waste Stockpiles

There will be an ore stockpile consisting of four piles of differing grades. Each pile will be approximately 6,500 m^3.

It is estimated that about 1% of the waste rock brought to surface will be mineralized but will not contain high enough grade to be processed through the mill economically, and therefore is not stockpiled in the ore stockpile area. This material is stored in the special waste rock stockpile area with an anticipated pile volume of 60,000 m^3. The special waste will be processed during normal operations, to ensure the mill head grade remains below the 5% U_3O_8 design limit. The remaining special waste will be processed at end of mine life, with the resultant tailing being deposited UG in the UGTMF chambers.

Both the ore and special waste stockpiles will be dual lined with high-density polyethylene (HDPE) and will be self-contained facilities capable of holding a full probable maximum precipitation (PMP) 24-hour event.

1.16.2 Environmental Studies

NexGen commenced collection of baseline data in 2015, with the majority of field studies commencing in 2018. Where necessary, some studies continued into 2019 and 2020 to complete the baseline data and information collection requirements, with some work ongoing into 2021. At the time of this report, NexGen has undertaken sufficient baseline data collection to complete a comprehensive EA.

1.16.3 Waste Rock Management Facility

Approximately 5.9 Mm^3 of waste rock will be generated over the course of the LOM. Of this total, 4.6 Mm^3 (78%) is PAG and 1.3 Mm^3 is NPAG. The PAG and NPAG waste rock will have separate storage areas. The PAG and NPAG waste rock will be stockpiled with 2H:1V side slopes and the top of the finished stockpile will tie into the hill to the south; the overall height will not exceed the highest nearby topography. The PAG storage area will be HDPE lined and the NPAG storage area will not be lined.

1.16.4 Water Management

The water management infrastructure has been designed to maximize the diversion of non-contact surface runoff water away from the general site footprint and developed features. Precipitation events and snow melt runoff that come in contact with disturbed infrastructure areas, or potential contact zones, are captured, collected, and directed to respective impound areas identified as site runoff ponds or collection areas.

All ponds and pads containing mineralized or radiologically contaminated material have been designed to accommodate a PMP 24-hour event. These areas are self-contained in that the initial precipitation events are contained within the feature itself. The initial precipitation event does not exit elsewhere until pumped. These contained waters are tested before release to the environment based on regulatory requirement; water that does not meet specification will report to the effluent treatment plant for treatment.

The capture zones for Site Runoff Pond #1 have potential contact with mineralized or radiologically contaminated material. Site Runoff Pond #1 is designed to capture a PMP 24-hour event. Draw down is by sump pump to the site settling pond.

Site Runoff Pond #2 is designed to capture a 1:100 year 24-hour precipitation event. The pond contents will be tested, and if suitable for release, will be released to environment. If tested and not suitable for release, pond contents will be pumped to the site settling pond. In the case of a PMP 24-hour precipitation event, Site Runoff Pond #2 will capture and collect runoff to full capacity of the pond, prior to overflowing additional precipitation to the west bermed runoff collection area.

Six contact water storage ponds are planned, including four fill-test-release monitoring ponds for treated effluent, one contingency pond, and one feed settling pond. Each monitoring pond and the contingency pond is sized for 5,000 m^3 of capacity and will maintain 1 m of freeboard as contingency for a PMP 24-hour event. The feed settling pond will have a capacity of 16,000 m^3 with 1 m freeboard. Approximately 1,100 m^3 of the settling pond capacity is reserved for a 1:100 year 24-hour precipitation event which includes runoff collecting immediately surrounding the Production Shaft and in the pipe containment corridor.

All other water conveyance and containment structures have been designed to accommodate a 1:100 year 24-hour precipitation event as well as the anticipated volumes of water generated under routine and non-routine operating conditions.

1.16.5 Closure and Reclamation Planning

Following the completion of mining and milling activities, a detailed decommissioning plan will be developed in accordance with Provincial and Federal regulations and guidelines. Once finalized, the plan and an application for approval to decommission will be submitted to Provincial and Federal authorities. Following approval, decommissioning activities will commence.

Decommissioning will be preceded by the orderly cessation of operations and transition of the operation into a safe inactive state. Production mining will be completed, and active mining areas backfilled and secured. The mill processing circuits will be systematically shut down, flushed, and cleaned. Surface facilities, infrastructure, and equipment will be cleaned, as necessary, scanned, and prepared for decommissioning.

Wherever practicable, surface and UG infrastructure, equipment, and materials not required during the decommissioning phase and which meet radiological criteria for off-site removal will be salvaged, sold, or transferred off-site for recycling or disposal. Remaining infrastructure, equipment and materials will undergo final decommissioning on-site.

1.16.6 Permitting

There are several federal and provincial regulatory approvals required for a new uranium mine and mill development. Federally, under the authority of the Nuclear Safety Control Act (NSCA), proponents wishing to carry out uranium mining and milling must first obtain a licence from the federal nuclear regulator, the Canadian Nuclear Safety Commission (CNSC). The CNSC licensing process is in progress. Before the CNSC can make a licensing decision, proponents are required to undergo an EA of the proposed project. As the Rook I Project falls under both federal and provincial jurisdictions for EA, each of the CNSC and the Saskatchewan Ministry of Environment (ENV) – Environmental Assessment Branch (EA Branch) will require an EA prior to project approval. The EA process for the Rook I Project is in progress as of the effective date of this report, and preparation of a Draft EIS is underway.

As development of the Draft EIS and licensing applications are in progress, any findings, including any notable issues that could materially impact NexGen's ability to extract the Mineral Resources, are not yet available for inclusion in the Technical Report. Furthermore, no recommendations from the EA or licensing processes for future monitoring and/or management of environmental and social aspects of the Rook I Project have been determined. Therefore, any consideration regarding specific monitoring and management plans are not included in the Technical Rreport.

1.16.7 Social or Community Impacts

NexGen has engaged regularly and established relationships with local communities and Indigenous groups since 2013. Community and Indigenous engagement have evolved since the submission of the 2018 TechnicalRreport. Engagement mechanisms have included notification letters, meetings with leadership, establishing joint working groups (JWGs) for detailed discussions, and providing funding for traditional land use studies. The engagement process will continue throughout the EA and licensing processes.

In the second half of 2019, NexGen entered into Study Agreements (Agreements) with the following four Indigenous groups.

- Clearwater River Dene Nation
- Métis Nation – Saskatchewan (MN-S), including as on behalf of the Locals of MN-S Northern Region II
- Birch Narrows Dene Nation
- Buffalo River Dene Nation

The Agreements provide a framework for working collaboratively to advance the EA and exchange information that will be used to inform the Crown as the Crown undertakes its duty to consult.

The Agreements provide funding to each Indigenous group and outline a collaborative process for formal engagement to support the inclusion of Indigenous knowledge in the EA. The Agreements also outline processes for identifying potential effects to Indigenous rights, treaty rights, and socio-economic interests, and avoidance and accommodation measures in relation to the Project.

1.17 Markets and Contracts

Marketing studies and commodity price assumptions are based on research and forecasts by UxC LLC (UxC).

NexGen is considering selling production from the Project through all avenues of selling uranium including long-term contracts that would be entered into with buyers. It is expected that any such contracts would be within industry norms for such uranium contracts. Contracts have currently not been entered into for the Project.

The financial analysis assumes that 100% of uranium produced from the planned Rook I Project can be sold at long-term price of US$50/lb U_3O_8, using an exchange rate of C$1.00 = US$0.75, which includes the cost to ship the YC product to the final processing site.

1.18 Capital Cost Estimates

The estimate meets the classification standard for a Class 3 estimate as defined by Association for the Advancement of Cost Engineering (AACE) International and has an intended accuracy of ±15%. The estimate is reported in Q4 2020 Canadian dollars. Table 1-3 outlines the estimated capital cost for supplying, constructing, and pre-commissioning the Project, and is inclusive of the early works activities.

Mining capital costs primarily comprise the following areas: shaft sinking, lateral mine development, and stationary mine infrastructure. Mine mobile equipment is assumed to be purchased on a lease-to-own basis, with the costs incurred in the lease payments. Process plant costs include the construction of the entirety of the process plant facility. Infrastructure costs include provision for the LNG power plant, as well as site preparation, permanent camp, maintenance shop, fuel storage, administration and dry facility, water treatment systems, airstrip, and site roads. Indirect costs include temporary construction facilities, construction services and supplies, and construction management (CM) costs, construction equipment, freight, Owner's costs, and contingency.

NexGen is preparing a pre-commitment early works program that will encompass all scheduled activities planned for Year -4 Month 1 through Month 6. This plan will advance certain elements of the overall scope and mitigate project risks. The program includes work and the associated costs that NexGen intends on expending prior to an FID.

The scope of the pre-commitment early works program includes the following (at a high level).

- Clearing and grubbing.
- Site levelling and road construction.
- Batch plant construction.
- Initial camp construction.
- Shaft-sinking preparations, including freeze hole drilling, freeze plant installation, and sinking plant installations).

Stantec estimates the pre-commitment early works program will cost approximately $157.9 million.

Table 1-3: Total Capital Cost Estimate

Description	Units	Cost
Pre-commitment early works	***$ million***	***157.9***
Project Capital		
UG Mining	$ million	240.0
Processing	$ million	216.4
Site Development	$ million	27.7

Description	Units	Cost
On-Site / Off-Site Infrastructure	$ million	118.9
Subtotal Project Direct Costs	***$ million***	***602.9***
Project indirect costs	$ million	326.5
Project Owner's Costs	$ million	97.9
Subtotal Project Direct and Indirect Costs	***$ million***	***1,027.2***
Project Contingency	$ million	114.8
Total Project Capital	**$ million**	1142.0
Pre-production Capital Cost (Pre-Commitment & Project)	***$ million***	***1,299.9***
Sustaining	$ million	362.4
Closure	$ million	69.5
Total	**$ million**	**1,731.8**

Note:
1. Pre-commitment capital costs include contingency.
2. Totals may not sum due to rounding.

Sustaining capital incorporates all capital expenditures after the pre-production period of Year –4, Year -3, Year -2, and Year -1. Reclamation costs of $78.6 million have been included in Years 12 through Year 16, less $9.1 million in salvage value.

1.19 Operating Cost Estimates

Operating cost estimates were developed to present annual costs for production. Unit costs are expressed as $/tonne processed and $/lb U_3O_8. Operating costs were allocated to either mining, process, tailings facility and paste plant, or general and administration (G&A). LOM operating costs are estimated to be $1,769.8 million. LOM operating costs are summarized in Table 1-4.

UG mining occurs during Year -2 to Year 11 (note in Year -2 and Year -1, UG mining costs are capitalized). UG mining begins with capital development in Year -2 and the capitalized development continues through the LOM.

Table 1-4: Operating Cost Estimate Summary (Year 1 to Year 11 inclusive)

Description	LOM Cost ($ million)	Average Annual ($ million)	Unit Cost ($/t processed)	Unit Cost ($/lb U_3O_8)
Mining	691.3	64.6	151.09	2.96
Processing	647.0	60.5	141.41	2.77
Tailing Facility and Paste Plant	144.0	13.5	31.46	0.62
General and administration	287.5	26.9	62.84	1.23
Total	**1,769.8**	**165.4**	**386.80**	**7.58**

Notes:
1. Totals may not sum due to rounding.
2. Average annual cost based on 10.7 years

G&A costs include labour, camp and catering costs, flights to and from site, insurance premiums, general maintenance of the surface buildings, and marketing and accounting functions. Allowances were included for reimbursable fees paid to the CNSC.

1.20 Economic Analysis

The results of the economic analysis represent forward-looking information that is subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in this report include, but are not limited to, statements with respect to future uranium prices, estimation of Mineral Resources and Mineral Reserves, estimated mine production and uranium recovered, estimated capital and operating costs, and estimated cash flows generated from the planned mine production. Actual results may be affected by the following.

- Differences in estimated initial capital costs and development time from what has been assumed in the 2021 FS.
- Unexpected variations in quantity of ore, grade, or recovery rates, or presence of deleterious elements that would affect the process plant or waste disposal.
- Unexpected geotechnical and hydrogeological conditions from what was assumed in the mine designs, including water management during construction, mine operations, and post mine closure.
- Differences in the timing and quantity of estimated future uranium production, costs of future uranium production, sustaining capital requirements, future operating costs, assumed currency exchange rate, requirements for additional capital, unexpected failure of plant, or equipment or processes not operating as anticipated.
- Changes in government regulation of mining operations, environment, and taxes.
- Unexpected social risks, higher closure costs and unanticipated closure requirements, mineral title disputes or delays to obtaining surface access to the property.

If additional mining, technical, and engineering studies are conducted, these may alter the project assumptions presented in this report and may result in changes to the calendar timelines and the information and statements contained in this report.

Development and licensing approvals are not currently in place, and statutory permits, including environmental permits, are required to be granted prior to mine commencement.

The Project has been evaluated using discounted cash flow analysis. Cash inflows consist of annual revenue projections. Cash outflows consist of project capital expenditures, sustaining capital costs, operating costs, taxes, royalties, and commitments to other stakeholders. These are subtracted from revenues to arrive at the annual cash projections.

Cash flows are taken to occur at the mid point of each period. To reflect the time value of money, annual cash flow projections are discounted to the Project valuation date using the yearly discount rate. The discount rate appropriate to a specific project can depend on many factors, including the type of commodity, the cost of capital to the project, and the level of project risks (e.g., market risk, environmental risk, technical risk, and political risk) in comparison to the expected return from the equity and money markets.

The base case discount rate for the 2021 FS is 8%. The discounted present values of the cash flows are summed to arrive at the Project's NPV. In addition to the NPV, the IRR and the payback period are also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. The payback period is calculated as the time required to achieve positive cumulative cash flow for the Project from the start of production.

Taxes and depreciation for the Project were modelled based on input from NexGen, as well as a review of the Guideline: Uranium Royalty System, Government of Saskatchewan, June 2014. In addition, NexGen has opening balances of Canadian Exploration Expense (CEE) and operating losses that were applied in the tax model.

On a pre-tax basis, the NPV at 8% is $5,577.0 million, the IRR is 64.9%, and the assumed payback period is 0.8 years. On a post-tax basis, the NPV at 8% is $3,465.0 million, the IRR is 52.4% and the assumed payback period is 0.9 years.

A summary of the LOM cashflow is provided in Table 1-5 and Figure 1-2. Table 1-6 summarizes the economic results of the 2021 FS, with the NPV at 8% base case highlighted.

Table 1-5: LOM Cashflow Forecast Summary Table

Description	Units	Value
Gross revenue	$ million	15,573.2
Less: transportation	$ million	0
NSR	$ million	15,573.2
Less: provincial revenue royalties	$ million	(1,129.1)
Net revenue	$ million	14,444.1
Less: total operating costs	$ million	(1,769.8)
Operating cash flow	$ million	12,674.3
Less: capital costs	$ million	(1,573.9)
Pre-tax cash flow	$ million	11,100.4
Less: provincial profit royalties	$ million	(1,683.5)
Less: taxes	$ million	(2,404.5)
Post-tax cash flow	***$ million***	**7,012.4**

Figure 1-2: Undiscounted After-Tax Cash Flow

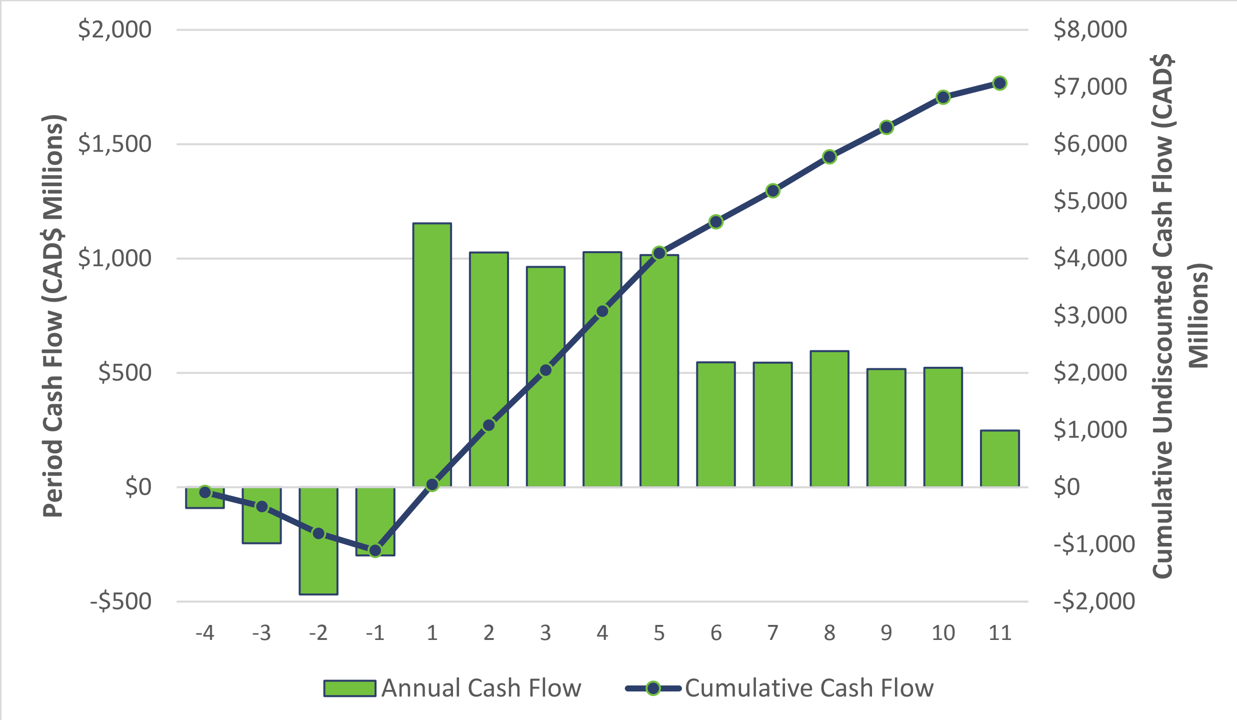


Table 1-6: 2021 Feasibility Study Forecast Economic Results

Description	Units	Value
Pre-Tax		
NPV at 8%	$ million	5,577
NPV at 10%	$ million	4,745
NPV at 12%	$ million	4,051
Internal rate of return	%	64.9%
Payback period	Years	0.8
After-Tax		
NPV at 8%	$ million	3,465
NPV at 10%	$ million	2,930
NPV at 12%	$ million	2,484
Internal rate of return	%	52.4%
Payback period	Years	0.9

Note: Payback period is calculated from the start of production

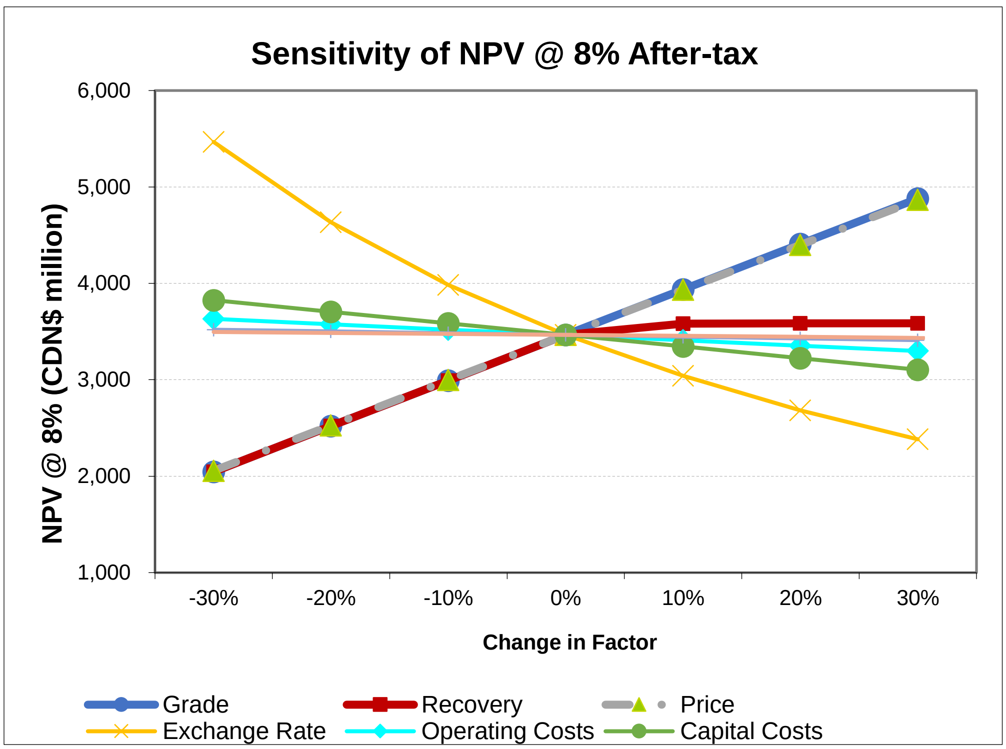
1.21 Sensitivity Analysis

The cash flow model was tested for sensitivity to variances regarding the following.

- Head grade
- Process recovery
- Uranium price
- Overall operating costs
- Overall capital costs
- Labour costs
- Reagent costs
- CAD to USD exchange rate

Figure 1-3 illustrates the results of the sensitivity analysis. The anticipated Project cash flow is most sensitive to fluctuations in the price of uranium, head grade, and process recovery. YC is primarily traded in US dollars, whereas capital and operating costs for the Project are primarily priced in Canadian dollars. Therefore, the CAD to USD exchange rate may significantly influence project economics.

Figure 1-3: Sensitivity Analysis



1.22 Risks and Opportunities

NexGen and its lead consultants have assessed critical areas of the Project and identified risks associated with the technical and cost assumptions used. The main risks identified in the Project include: assumptions around the prevalence of mineralized material in areas designated for mine infrastructure, assumptions around ground freezing and overall shaft development, adverse ground conditions as they relate to planned mining excavations, material handling systems unable to meet planned and peak production, commissioning of the UGTMF being slower than anticipated resulting in delays to first production, regulatory risks around permitting, and stakeholder engagement, and risks around cost escalation and project execution.

NexGen and its lead consultants performed an opportunities analysis. Opportunities that were recognized included: a potential expansion of Mineral Resources, and corresponding extension of the mine operating life, improvements to the mine extraction factor, reduction in mining operating costs and improved safety by considering remote or autonomous mining equipment, reductions in mining and process water usage through recycling, finalize the site water management philosophy and optimize the required infrastructure, consider heat recovery opportunities from the acid plant and power plant, evaluate alternative energy options including renewables and connecting to a provincial grid, and advancing critical early works construction packages to streamline overall project execution.

1.23 Interpretation and Conclusions

Under the assumptions presented in this report, the Project indicates positive economics. The anticipated Project cash flow is most sensitive to the price of uranium, head grade, and process recovery. The Canadian dollar to United States dollar exchange rate significantly influences Project economics.

1.24 Recommendations

Due to the positive, robust economics, it is recommended to advance the Rook I Project to the next phase of engineering. The recommended development path is to continue to advance the environmental assessment and licensing efforts while concurrently advancing key activities that will provide further project definition and reduce project execution timeline risks. Associated project risks are manageable, and identified opportunities can provide enhanced economic value.

Engineering and field investigations should be advanced in support increased certainty of costs and project timelines in preparation for permit approvals and a FID.

This following subsections list the programs that are recommended for the next stage of engineering work for the Rook I Project.

1.24.1 Engineering

It is recommended that NexGen proceeds to Basic Engineering. Basic engineering design forms the basis for later successful completion of the detailed engineering, procurement, construction, and commissioning work, and further provides NexGen valuable information to finalize internal discussion and evaluation of the feasibility of the Project.

The target for basic engineering to create a Class 2 Estimate along with the related Level 4 Schedule.

The total estimated cost for basic engineering is $30–35 million.

1.24.2 Site Investigations

It is recommended that NexGen proceeds with site investigations to support Basic Engineering, including, but not limited to the following.

- Detailed materials characteristics and quantification assessment to confirm borrow source locations and available volumes of aggregates.
- Drill hole investigations of nuisance mineralization observed in the footwall of Arrow proximal to LOM infrastructure, the quartz vein observed in GAR-18-013 (Exhaust Shaft pilot hole), and the northern extents of the UGTMF.
- Hydrogeological studies to increase NexGen's understanding of the impact of groundwater on the UG mine and mine dewatering requirements.
- Investigate near surface and subsurface conditions in the area of proposed surface infrastructure, focusing on the Mine Terrace and Waste Rock Storage Facility.

The total estimated project cost for the geotechnical, geomechanical, hydrological and surface material assessment is $8–9 million.

1.24.3 Process Plant Optimizations

The following studies are proposed.

- Loaded strip acid recovery
- Gypsum belt filter optimization
- YC particle size enhancement
- YC belt filter optimization
- Clarifier optimization
- Paste plant optimization
- Geo-metallurgical characterization
- Mine water pre-treatment technology

The total estimated cost for this program is $1.0–1.5 million.

2.0 INTRODUCTION

The following consultants were retained by NexGen Energy Ltd. (NexGen) to complete a technical report regarding the results of 2021 NexGen Rook I Project (the Project) Feasibility Study for the Arrow uranium deposit in Saskatchewan, Canada.

- Stantec Consulting Ltd. (Stantec)
- Wood Canada Ltd. (Wood)
- RPA, now a part of SLR Consulting Ltd. (SLR)
- Golder Associates (Golder)

2.1 Terms of Reference

This report is prepared as an NI 43-101 Technical Report for NexGen by Stantec, Wood, RPA and Golder, and will be filed with the Canadian Securities Administrators (CSA) on the System for Electronic Document Analysis and Retrieval (SEDAR) filing system. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort based on.

- Information available at the time of preparation
- Data supplied by outside sources
- The assumptions, conditions, and qualifications set forth in this report.

This technical report is written in Canadian English and it uses SI units to express measurements, unless otherwise indicated. Monetary units are expressed in Canadian dollars (CAD), with the exception of uranium prices, which are expressed in United States dollars (USD).

2.2 Qualified Persons

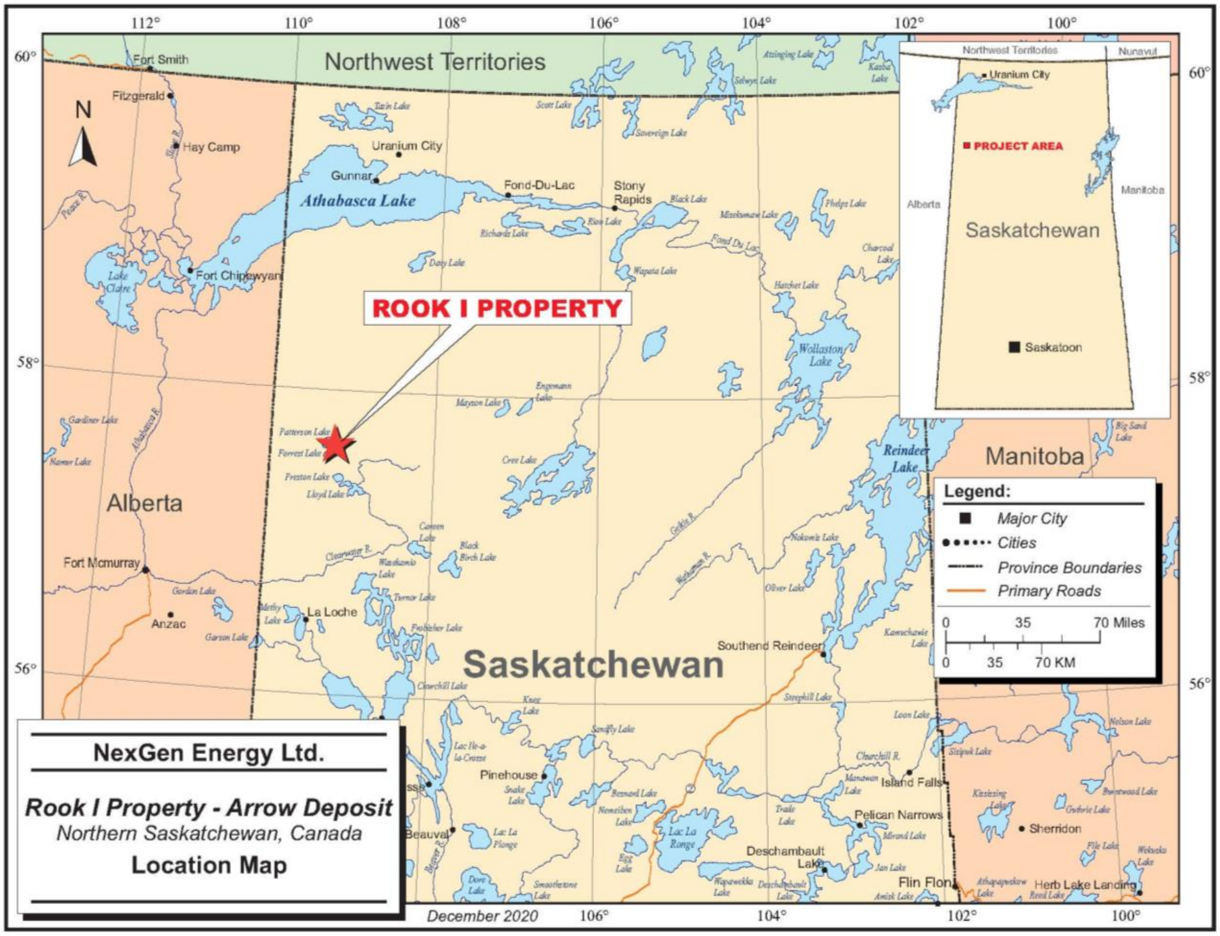
The following persons serve as QPs, as defined by NI 43-101 standards and regulations.

- Mr. Mark Hatton, P.Eng., Project Manager, Stantec
- Mr. Paul O'Hara, P.Eng., Manager Process, Wood
- Mr. Mark Mathisen, C.P.G., Principal Geologist, RPA
- Mr. Dan Walker, Ph.D., P.Eng., Senior Hydrotechnical / Water Resources Engineer, Golder

2.3 Site Visits and Scope of Personal Inspection

Figure 2-1 shows a map that indicates where the Rook I Project property and site are located.

Figure 2-1: Location Plan



Note: Figure courtesy NexGen, 2018.

Mr. Mark Hatton (Stantec), Mr. Paul O'Hara (Wood), and Mr. David A. Ross, M.Sc., P.Geo, (formerly Principal Geologist with RPA) visited the property on 12 June 2019. The group toured the project site and viewed the area proposed for the plant site and shaft collars. They reviewed core handling, logging, sample preparation, and storage procedures followed at site. They also analyzed the sample analysis protocols and density measurement system used at site.

Mr. Paul O'Hara visited the property on 16 May 2018. Mr. Paul O'Hara toured the project site and viewed the area proposed for the plant site and shaft collars. During the visit, Mr. Paul O'Hara inspected the camp, visited the core storage area, and reviewed the core handling procedures followed at site.

Mr. Mark Mathisen visited the property from 19–20 June 2016, and from 22–25 July 2017. During the visits, Mr. Mark Mathisen visited operating drill sites, reviewed quality assurance and quality control (QA/QC) logging procedures, and viewed selected drill core samples.

2.4 Effective Dates

The following effective dates apply to information discussed throughout this report.

- The effective date of the Mineral Resource estimate reported in Section 14.0 is 19 July 2019; diamond drill results from NexGen's Winter 2019 campaign have been incorporated into this report
- The effective date of the Mineral Reserve estimate is 21 January 2021
- The effective date of the final capital and operating cost estimates is 21 January 2021
- The effective date of the financial analysis is 21 January 2021
- The effective date of the NexGen news release is 22 February 2021

The overall effective date of this report is 22 February 2021.

2.5 Information Sources and References

The key references for this technical report are the following.

- Wood and RPA, 2018: NexGen Energy Rook I Project, Pre-feasibility Study: report prepared for NexGen Energy, 05 November 2018, 454 p.
- Measured and Indicated Resource Block Model (arw_4x4x4_id2_ok_2019Q3_rev3), issued 02 October 2019.
- North Rock Engineering, February 2021 Arrow Deposit – Basement Mining Geotechnical Assessment, prepared and dated February 2021.
- Newmans Geotechnique, December 2019: 18113041-R-001 NexGen Arrow Feasibility Study.
- Golder Associates, March 2020: 20200331_nexgen arrow deposit – production and Exhaust Shaft feasibility design report rev1.

RPA, now a part of SLR, was retained by NexGen to carry out an audit of the current Mineral Resource estimate for the Arrow Deposit, and subsequently prepare Sections 4.0–12.0, 14.0, and 23.0 for NexGen's use in a feasibility study and an independent technical report regarding NexGen's Rook I property in Saskatchewan, Canada.

Reports and documents listed in Section 2.7, Section 3.0, and Section 27.0 were also used to support the preparation of this report.

Additional information was sought from NexGen personnel where required.

2.6 Previous Technical Reports

NexGen has previously filed the following technical reports for the Project.

- McNutt, A.J., 2014: Technical Report on the Rook I Property Saskatchewan, Canada: technical report prepared for NexGen Energy Ltd., effective date 28 February 2014
- McNutt, A.J., 2015a: Technical Report on the Rook I Property Saskatchewan, Canada: technical report prepared for NexGen Energy Ltd., effective date 15 May 2015
- McNutt, A.J., 2015b: Technical Report on the Rook I Property Saskatchewan, Canada: technical report prepared for NexGen Energy Ltd., effective date 18 November 2015
- McNutt, A.J., 2015c: Technical Report on the Rook I Property Saskatchewan, Canada: technical report prepared for NexGen Energy Ltd., effective date 30 November 2015
- Mathisen, M.B. and Ross, D.A., 2016: Technical Report on the Rook I Property Saskatchewan, Canada: technical report prepared by Roscoe Postle Associates for NexGen Energy Ltd., effective date 13 April 2016
- Mathisen, M.B. and Ross, D.A., 2017: Technical Report on the Rook I Property Saskatchewan, Canada: technical report prepared by Roscoe Postle Associates for NexGen Energy Ltd., effective date 31 March 2017
- Cox, J.J., Robson, D.M., Mathisen, M.B., Ross, D.A., Coetzee, V., and Wittrup, M., 2017: Technical Report on the Preliminary Economic Assessment of the Arrow Deposit, Rook I Property, Province of Saskatchewan, Canada: technical report prepared by Roscoe Postle Associates for NexGen Energy Ltd., effective date 31 July 2017
- O'Hara, P, Cox, J.J., Robson, D.M., and Mathisen, M.B., 2018: Technical Report on the Pre-feasibility Study of the Arrow Deposit, Rook I Property, Province of Saskatchewan, Canada: technical report prepared by Wood and Roscoe Postle Associates for NexGen Energy Ltd., effective date 05 November 2018

2.7 List of Abbreviations

The following abbreviations are used throughout this report to express units of measurement.

A	annum	kWh	kilowatt-hour
A	ampere	L	litre
bbl	barrels	lb	pound
btu	British thermal units	L/s	litres per second
°C	degree Celsius	m	metre
C$	Canadian dollars	M	mega (million); molar
cal	calorie	m^2	square metre
cfm	cubic feet per minute	m^3	cubic metre
cm	centimetre	µ	micron
cm^2	square centimetre	MASL	metres above sea level
d	day	µg	microgram
dia	diameter	m^3/h	cubic metres per hour
dmt	dry metric tonne	mi	mile

dwt	dead-weight ton
°F	degree Fahrenheit
ft	foot
ft^2	square foot
ft^3	cubic foot
ft/s	foot per second
g	gram
G	giga (billion)
Gal	Imperial gallon
g/L	gram per litre
Gpm	Imperial gallons per minute
g/t	gram per tonne
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
ha	hectare
hp	horsepower
hr	hour
Hz	hertz
in.	inch
in^2	square inch
J	joule
k	kilo (thousand)
kcal	kilocalorie
kg	kilogram
km	kilometre
km^2	square kilometre
km/h	kilometre per hour
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
min	minute
µm	micrometre
mm	millimetre
mph	miles per hour
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
oz	Troy ounce (31.1035 g)
oz/st, opt	ounce per short ton
ppb	part per billion
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
s	second
st	short ton
stpa	short ton per year
stpd	short ton per day
t	metric tonne
tpa	metric tonne per year
tpd	metric tonne per day
US$	United States dollar
USg	United States gallon
Usgpm	US gallon per minute
V	volt
W	watt
wmt	wet metric tonne
wt%	weight percent
yd^3	cubic yard
yr	year

3.0 RELIANCE ON OTHER EXPERTS

The QPs have relied on the expert reports listed throughout this section. Section 3.1 of the report was prepared by RPA, while Sections 3.2, 3.3, 3.4, and 3.5 were prepared by Stantec.

3.1 Mineral Tenure, Surface Rights, and Encumbrances

RPA has relied exclusively on the land tenure holdings assessment provided by NexGen in respect to the legal matters contained in Section 4.1 – Land Tenure and RPA expresses no opinion as to the ownership status of the Property.

3.2 Taxation

The QPs have relied upon, and disclaim responsibility for, information supplied by NexGen Energy Ltd. staff and experts retained by NexGen Energy Ltd. for information related to taxation as follows.

- Ernst and Young, 2020: Taxation and Royalties: letter prepared by Ernst and Young for Stantec Consulting Ltd. and NexGen Energy Ltd., 21 January 2021, 3 pages.

This information is used in support of the financial analysis in Section 22.0, and the Mineral Reserve estimation in Section 14.0.

3.3 Royalties

The QP has relied upon, and disclaim responsibility for, information supplied by NexGen Energy Ltd. staff and experts retained by NexGen Energy Ltd. for information related to taxation and royalties as follows.

- Ernst and Young, 2020: Taxation and Royalties: letter prepared by Ernst and Young for Stantec Consulting Ltd. and NexGen Energy Ltd., 21 January 2021, 3 pages.

This information is used in support of the financial analysis in Section 22.0, and the Mineral Reserve estimation in Section 14.0.

3.4 Market and Uranium Price

The QP disclaims responsibility for the marketing and uranium price forecast information included in this report as the QPs relied on experts retained by NexGen for this information. Marketing and uranium pricing information was sourced from the following documents.

- UxC Special Report November 2020, (Ux Consulting, 13 November 2020)

- NexGen Feasibility Study U_3O_8 Final Marketing Memo (RPA, 06 October 2020)

Marketing and uranium pricing information from these sources is referenced in Section 19.0 of this report. The information is also used in support of the Mineral Reserve estimate in Section 14.0, and the financial analysis in Section 22.0.

The QPs consider it acceptable to rely on UxC for this information as the company is one of the nuclear industry's leading market research and analysis companies. UxC offers a wide range of services spanning the entire nuclear fuel cycle, with a special focus on market-related issues. Publications are the primary focus of UxC's activities, but UxC's team of experts also provide custom services for clients in all areas of the nuclear energy market.

UxC publishes Ux Weekly (a newsletter that reports the weekly industry standard Ux U_3O_8 Price®). UxC also regularly publishes market outlook reports regarding uranium enrichment, conversion, and fabrication, and nuclear power.

3.5 Environmental, Permitting and Social or Community Considerations

The QP for Environmental, Permitting and Social or Community Considerations is relying upon reports written by technical experts who are not qualified persons. All technical information included in Sections 1.16.2, 1.16.6, 1.16.7, 20.0, 20.1, 20.2, 20.4.1, 20.5, 20.6.1, 20.7, and 20.8 are completely reliant upon the information provided in documents identified in Section 27.0.

4.0 PROPERTY DESCRIPTION AND LOCATION

The NexGen Rook I property is located in northern Saskatchewan, approximately 40 km east of the Alberta–Saskatchewan border, 150 km north of the town of La Loche (see Figure 4-1), and 640 km northwest of the city of Saskatoon.

The property lies within parts of National Topographic System (NTS) map sheets 74F/7, 74F/10, and 74F/11, and it is approximately centred at Universal Transverse Mercator (UTM) coordinates of 620,000 mE and 6,385,000 mN (NAD 83, Zone 12N). It is shaped in a rectangular fashion with approximate dimensions of 38 km (northwest–southeast) by 10 km (northeast–southwest). The Arrow Deposit is located at approximate UTM coordinates of 604,350 mE and 6,393,600 mN.

Figure 4-1 shows a map of part of Saskatchewan that indicates where the NexGen Rook I property is located.

4.1 Land Tenure

The NexGen Rook I property consists of 32 contiguous mineral claims with a total area of 35,065 ha. All claims are 100% owned by NexGen. Six of the 32 claims are subject to: (i) a 2% NSR royalty payable to ARC, and (ii) a 10% production carried interest with Terra Ventures Inc. (Terra); however, the Arrow Deposit is located outside of the six claims. The NSR may be reduced to 1% upon payment of $1.0 million to ARC. The property formerly consisted of nine larger dispositions which were acquired by NexGen in 2012. In 2015, NexGen divided eight of those dispositions into 32 smaller dispositions to accommodate a more efficient spreading of mineral assessment credits over the property.

All claims are in good standing until at least 2039, and the claim that hosts the Arrow Deposit (S-113927) is in good standing until 2042.

Table 4-1 presents general information regarding all of the Rook I claims (e.g., anniversary dates, areas, annual expenditures), and Figure 4-2 shows a map of all the claims that indicates where the Arrow Deposit is located.

Figure 4-1: NexGen Rook I Project Property Location Map

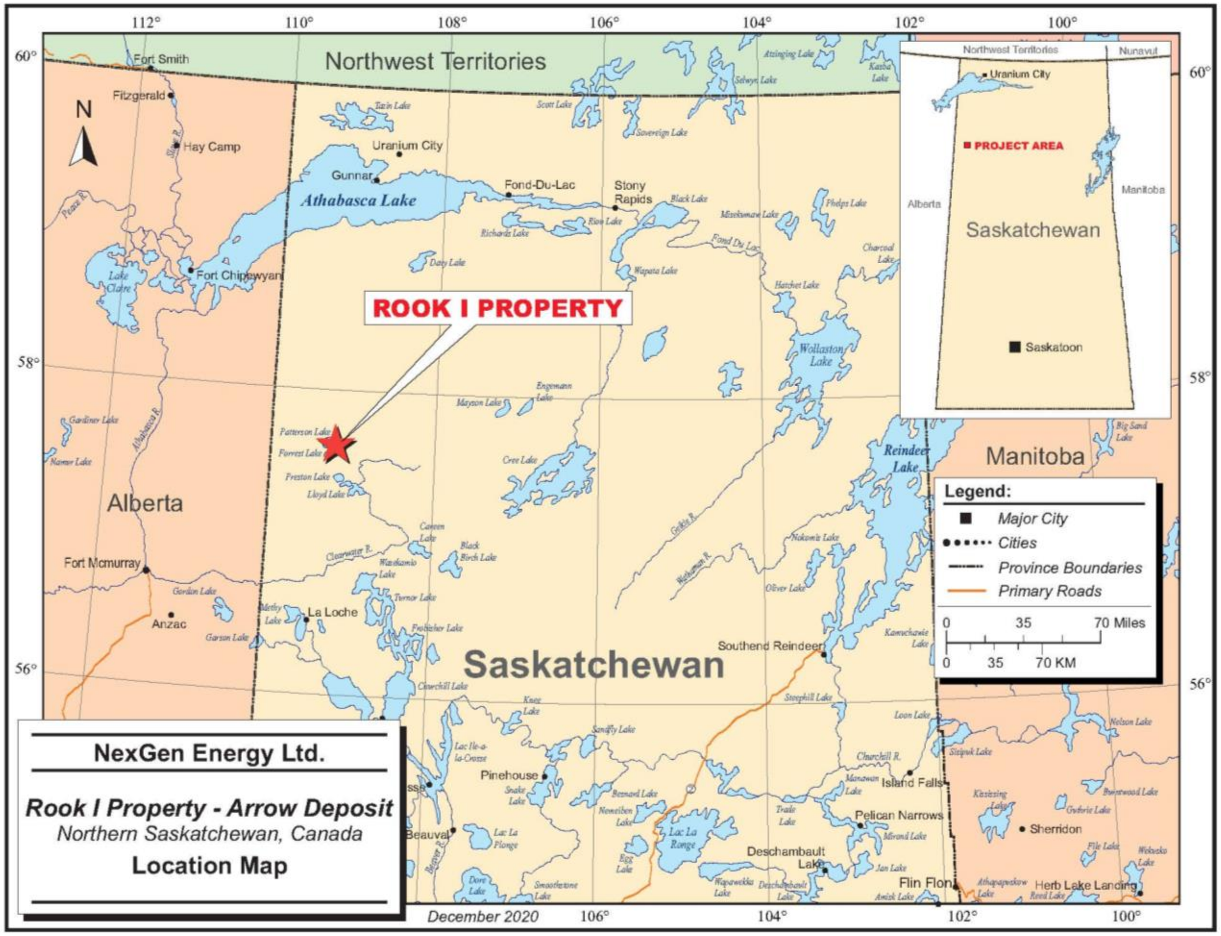
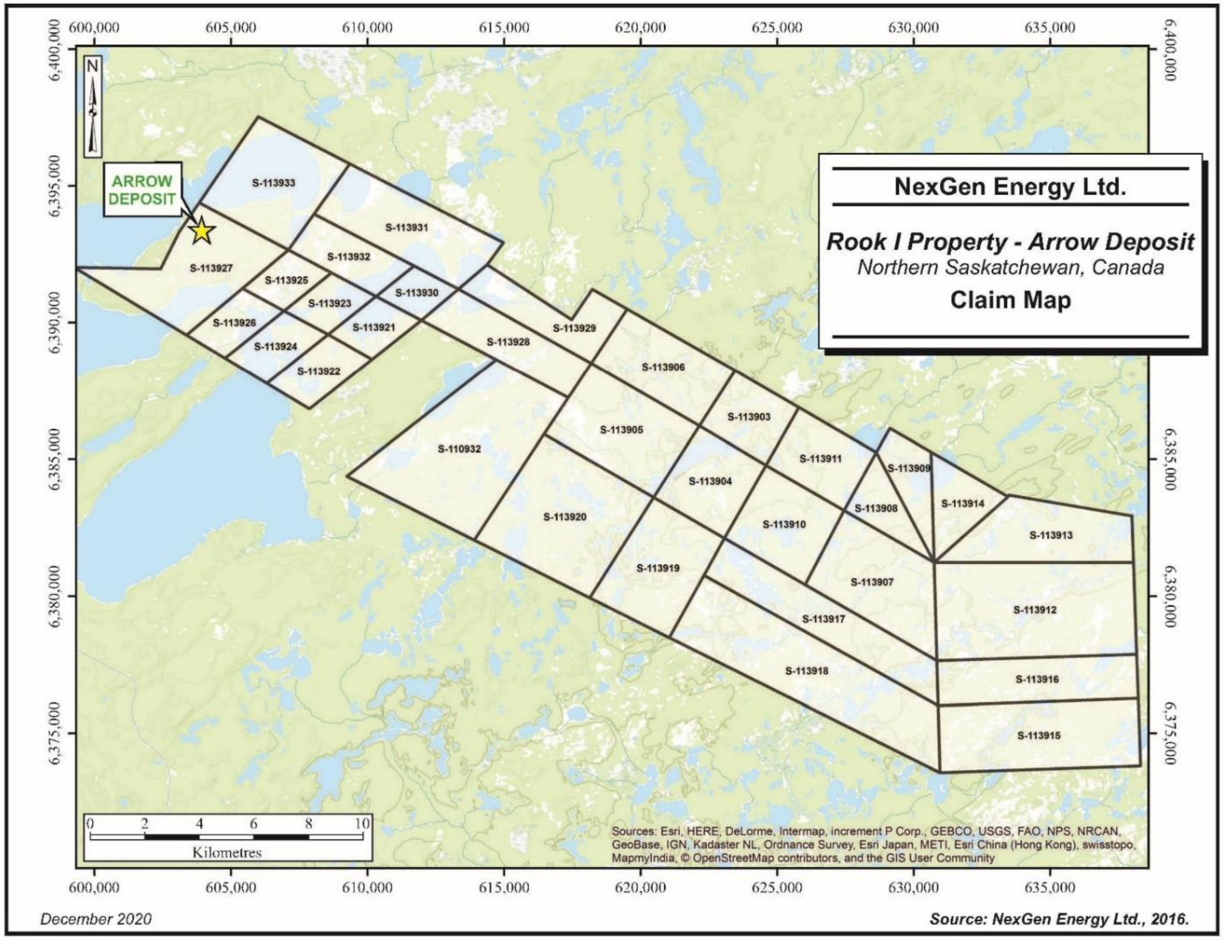


Table 4-1: Rook I Claims

Disposition Number	Previous Disposition Number	NTS	Record Date	Anniversary Date	In Good Standing Until	Area (ha)	Annual Expenditure ($)
S-110932	S-110932	74F/11	17-Mar-08	17-Mar-19	13-Jun-40	2,558	63,950
S-113903	S-110575	74F/10	13-Feb-07	13-Feb-19	13-May-43	673	16,825
S-113904	S-110575	74F/10	13-Feb-07	13-Feb-19	13-May-43	900	22,500
S-113905	S-110575	74F/10, 74F/11	13-Feb-07	13-Feb-19	13-May-43	1,432	35,800
S-113906	S-110575	74F/10, 74F/11	13-Feb-07	13-Feb-19	13-May-43	1,092	27,300
S-113907	S-110574	74F/10	13-Feb-07	13-Feb-19	13-May-43	1,436	35,900
S-113908	S-110574	74F/10	13-Feb-07	13-Feb-19	13-May-40	462	11,550
S-113909	S-110574	74F/10	13-Feb-07	13-Feb-19	13-May-42	492	12,300
S-113910	S-110574	74F/10	13-Feb-07	13-Feb-19	13-May-43	1,029	25,725
S-113911	S-110574	74F/10	13-Feb-07	13-Feb-19	13-May-43	800	20,000
S-113912	S-110573	74F/10	13-Feb-07	13-Feb-19	13-May-40	2,539	63,475
S-113913	S-110573	74F/10	13-Feb-07	13-Feb-19	13-May-42	1,280	32,000
S-113914	S-110573	74F/10	13-Feb-07	13-Feb-19	13-May-40	560	14,000
S-113915	S-110572	74F/10, 74F/7	13-Feb-07	13-Feb-19	13-May-40	1,806	45,150
S-113916	S-110572	74F/10	13-Feb-07	13-Feb-19	13-May-43	1,187	29,675
S-113917	S-110934	74F/10	17-Mar-08	17-Mar-19	13-Jun-39	1,385	34,625
S-113918	S-110934	74F/10, 74F/7	17-Mar-08	17-Mar-19	13-Jun-39	2,481	62,025
S-113919	S-110933	74F/10, 74F/11	17-Mar-08	17-Mar-19	13-Jun-40	1,328	33,200
S-113920	S-110933	74F/10, 74F/11	17-Mar-08	17-Mar-19	13-Jun-40	2,098	52,450
S-113921	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-42	392	9,800
S-113922	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-42	498	12,450
S-113923	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-42	378	9,450
S-113924	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-42	475	11,875

Disposition Number	Previous Disposition Number	NTS	Record Date	Anniversary Date	In Good Standing Until	Area (ha)	Annual Expenditure ($)
S-113925	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-42	360	9,000
S-113926	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-42	429	10,725
S-113927	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-42	1,514	37,850
S-113928	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-42	920	23,000
S-113929	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-42	811	20,275
S-113930	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-42	303	7,575
S-113931	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-42	1,395	34,875
S-113932	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-42	627	15,675
S-113933	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-42	1,425	35,625
Total						**35,065**	**876,625**

Figure 4-2: Claim Map



4.2 Mineral Rights

In Canada, natural resources fall under provincial jurisdiction. All Mineral Resource rights in the province of Saskatchewan are governed by the Crown Minerals Act and the Mineral Tenure Registry Regulations, 2012. Both are administered by the Saskatchewan Ministry of the Economy. Mineral rights are owned by the Crown and are distinct from surface rights.

To ensure that mineral claims are maintained good standing in Saskatchewan, the claim holder must undertake prescribed minimum exploration work on a yearly basis. The current requirements are $15/ha per year for claims that have existed for 10 years or less, and $25/ha per year for claims that have existed in excess of 10 years.

All dispositions at the property are subject to minimum work requirements of $25/ha per year as they were recorded in 2005 to 2008. Excess expenditures can be accumulated as credits for future years, and it is also possible to group contiguous claims and apply work from one disposition to several dispositions, with a maximum grouping size of 18,000 ha.

Mineral claims in good standing may be converted to mineral leases upon application. Mineral leases allow for mineral extraction, have 10-year terms, and are renewable. Surface facilities constructed in support of mineral extraction require a surface lease. Surface leases have 33-year maximum terms and are also renewable.

As of 06 December 2012, mineral dispositions are defined by the government as electronic mineral claims parcels that have been logged in the MARS using a GIS. MARS is a web-based, electronic tenure system that is used for the purpose of issuing and administrating mineral permits, claims, and leases. Administration of mineral dispositions is also web-based, and mineral claims are now acquired through electronic map staking.

As of the effective date of this report, all 32 mineral claims comprising the property are in good standing and are registered in the name of NexGen. NexGen has the required surface rights associated with the mineral claims that make up the property and has legal access to all the property mineral claims for exploration programs.

4.3 Royalties and Other Encumbrances

Six of the 32 claims that make up the Property are subject to a 2% NSR royalty payable to ARC, and a 10% production carried interest with Terra; however, the Arrow Deposit is located outside of the six claims. These claims are S-113928, S-113929, S-113930, S-113931, S-113932, and S-113933. The NSR may be reduced to 1% upon payment of $1.0 million to ARC.

The 10% production carried interest provides Terra with a right to 10% of potential future production, provided Terra repays NexGen (from 75% of the holder's share of production) their 10% pro rata portion of the collective expenditure from 20 June 2005. The Mineral Resources reported in Section 14.0 of this report do not occur within claims covered by the 2% NSR or 10% production carried interest and therefore the Arrow Deposit is free of royalties.

Other than as set forth above, the property is not subject to any royalties, back-in rights, payments or other agreements and encumbrances.

4.4 Permitting

To conduct exploration activities in Saskatchewan, the owner must be registered in the province and the requisite permits must be acquired. To carry out exploration on the ground, the following permits are required.

- Surface Exploration Permit
- Forest Product Permit
- Aquatic Habitat Protection Permit

Drill programs also require a Term Water Rights Permit from the Saskatchewan Watershed Authority, and notice must be given to the ENV, the Heritage Resource Branch, and the Water Security Agency. If exploration work is being staged from a temporary work camp, a Temporary Work Camp Permit is also required.

Temporary work camps typically trigger the need for a Term Water Rights Permit if surface water will be used for camp purposes. Relevant agency notification requirements also apply. NexGen has all required permits to conduct its proposed mineral exploration. However, additional permits will be required for development.

Canada North Environmental Services LP (CanNorth) completed a Heritage Resources Impact Assessment (HRIA) for the Project from 19-22 June 2018. The field assessment was completed under the Archaeological Resource Investigation Permit No. 18-068. The Heritage Study Area (HSA) established encompassed the project area, and three general areas within the HSA required a HRIA based on defined criteria.

In total, 180 ha were assessed using a combination of pedestrian reconnaissance, post-impact inspections of disturbed areas, and excavation of 239 subsurface shovel probes (Canada North Environmental Services, 2018). No new heritage resources were identified throughout the survey area.

On 26 November 2018, the Heritage Conservation Branch confirmed that the HRIA met the requirements of Section 63 of the Heritage Property Act and no further assessment was deemed necessary (Government of Saskatchewan 2018 letter to CanNorth).

RPA is not aware of any environmental liabilities to which the property is subject. RPA is not aware of any other significant factors and risks that may affect access, title, or NexGen's right or ability to perform the proposed work program on the property.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The property is most easily accessed via all-weather gravel Highway 955, which runs north-south, approximately km west of the Arrow Deposit. The highway is maintained year-round by the Province of Saskatchewan. Highway 955 begins in La Loche—the population centre nearest to the property—and continues 75 km to the north of the property to the decommissioned Cluff Lake Mine site.

La Loche is located 150 km to the south of the property and is connected to Prince Albert and Saskatoon via paved provincial highways. Fort McMurray, Alberta, is 180 km southwest of the property and can be reached via a winter road from December to April.

From Highway 955, a 13 km long, all-weather, single-lane gravel road provides access to the western portion of the property, including the Arrow Deposit area. There are several passable four-wheel drive roads and trails that allow for access to much of the property. Fixed wing aircrafts on floats can land on lakes on and near the property, and remote areas of the property are accessible via helicopter.

5.2 Climate

The property experiences a subarctic climate typical of mid-latitude continental areas. Temperatures range from greater than 30 °C in the summer to colder than –40 °C during the winter.

In area of the province in which the property is located, winters are long and cold, with mean monthly temperatures of below freezing for seven months. Annual precipitation is approximately 0.5 m, half of which is rain during the warmer months, and the remainder of which is 70-100 cm of snow.

Ice formation on water bodies typically begins in October, with break-up occurring in April. Drilling can be carried out year-round, although ground access is affected by freeze-up and break-up. Ground geological and geochemical surveys are typically restricted to the summer months when the ground is free of snow.

5.3 Local Resources

Fuel, groceries, emergency medical services, and basic construction services are available in La Loche. Buffalo Narrows is 100 km south of La Loche, which also has fixed-wing float planes for charter.

An outfitters lodge is located approximately 20 km north of the Rook I property, on Highway 955. Other services are available in abundance in Prince Albert and Saskatoon.

5.4 Infrastructure

There is no permanent infrastructure on the property other than core logging, storage buildings, and an exploration camp. There is a power line 70 km south of the property; however, the transmission capacity of this line is unsuitable for a major industrial site.

The property has sufficient space for an UG mining operation, including space for waste rock storage areas (WRSAs). Water is readily available.

5.5 Physiography

The topography of the property area varies, with drumlins and lakes / wetlands dominating the northwest and southeast parts of the property, respectively, and lowland lakes, rivers, and muskegs dominating the central part of the property. Elevations range from 583 masl in drumlins, to 480 masl in lowland lakes. The elevation of Patterson Lake is 499 m. Bedrock outcrops are very rare but do exist in areas of the southeastern half of the property.

The northwestern part of the property lies over portions of Patterson Lake and Forrest Lake, two of the largest waterbodies within 100 km of the property. Both lakes are part of the Clearwater River watershed. The Clearwater River extends east-southeast from Beet Lake, and eventually drains south of the property.

The property is covered by boreal forest common to the Canadian Shield. The most common trees are jack pine and black spruce, with some poplar and birch clusters. Tamarack, stunted black spruce, willow, and alder trees are also common in the lower wetland areas.

Wildlife species common to the area include moose, deer, black bears, wolves, and a variety of other mammals commonly found in boreal forest ecosystems. Common fish species include pickerel (walleye), lake trout, rainbow trout, northern pike, whitefish, and perch.

6.0 HISTORY

6.1 Prior Ownership

Pursuant to an agreement to purchase mineral claims dated 20 June 2005 (as amended), Titan Uranium Inc. (Titan) purchased disposition S-108095 (now S-113928 through S-113933) from 455702 B.C. Ltd. and 643990 B.C. Ltd. The remainder of the claims comprising the property were subsequently ground-staked by Titan in 2007 and 2008.

In 2012, pursuant to a mineral property acquisition agreement between Titan and Mega Uranium Ltd. (Mega), Titan sold the Rook I property to Mega. NexGen acquired the property from Mega following an asset purchase agreement dated 14 November 2012.

6.2 Exploration and Development History

A summary of the NexGen Rook I Project (the Project) site exploration history is provided in Table 6-1.

Table 6-1: Exploration History

Year	Operator	Comment
1968–1970	Bow Valley Company Ltd. Wainoco Oil and Chemicals Ltd. Canada Southern Petroleum and Gas Ltd.	Bow Valley Company Ltd.'s Permits 1 and 6, Wainoco Oil and Chemicals Ltd.'s Permit 1, and the Canada Southern Petroleum and Gas Ltd. Permit 6 covered parts of what is now the project area. The companies completed airborne magnetic and radiometric surveys, and carried out prospecting and geochemical sampling. Results were not encouraging, and the permits were dropped.
1974	Uranerz Exploration and Mining Ltd.	Inexco Permits 1 and 2 covered the project area. Completed geological mapping, prospecting, lake sediment sampling, and a helicopter-borne radiometric survey. No significant results were returned.
1976–1982	Canadian Occidental Petroleum Ltd. (CanOxy). Houston Oil and Gas Ltd. Hudson Bay Exploration and Development Company Ltd. (HBED) Kerr Addison Mines Ltd. (Kerr) SMDC (now Cameco)	CanOxy had claims CBS 4745, 4756, 4747, and 4748 covering most of the area of current dispositions S-110932 and S-113921 through S-113933. Houston Oil and Gas Ltd. had one claim (CBS 5680) covering parts of claims S-113903 through S-113906. HBED had two small claims covering S-113919 and S-113920, and Kerr had claims covering parts of S-113903, S-113904, and S-113907 through S-113914. SMDC (now Cameco), had MPP 1076 (later CBS 8807), which covered parts of S-113929, S-113931, and S-113933. These companies completed airborne INPUT EM surveys which detected numerous conductors, many of which were subject to ground surveys prior to drilling. Airborne magnetic-radiometric surveys were also completed and followed up by prospecting, geological mapping, lake sediment surveys, and some soil and rock geochemical sampling. Few anomalies were found other than those located by the airborne and ground EM surveys.
1977–1979	Kerr	Kerr drilled 24 holes in the project site area, one of which was drilled in current disposition S-113903. No other holes were drilled on the property. No significant results were intersected.
1978–1980	CanOxy	CanOxy drilled 41 holes for its CLU project during 1978 through 1987; however, only 20 of these were on the project site dispositions. Drilling did not intersect any uranium mineralization, but did intersect thick glacial till deposits, basement regolith, and geological structures.
1980–1982	SMDC (now Cameco)	SMDC (now Cameco) drilled 13 holes, PAT-01 to PAT-13 on what is now S-113933; this identified the Bow occurrence. Mineralization and alteration were reported to be similar to that seen at unconformity-associated uranium deposits in the Athabasca Basin.
1978-1982	HBED	HBED drilled three holes on claims which cover part of what is now S-113920. They intersected graphitic gneisses, but no radioactivity was discovered.

Year	Operator	Comment
2005–2008	Titan	Titan carried out airborne time-domain EM surveys, MegaTEM and VTEM, which detected numerous strong EM anomalies. A ground MaxMin II survey in 2008 confirmed the airborne anomalies.
2012	Mega	Following a mineral property acquisition agreement between Mega and Titan dated 1 February 2012, Mega acquired all nine dispositions comprising the project site. A gravity survey was completed over 60% of S-113921 through S-113933 (Creamer and Gilman, 2013a), which defined several regional features and some more local smaller scale features. Simultaneously, Mega sampled organic rich soils and prospected the same area as the gravity survey. No soil geochemical anomalies or radioactive boulders were found.
2012	NexGen	NexGen acquired Mega's interest in the property.
2013–2020	NexGen	NexGen completed ground gravity surveys, ground DCIP surveys, an airborne magnetic-radiometric-VLF survey, airborne VTEM survey, an airborne ZTEM survey, airborne gravity survey, radon-in-water geochemical survey, ground radiometric and boulder prospecting program, and core drilling. Discovered Area A occurrence in 2013, Arrow Deposit in 2014, Camp East, Harpoon and Cannon occurrences in 2016, and South Arrow Discovery in 2017. After discovery of the Arrow Deposit, NexGen completed additional drilling, Mineral Resource estimates, a preliminary economic assessment (PEA) in 2017, a PFS in 2018, and a FS in 2021.

6.3 Historical Resource Estimates

No resource estimates have been prepared by previous owners or previous claim holders of the area.

6.4 Past Production

There has been no production on the property up to the effective date of this report.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The NexGen Rook I property is located along the southwestern rim of the Athabasca Basin, a large Paleoproterozoic-aged, flat-lying, intracontinental, fluvial, redbed sedimentary basin. The Athabasca Basin covers much of northern Saskatchewan and part of northern Alberta (Jefferson et al., 2007).

The Athabasca Basin is oval-shaped at surface, with approximate dimensions of 450 km by 200 km (see Figure 7-1). The Athabasca Basin reaches a maximum thickness of approximately 1,500 m near the centre, and it consists principally of unmetamorphosed sandstone, with local conglomerate beds that are collectively known as the Athabasca Group. Every geologic unit comprising the Athabasca Group contains crossbedding and ripple cross-lamination. Most units also contain single-layer thick quartz pebble or granule beds.

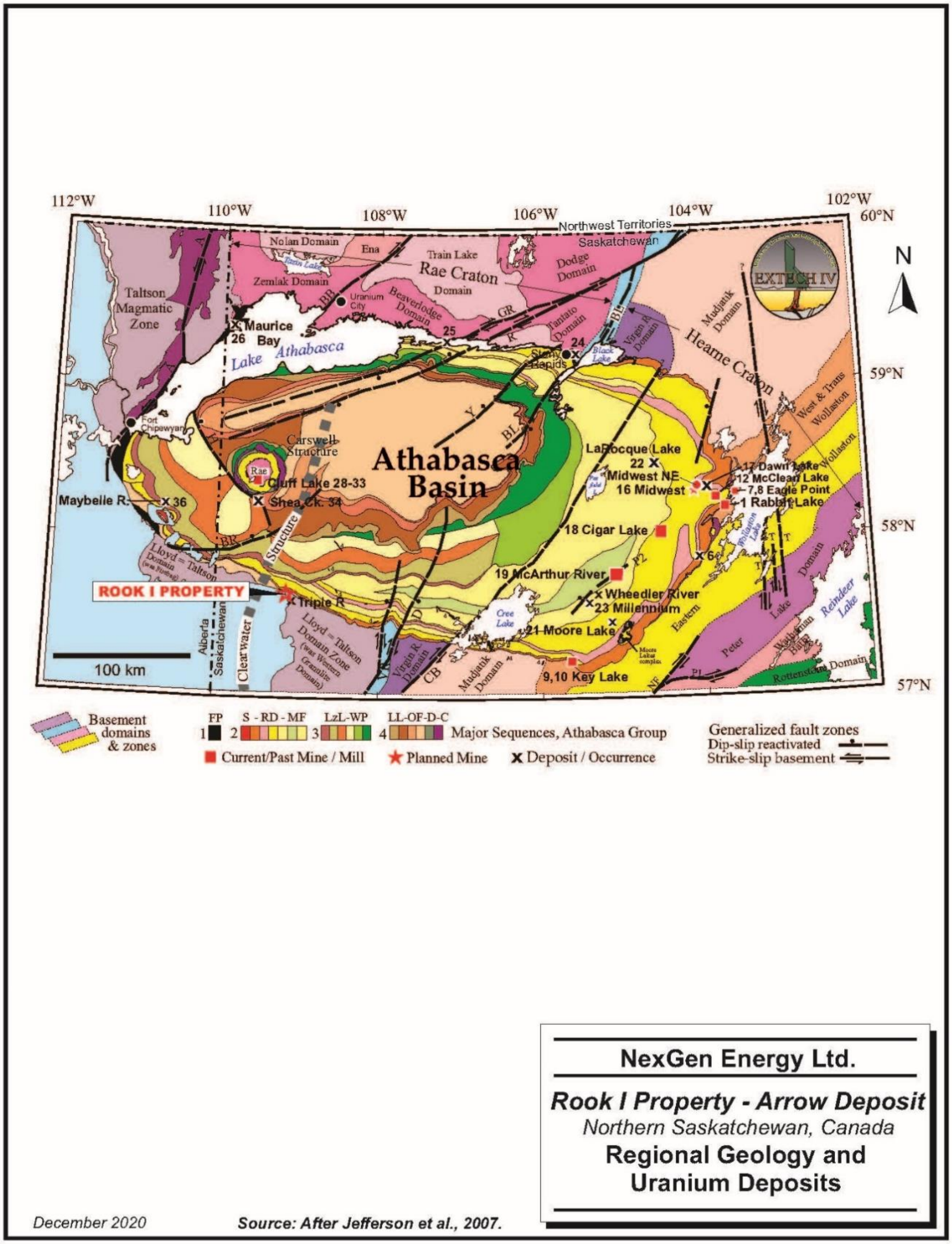
The base of the Athabasca Group is marked by an unconformity with the underlying crystalline basement rocks of the Archean to Paleoproterozoic-aged Hearne and Rae provinces to the east and west, respectively, and of the Proterozoic Taltson Magmatic Zone (TMZ) to the west (Card et al., 2007). The Rae Province consists predominantly of metasedimentary supracrustal sequences and granitoid rocks.

In contrast, the Hearne Province consists primarily of granitoid gneisses that contain supracrustal rocks. The TMZ is characterized as a basement complex that was intruded by both continental magmatic arc granitoid rocks and peraluminous granitoid rocks. The Hearne and Rae Provinces are separated near the centre of the Athabasca Basin by the northeast trending Snowbird Tectonic Zone.

The Athabasca Group basal unconformity is spatially related to all significant uranium occurrences in the region. The basement immediately below the unconformity typically has a paleoweathered profile. It ranges in thickness from a few centimetres, to up to 220 m where fluid migration was aided by fault zones (MacDonald, 1980).

The paleoweathered profiles consist of a thin bleached zone at the unconformity, which grades into a hematite altered zone, and then a chlorite altered zone before alteration features dissipate. The southwestern portion of the Athabasca Group is overlain by flat-lying Phanerozoic rocks from the Western Canada Sedimentary Basin (i.e., mudstones, siltstones, and sandstones).

Figure 7-1: Regional Geology and Uranium Deposits



7.2 Local and Property Geology

The oldest rocks in the area of the Rook I property are in the TMZ. Within the property, the TMZ consists mostly of granitic, granodioritic, tonalitic, dioritic, and locally gabbroic gneisses (see Figure 7-2). There are also local bodies of graphitic and chloritic semipelitic to pelitic gneisses that typically occur as discontinuous, elongated, north-northeast trending lenses. They range in length from less than one kilometre to greater than 10 km (Grover et al., 1997).

The quartz-feldspar-garnet-biotite (± graphite) gneisses are the predominant host rock of uranium mineralization in basement settings in the area, including the Arrow Deposit. All lithologies present in the TMZ have been metamorphosed at upper amphibolite to granulate facies conditions.

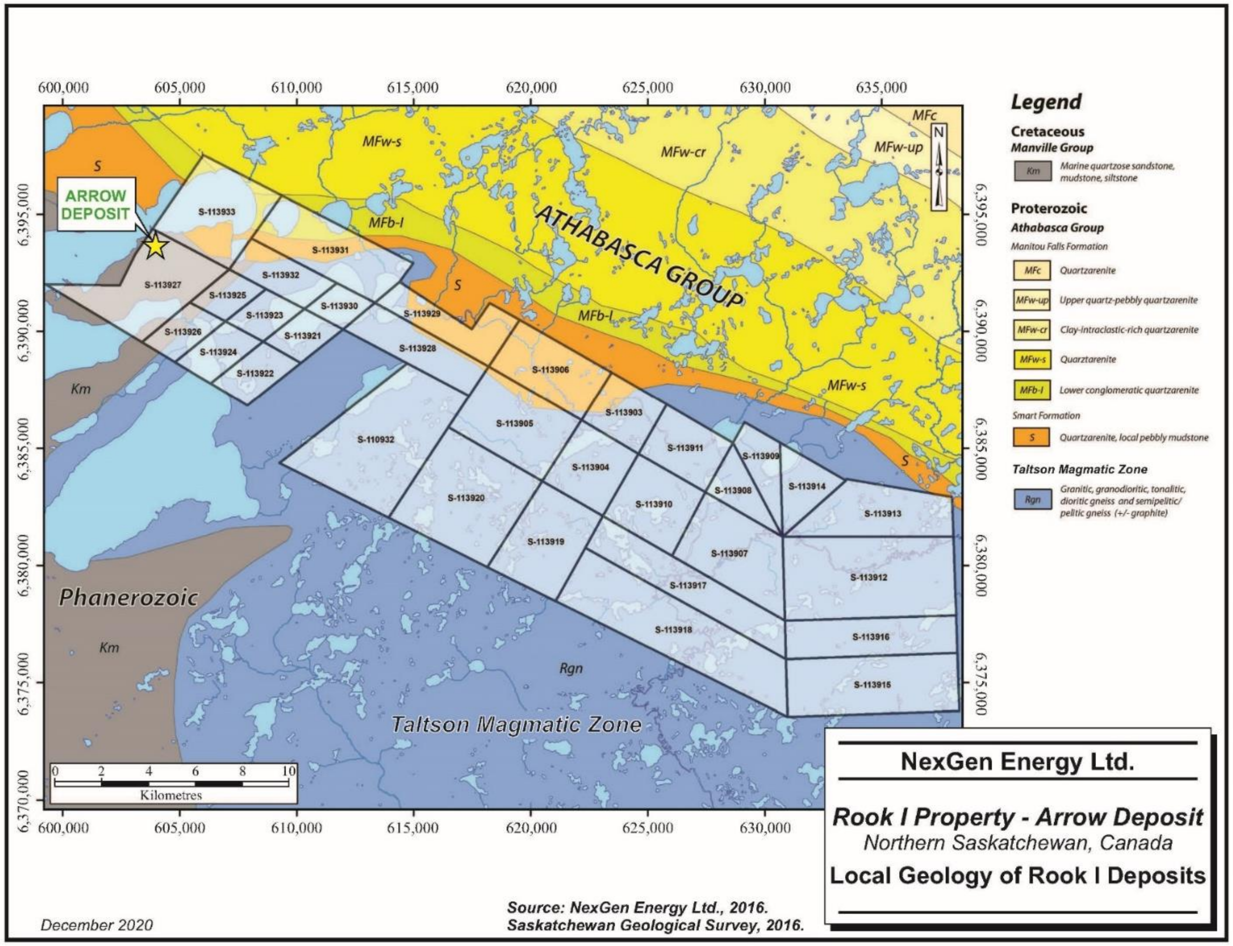
The Rook I property straddles the Athabasca Group basal unconformity. Overlying the basement rocks in the area of the property are the flat-lying sandstones of the Athabasca Group. Where they will be intersected during drilling, the Athabasca Group rocks are likely part of the Smart and Manitou Falls formations. These formations are characterized by both uniform quartz arenite beds and rare pebble conglomerate beds.

Phanerozoic rocks of the Cretaceous Mannville Group and Devonian La Loche Formation overlie the Athabasca Group and basement rocks in portions of the western side of the property, and above the Arrow Deposit. The Mannville Group is characterized by both non-marine and marine shales and sandstones.

A coal bed marker horizon at the bottom of the Mannville Group is often observed in drill core. The La Loche Formation consists of arenitic to arkosic sandstones and conglomerates.

The Clearwater Domain is immediately west of the property. It is a northeast trending belt of granitic rocks ranging in width from 20 km to 25 km. Although poorly exposed, the Clearwater Domain is marked by an aeromagnetic high that overprints the magnetic signature of the TMZ (Card et al., 2007). Where they are intersected during drilling, the felsic intrusive rocks of the Clearwater Domain often exhibit anomalous uranium concentrations.

Figure 7-2: Local Geology of Rook I Property



The property and surrounding area are covered by Pleistocene glacial deposits composed of sand, Athabasca Group sandstone boulders, and rare basement and Mannville Group boulders.

Glacial geomorphological topographic features are common; these include northeast to east-northeast trending drumlins, outwashes, hummocky terrain, and kettle lakes. The glacial deposits are typically at least 30 m thick, and may be up to 100 m thick. The glacial overburden over the Arrow Deposit is approximately 60 m thick.

7.3 Mineralization

Mineralization occurs at the following seven locations on the property (see Figure 7-3) and is exclusively hosted in basement lithologies below the unconformity within the Athabasca Group.

- Arrow Deposit
- South Arrow Discovery
- Harpoon occurrence
- Bow occurrence
- Cannon occurrence
- Camp East occurrence
- Area A occurrence

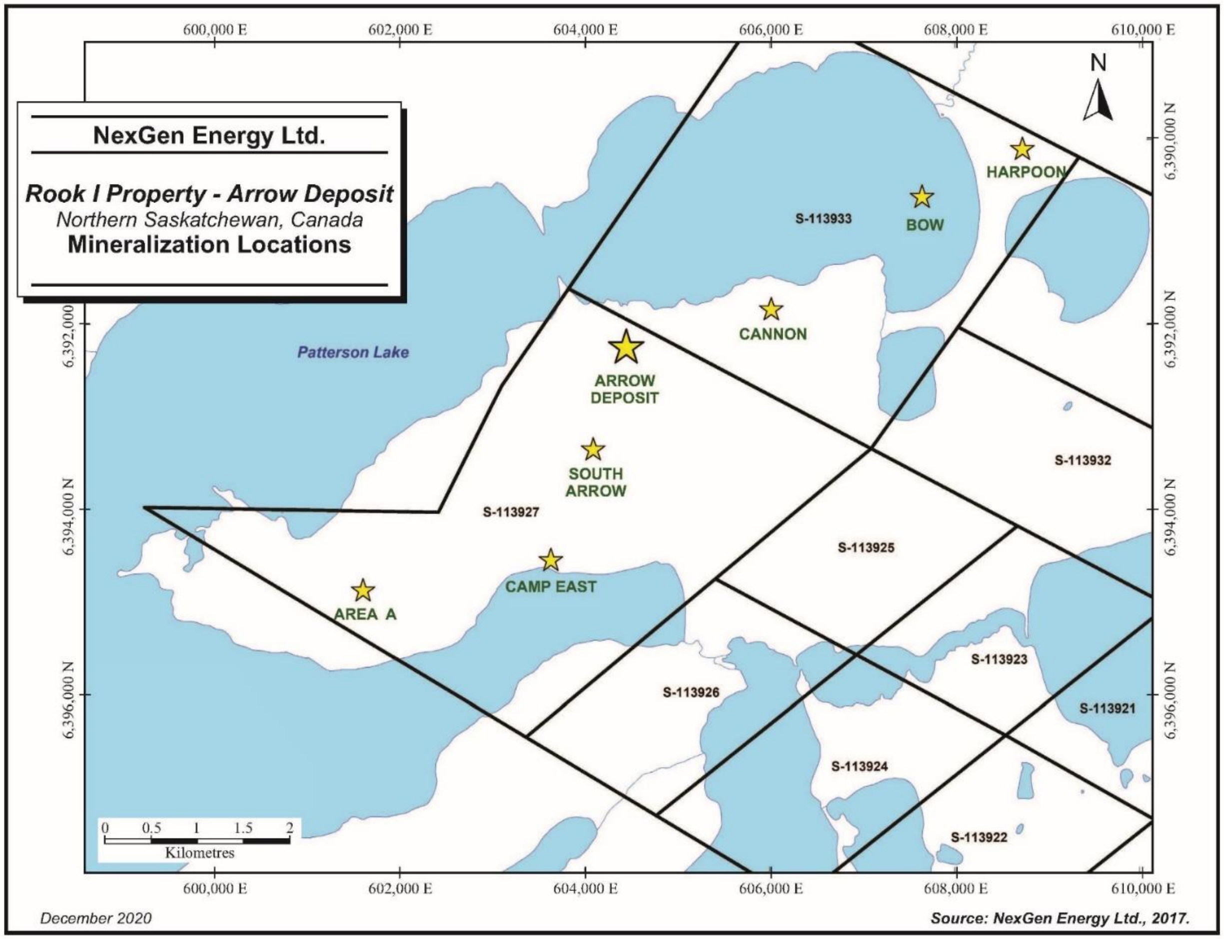
7.3.1 Arrow Deposit

Mineralization at the Arrow Deposit is defined by an area comprised of several steeply-dipping shears (i.e., A0 through A5), which locally host HG uranium mineralization. The mineralized area is 315 m wide, with an overall strike of 980 m.

Mineralization occurs 100 m below surface and extends to a depth of 950 m. The individual shear zones vary in thickness from two metres to 60 m. The Arrow Deposit is open in most directions and at depth.

Uranium mineralization at the Arrow Deposit is closely associated with narrow, strongly graphitic quartz-feldspar-garnet-biotite gneisses, which represent discrete shear zones. High grade uranium zones often occur immediately adjacent to heavily sheared and strongly graphitic zones.

Figure 7-3: Mineralization Locations



The Arrow Deposit is thought to be hosted in quartz-feldspar-garnet-biotite (± graphite) gneisses that are predominantly silicified. They consist of garnet porphyroblast pseudomorphs, which are now almost exclusively altered to chlorite, hematite, illite, or sudoite. Other minor mineral phases present include plagioclase, potassium feldspar, biotite, muscovite, and amphibole, in varying concentrations.

Additionally, the Arrow Deposit is also marked by the presence of intermediate orthogneisses consisting of quartz monzodioritic to quartz dioritic gneiss with tonalitic, granodioritic, and granitic gneiss. The main foliation present in the Arrow Deposit area trends towards the northeast and has vertical to sub-vertical dips.

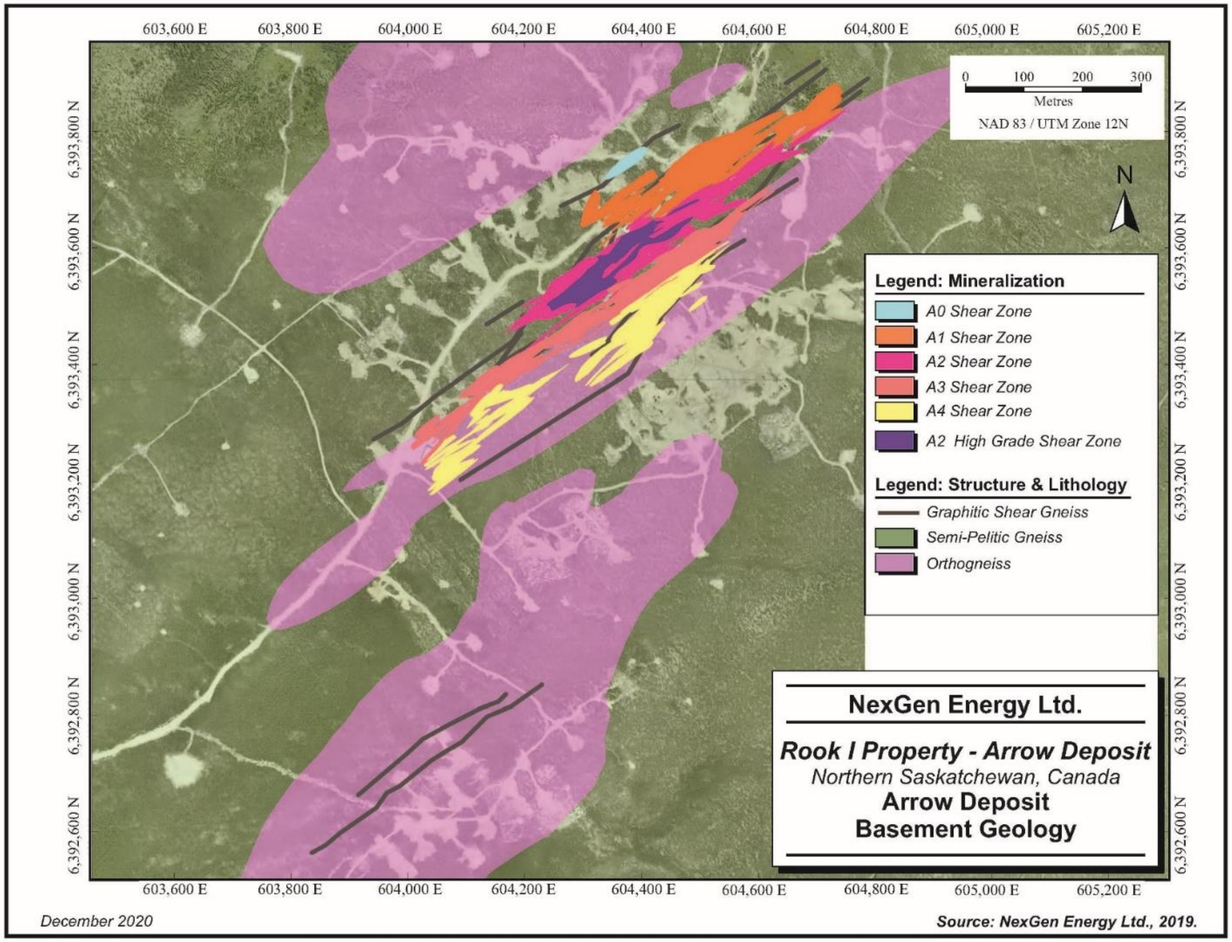
Hydrothermal alteration that occurs in the vicinity of the Arrow Deposit is extensive and several distinct styles have been observed.

- Quartz–sericite–sudoite–illite alteration
  - A pervasive alteration assemblage that nearly completely replaces the host rock, although pre-alteration textures are often preserved.
- Hematite alteration
  - Pervasive and brick red in colour.
- Dravite
  - Occurs in centimetre- to decimetre-wide breccia vein bodies beginning tens of metres from HG uranium mineralization and increasing in size and frequency closer to mineralization.
- Drusy quartz
  - Centimetre sized veins that occur ubiquitously in the vicinity of the deposit. Where proximal to HG mineralization, these veins are often pink coloured.

Mineralization at the Arrow Deposit occurs within six graphitic shears, referred to as the A0, A1, A2, A3, A4, and A5 shear zones. Figure 7-4 shows the modelled mineralization of the shear zones; the modelled mineralization in the A5 shear zone has been combined with the A4 shear zone for reporting purposes. Each shear zone is oriented parallel to foliation which strikes at approximately 050° to 060° and dips vertically to sub-vertically. The mineralization within the shear zones is also oriented parallel and sub-parallel to the regional foliation.

Of the recognized main parallel structural shear zones (A0, A1, A2, A3, A4, and A5), the A2 and A3 shears host higher grade, thicker, and more continuous mineralization than the other shear zones. A continuous zone of higher-grade mineralization in the A2 shear is known as the higher grade A2 High Grade Zone (A2-HG).

Figure 7-4: Arrow Deposit Basement Geology



Two key types of uranium mineralization occur at the Arrow Deposit: open space fillings and chemical replacement style uranium mineralization.

Open space fillings include massive uraninite bodies interpreted to be uranium veins, and breccia bodies where the matrix nearly exclusively comprises massive uraninite. Uranium veins and breccias typically range in thickness from less than 0.1 m to greater than 1.0 m, and display sharp contacts with the surrounding wall rocks.

Clasts present in uranium breccias at the Arrow Deposit are fragments of the immediate wall rocks, and they often contain additional disseminated uraninite mineralization. Uranium breccias occur in both clast-supported and matrix-supported forms, with the latter typically hosting higher uranium grades. Both styles of open space filling mineralization are characterized by high uranium grades.

Chemical replacement types of mineralization at the Arrow Deposit include disseminated, worm-rock, and near-complete to complete replacement styles. Disseminated mineralization is typically associated with strong to intense hydrothermal alteration where uraninite occurs as fine to medium grained anhedral crystals, and as crystal agglomerates spread throughout the host in concentrations of typically less than 5% by modal composition.

Worm-rock style mineralization is named for the texture it causes in rocks, which is the result of redox reactions between uranium-bearing fluids and the host wall rocks. Typically, these redox fronts are less than 10 cm thick.

Near-complete to complete uraninite replacement of the host rock has also been observed at the Arrow Deposit. These zones range in thickness from less than 0.1 m to greater than 1.0 m. In contrast to open space fillings, they show gradual contacts.

Near-complete to complete replacement bodies also often contain centimetre-long vugs, which may once have been garnet porphyroblasts, pseudomorphs that are common in the host rocks. The presence of vugs in this style of mineralization (in some zones interpreted to be uranium veins) suggests that in at least some places, the vein may actually be the result of chemical replacement and not open space filling.

Uranium mineralization on the property dominantly occurs as uraninite. Other common uranium minerals include coffinite and secondary yellow-coloured minerals, currently interpreted to be autunite, carnotite, and/or uranophane. A green-coloured secondary uranium mineral interpreted to be torbernite has also been observed very locally. In zones of massive uraninite mineralization, blebs of a glassy black-coloured phase with conchoidal fracture, currently interpreted to be pyrobitumen, are locally observed.

7.3.2 Harpoon Occurrence

The Harpoon occurrence is located 4.7 km northeast of the Arrow Deposit, and it has been traced over a strike length of 350 m. Harpoon mineralization expresses parallel foliation, striking towards the northeast at approximately 035° to 045°, and dipping towards the southeast at approximately 60° to 70°. The Harpoon occurrence has currently been drilled to within 27 m of the northeast boundary of the Project.

Basement lithologies observed in the area of mineralization include porphyroblastic quartz-feldspar-garnet-biotite (± graphite) gneiss, and intermediate orthogneisses of varying composition. The occurrence is currently exclusively basement-hosted and occurs within a chloritic and graphitic shear zone that is heavily clay altered.

Uranium mineralization occurs as semi-massive to massive uraninite veining and as worm-rock styles, chemical solution fronts, replacement bodies, and as fracture coatings.

7.3.3 Bow Occurrence

The Bow occurrence is located 3.7 km northeast of the Arrow Deposit. Anomalous uranium values occur at or just below the unconformity in fractured, slickensided, and sometimes brecciated sandstone and basement quartz feldspar-biotite (± graphite) gneisses. A strongly silicified unit was also noted in several drill holes.

Basement rocks are described as strongly bleached and clay altered. While no continuity has been established to date, the alteration and host rocks described are similar to what is seen in unconformity associated uranium deposits elsewhere in the Athabasca Basin.

7.3.4 Cannon Occurrence

The Cannon occurrence is located 1.3 km northeast of the Arrow Deposit. Basement lithologies present at the Cannon occurrence area largely consist of porphyroblastic quartz-feldspar-garnet-biotite (± graphite) gneiss and intermediate orthogneisses, with relatively narrow intervals of chloritic and graphitic mylonite, the latter of which host the LG uranium mineralization discovered to date.

Strong hydrothermal alteration, which typically includes illite-sudoite-hematite mineral assemblages, was commonly intersected in the basement rock in the area of the Cannon occurrence. The alteration zones remain open in all directions, and at the unconformity.

7.3.5 Camp East Occurrence

The Camp East occurrence is located approximately 2.3 km south-southwest of the Arrow Deposit. Lithologies in the area include porphyroblastic quartz-feldspar-garnet-biotite (± graphite) gneiss and intermediate orthogneisses. Chloritic and locally graphitic shear zones with widths ranging from one metre to tens of metres were intersected by drilling.

Drill holes that intersected weakly anomalous radioactivity also intersected very strong hydrothermal alteration over extensive core lengths intermittently over hundreds of metres. Two distinctive alteration styles are generally present in the Camp East occurrence area including the following.

- Near-complete to complete silica replacement, with accessory clay and hematite.
- Moderate to intense white clay and dravite alteration where near-complete to complete clay replacement is observed over core lengths of up to 12 m.

7.3.6 Area A Occurrence

Area A is situated approximately 3.5 km southwest of the Arrow Deposit. Visible uraninite was identified within a strongly hematite-altered breccia. Mineralization occurs within a 29 m wide shear zone marked by faults, fractures, a variety of veins, and breccias. The host rocks are garnetiferous quartz-plagioclase-biotite gneiss with minor graphite. Follow-up drilling failed to intersect mineralization.

7.3.7 South Arrow Discovery

The South Arrow Discovery is located 400 m south-southwest of the Arrow Deposit. The South Arrow Discovery consists of two parallel mineralized shear zones, with an overall strike of 290 m and is observed to occur within an 80 m wide area that extends from 110 m from surface to a depth of 550 m.

The shear hosting the South Arrow Discovery mineralization strikes to the northeast at approximately 045°, and dips towards the southeast between 70° and 83°. The mineralization at the South Arrow Discovery remains open in most directions and will require follow-up drilling.

Uranium mineralization at the South Arrow Discovery is exclusively basement-hosted, and lithologies observed in the area include porphyroblastic quartz-feldspar-garnet-biotite (± graphite) gneiss and intermediate orthogneisses. The mineralization consists of en-echelon uranium veins that occur within or proximal to chloritic and graphitic shears, with associated clay alteration. Uraninite mineralization occurs as semi-massive veining, worm-rock styles, chemical solution fronts, replacement bodies, and fracture coatings.

8.0 DEPOSIT TYPES

The Arrow Deposit is considered to be an example of a basement-hosted, vein-type uranium deposit.

At numerous locations in Saskatchewan, uranium deposits have been discovered at, above, and below the Athabasca Group unconformity. Mineralization can occur hundreds of metres into the basement, or can be perched up to 100 m above in the sandstone. No uranium has been identified at or above the unconformity within the Arrow Deposit.

Massive veins have been discovered in the basement, at depths ranging from immediately below the unconformity to greater than 800 m below it. Typically, uranium is present as uraninite / pitchblende, which occurs as veins, and semi-massive to massive replacement bodies.

In most cases, mineralization is also spatially associated with steeply-dipping, graphitic basement structures that have penetrated into the sandstones and offset the unconformity during successive reactivation events. Such structures are thought to represent both important fluid pathways chemical / structural traps for mineralization through geologic time as reactivation events have likely introduced further uranium into mineralized zones and provided a means for remobilization.

Two end members of unconformity-associated mineralization have been identified in the Athabasca Basin: egress type deposits and ingress type deposits (see Figure 8-1 and Figure 8-2).

Egress type deposits occur at or above the unconformity and are hosted by sandstone.

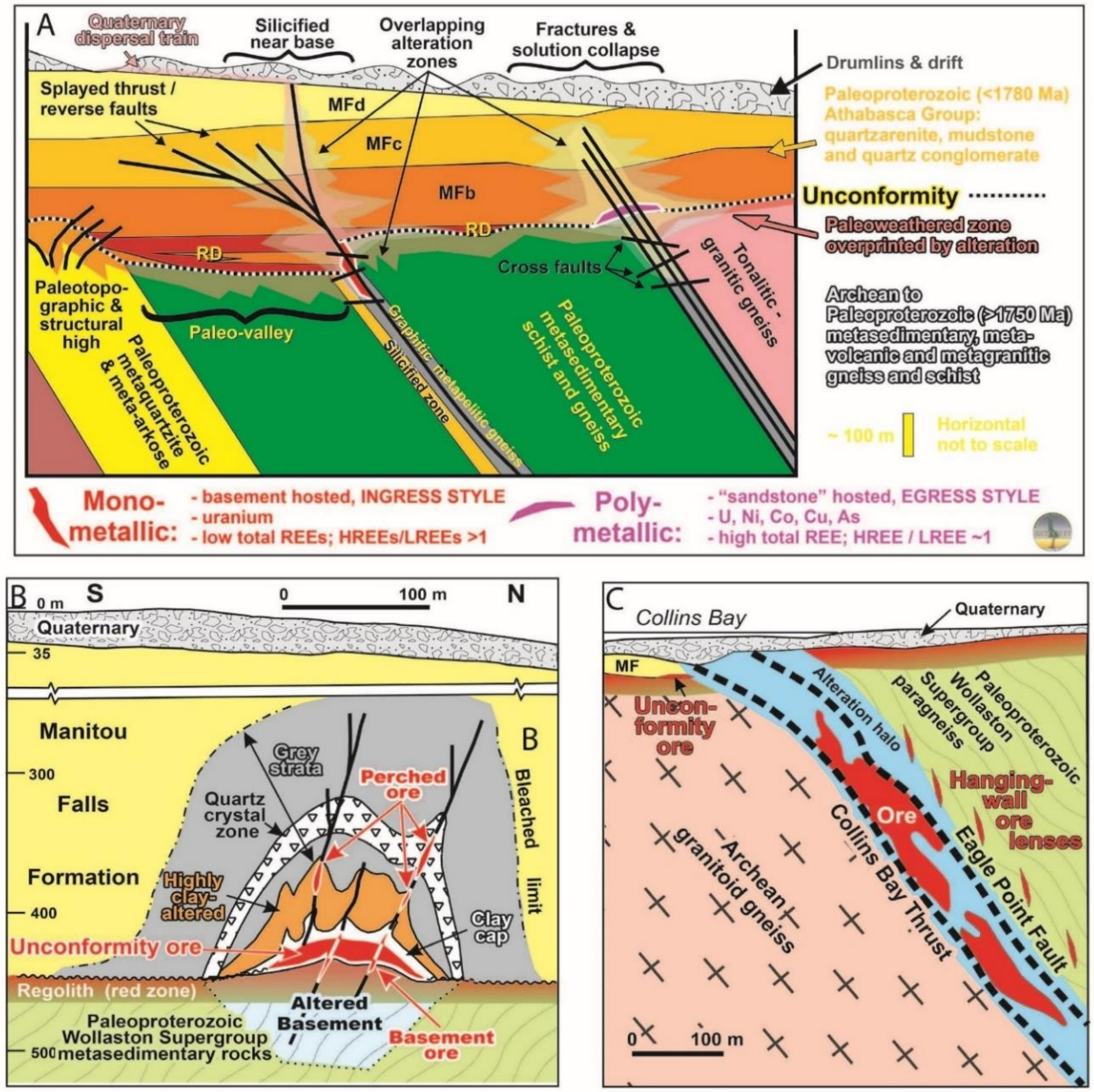
Ingress type deposits occur in basement rocks below the unconformity. The location and style of mineralization present at any deposit is the result of where fluid mixing between oxidizing basin fluids and reducing basement fluids occurred. If the two fluids interacted mostly at or above the unconformity, egress style mineralization is the result. Fluid mixing below the unconformity has led to the formation of ingress style mineralization.

Furthermore, egress style mineralization is often polymetallic and may contain appreciable concentrations of nickel (Ni), cobalt (Co), arsenic (As), and lead (Pb) in addition to uranium. Ingress style mineralization is typically monometallic, containing nearly exclusively uranium.

Unconformity-associated uranium deposits of the Athabasca Basin typically display extensive hydrothermal alteration halos, especially in the sandstones above major deposits where relatively high porosity / permeability allowed for increased fluid flux.

Where mineralization is basement-hosted, alteration is typically confined to structures in the basement. Chlorite, hematite, dravite, sudoite, illite, kaolinite, and dickite are often—but not always—key alteration phases associated with mineralization. Silicification and desilicification of sandstones is also empirically associated with mineralization at many deposits, especially those located at the unconformity and in the sandstone.

Figure 8-1: General Geological Setting of Unconformity Associated Uranium Mineralization

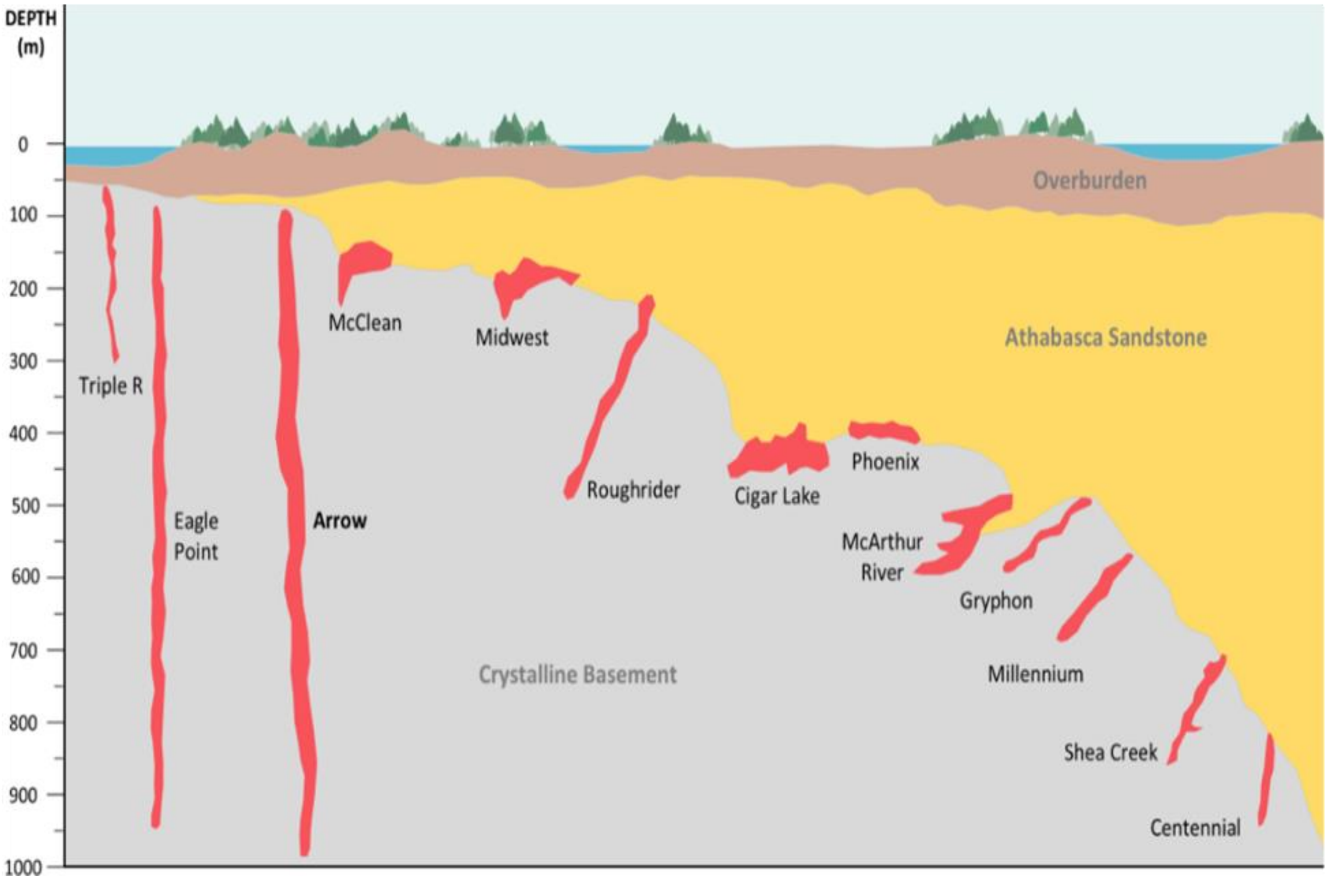


Features and examples of unconfomity associated-uranium deposits in the eastern part of the Athabasca Basin
(A) Generalized geological features of ingress- (monometallic) and egress (polymetallic) type deposits
(B) Egress type deposit - Cigar Lake
(C) Ingress type deposit - Eagle Point

NexGen Energy Ltd.
Rook I Property - Arrow Deposit
Northern Saskatchewan, Canada
General Geological Setting of Unconformity Associated Uranium Mineralization

December 2020 ***Source: after Jefferson et al., 2007.***

Figure 8-2: Athabasca Basin Deposit Setting



9.0 EXPLORATION

Since acquiring the Project property in December 2012, NexGen has carried out exploration activities, such as the following.

- Ground gravity surveys.
- Ground resistivity and DCIP surveys.
- Airborne magnetic-radiometric-VLF survey.
- Airborne VTEM survey.
- Airborne ZTEM survey.
- Airborne gravity survey.
- Radon-in-water geochemical survey.
- Ground radiometric and boulder prospecting program.

Diamond drilling programs have also helped to test several targets on the property, which is what resulted in the discovery of the Arrow Deposit in drill hole AR-14-01 (formerly known as RK-14-21) in February 2014.

9.1 Grids and Surveys

The collar locations of drill holes are spotted and surveyed by differential base station global positioning system (GPS) using the UTM Zone 12N NAD83 reference datum.

9.2 Geologic Mapping and Boulder Prospecting

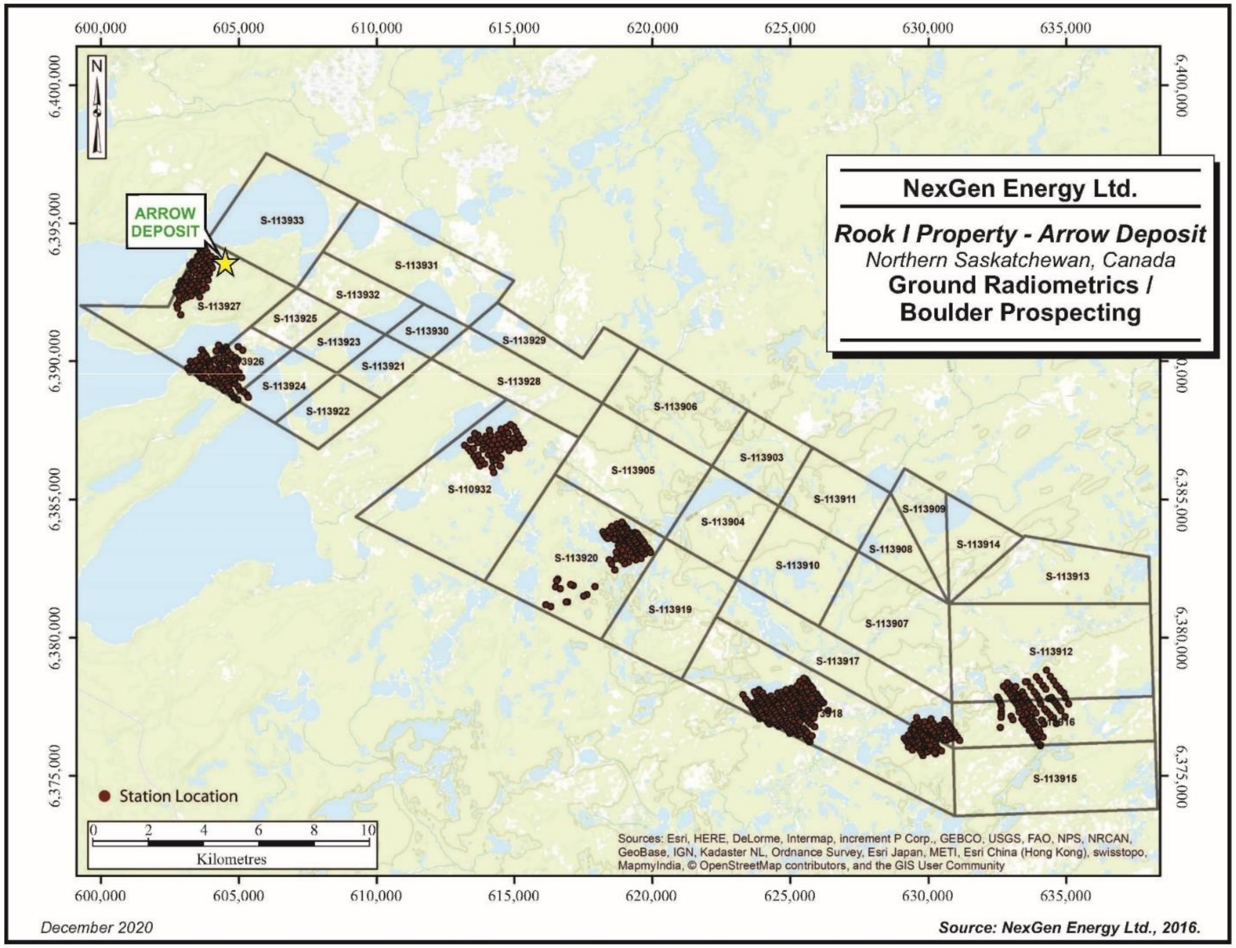
There is limited basement outcrop in the project area. Therefore, geological mapping of outcrops has not been used as a primary exploration tool.

In 2014, NexGen conducted a ground radiometric and boulder prospecting program to investigate many of the radiometric anomalies identified in 2013 by Goldak Airborne Surveys (Goldak) (see Figure 9-1).

Radioactivity was measured at 698 stations, where most of the boulders were Athabasca Group sandstones. Rare basement boulders were measured, and only two outcrops were observed. Where boulders were not present, background radioactivity was measured every 50 m along survey lines spaced 200 m apart.

Several anomalously irregular radioactive boulders were discovered; however, in each case, spectrometer analyses showed the radioactivity to be sourced from thorium. No samples were assayed.

Figure 9-1: Ground Radiometric / Boulder Prospecting



9.3 Geochemical Surveys

Due to the significant glacial-derived cover at the project site, surface geochemical sampling has not been used as a primary exploration tool.

In 2015, radon-in-water surveys were conducted by RadonEx Exploration Management Ltd. along parts of Patterson, Beet, and Naomi Lakes (Charlton, 2015) (see Figure 9-2 and Figure 9-3).

The surveys included collecting 1,942 near-bottom water samples. Radon was measured using electret ionization chamber technology after water samples were collected and stored in glass jars. Samples were spaced 25 m apart on lines typically spaced 200 m apart. The results showed multiple areas with anomalous radon gas concentrations.

Figure 9-2: 2015 Radon in Water Results – Patterson Lake

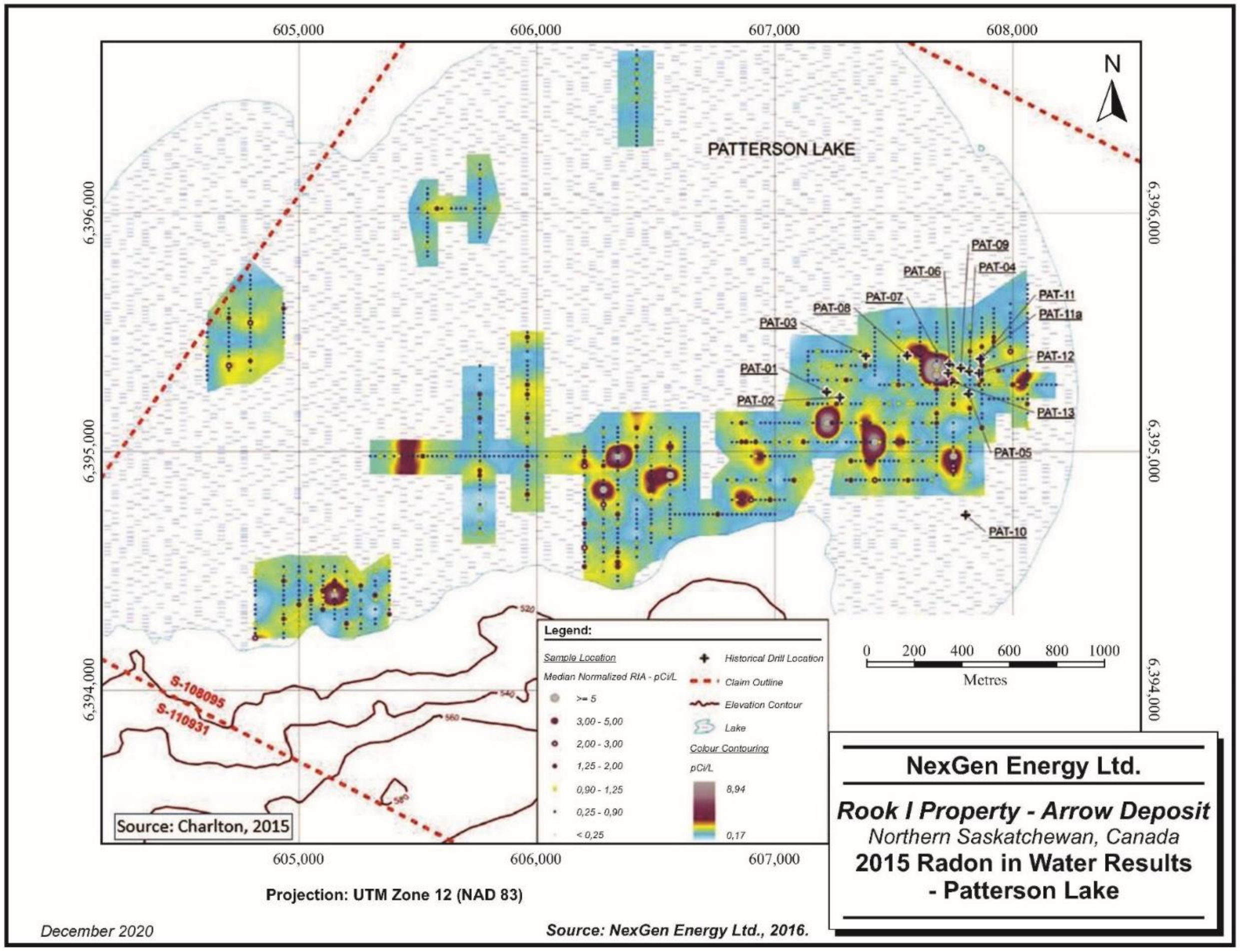
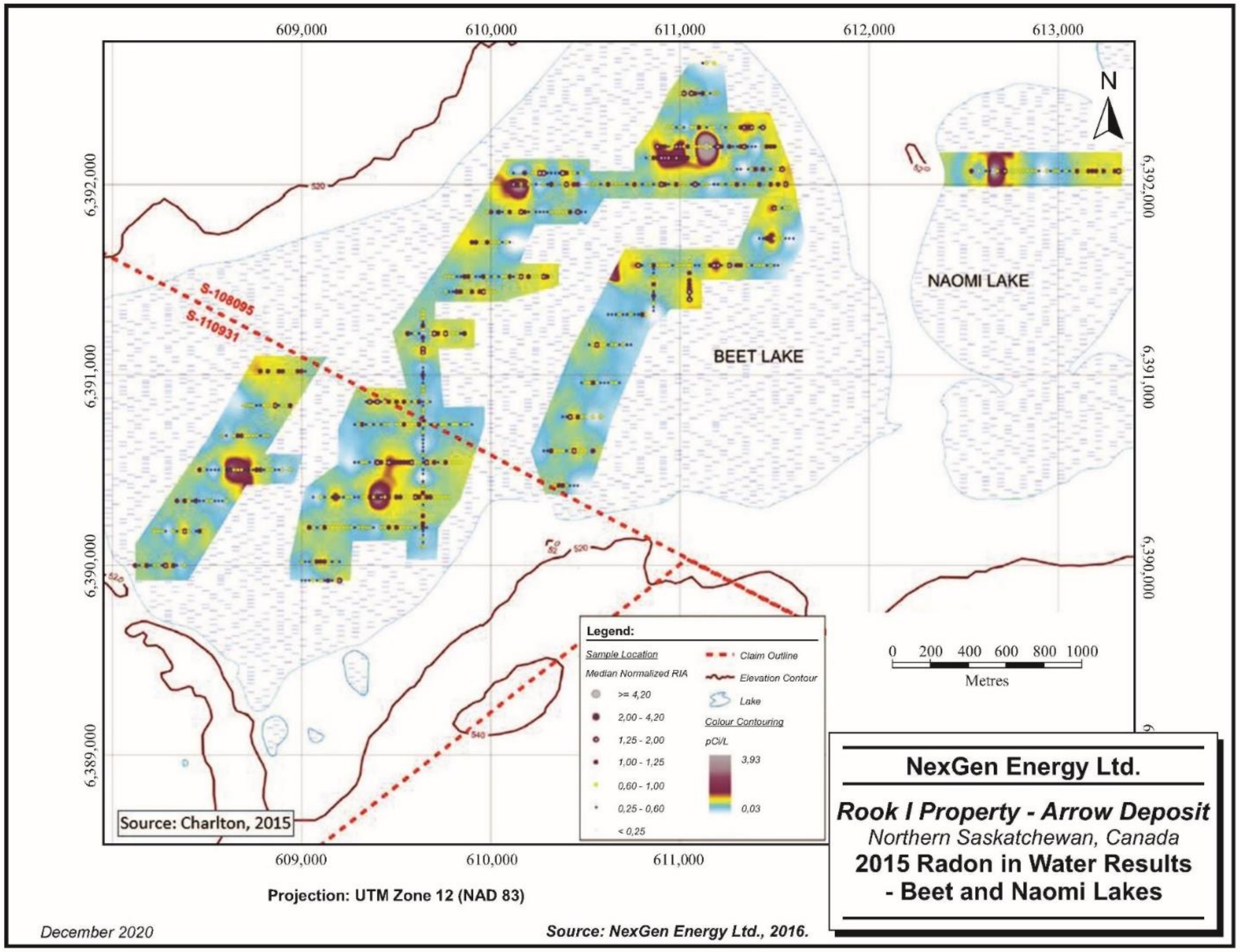


Figure 9-3: 2015 Radon in Water Results – Beet and Naomi Lakes



9.4 Geophysics

9.4.1 Ground Geophysical Surveys

Gravity

In the fall of 2013 and the winter of 2015, NexGen retained Discovery Geophysics International Inc. (Discovery) and MWH Geo-Surveys Ltd. (MWH) to complete ground gravity surveys over much of the western half of the Property (Koch, 2015; Koch, 2013) (see Figure 9-4).

In total, 12,867 gravity measurements were acquired within the survey areas, including a number of duplicate measurements acquired in areas surveyed by Mega prior to NexGen's acquisition of the Project property. Stations were spaced 50 m apart along lines spaced at 200 m, and were located by differential GPS. Features identified from the survey results are interpreted to be larger regional trends upon which smaller, more localized features occur.

These smaller features, exhibiting both relatively high- and low-gravity responses, may be the result of hydrothermal alteration in both sandstones and basement rocks. The discovery of the Arrow Deposit was partially the result of drill testing a circular gravity anomaly (gravity low) with an approximate diameter of 1 km.

It is thought that the gravity low present at the Arrow Deposit is the result of clay alteration (i.e., illite / dravite / sudoite) of the basement rocks within and adjacent to the Arrow Deposit.

DC Resistivity

In 2013, NexGen completed a DC resistivity survey over a small area on the westernmost portion of the Project property (Koch, 2013b) (see Figure 9-5). This survey was completed by Discovery on 200 m-spaced grid lines via a pole-dipole array with stations spaced at 50 m along lines.

The estimated depth penetration based on the array parameters used (i.e., n=1 through 8, and 0.5 through 7.5) was approximately 225 m. The survey successfully identified several prospective basement-hosted EM anomalies. It also identified a near-surface, flat-lying conductive horizon interpreted to be carbonaceous Manville Group rocks overlying the basement.

Figure 9-4: Residual Gravity

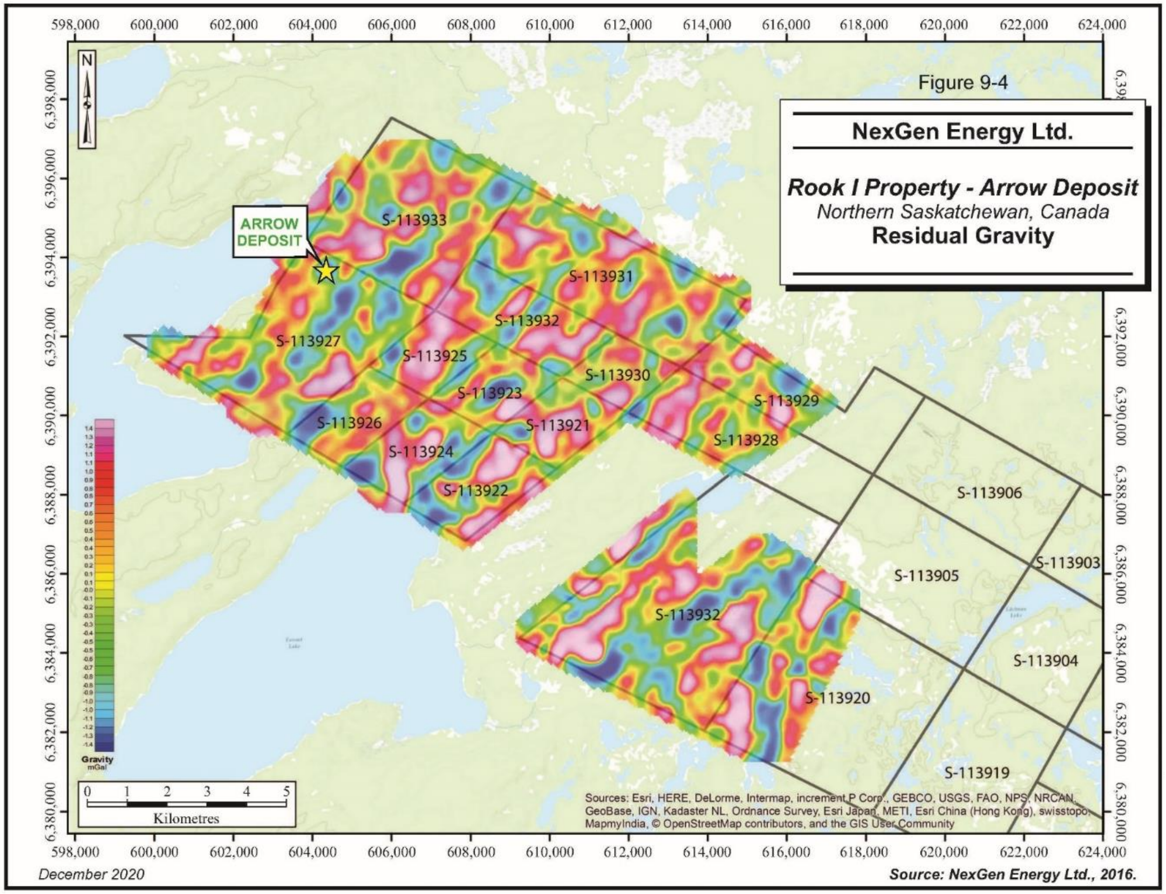


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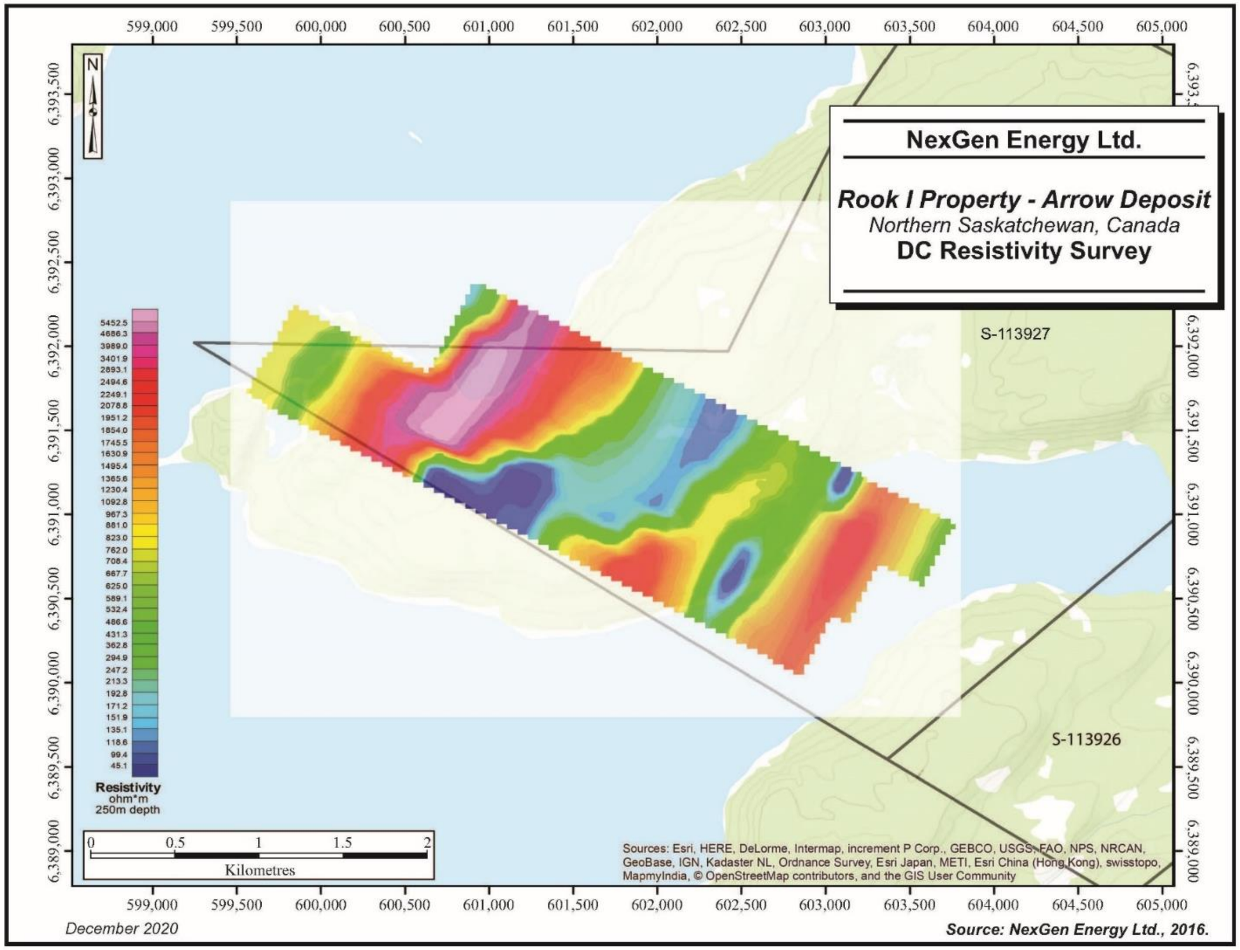
Figure 9-5: DC Resistivity Survey



3D DC Resistivity

In 2016 and 2017, NexGen retained Dias Geophysical to complete two 3D DCIP resistivity surveys of the Project property. The first survey was completed in the fall of 2016. It occurred over the Arrow Deposit, located within claim block S-113927 (Rudd and Lepitzki, 2017). The initial survey consisted of a 1.44 km by 1.44 km grid array, with 13 electrodes by 13 electrodes at 120 m inter-electrode spacing.

A resistivity anomaly was found adjacent to the Arrow Deposit, and a second anomaly was discovered approximately 400 m southwest of the Arrow Deposit along a parallel conductor. This second anomaly was tested in the spring of 2017, leading to the discovery of the South Arrow Discovery (Rudd and Thibaud, 2017).

Due to the discovery of the South Arrow Discovery—which was drilled on the basis of multiple geophysical indicators—an expanded 3D resistivity survey was completed over claim S-113927. The second expanded survey was completed in the fall of 2017. It consisted of an additional 1.56 km by 1.2 km grid, with 14 electrodes by 12 electrodes at 120 m inter-electrode spacing.

In both surveys, once the electrodes were placed, differential GPS coordinates were determined for each station. It was determined that the 3D resistivity completed on the property had a penetration depth of approximately 500 m below surface and indicated the presence of a gabbroic package.

9.4.2 Airborne Geophysical Surveys

Magnetic-Radiometric-VLF

In 2013, Goldak was retained by NexGen to fly a high resolution magnetic radiometric gradiometer – VLF EM survey over the entire NexGen Rook I Project property. The survey included 3,491 line-km flown on lines spaced 200 m apart (Goldak, 2013).

VLF data acquired as part of the survey has confirmed the widespread presence of basement structures on the property. Magnetic data acquired suggested highly variable geology on the property, and suggested that the property has a complex geological history. Radiometric data acquired showed a number of surficial radiometric anomalies (see Figure 9-6).

VTEM

In 2014, Aeroquest Airborne (now Geotech Ltd. [Geotech]), was retained by NexGen to fly a VTEM survey over a portion of the Project property (Pendrigh and Witherly, 2015) (see Figure 9-7). The survey included 793 line-km on lines spaced 100 m apart. Magnetic data was also collected in tandem with EM data.

Figure 9-6: Radiometrics Survey

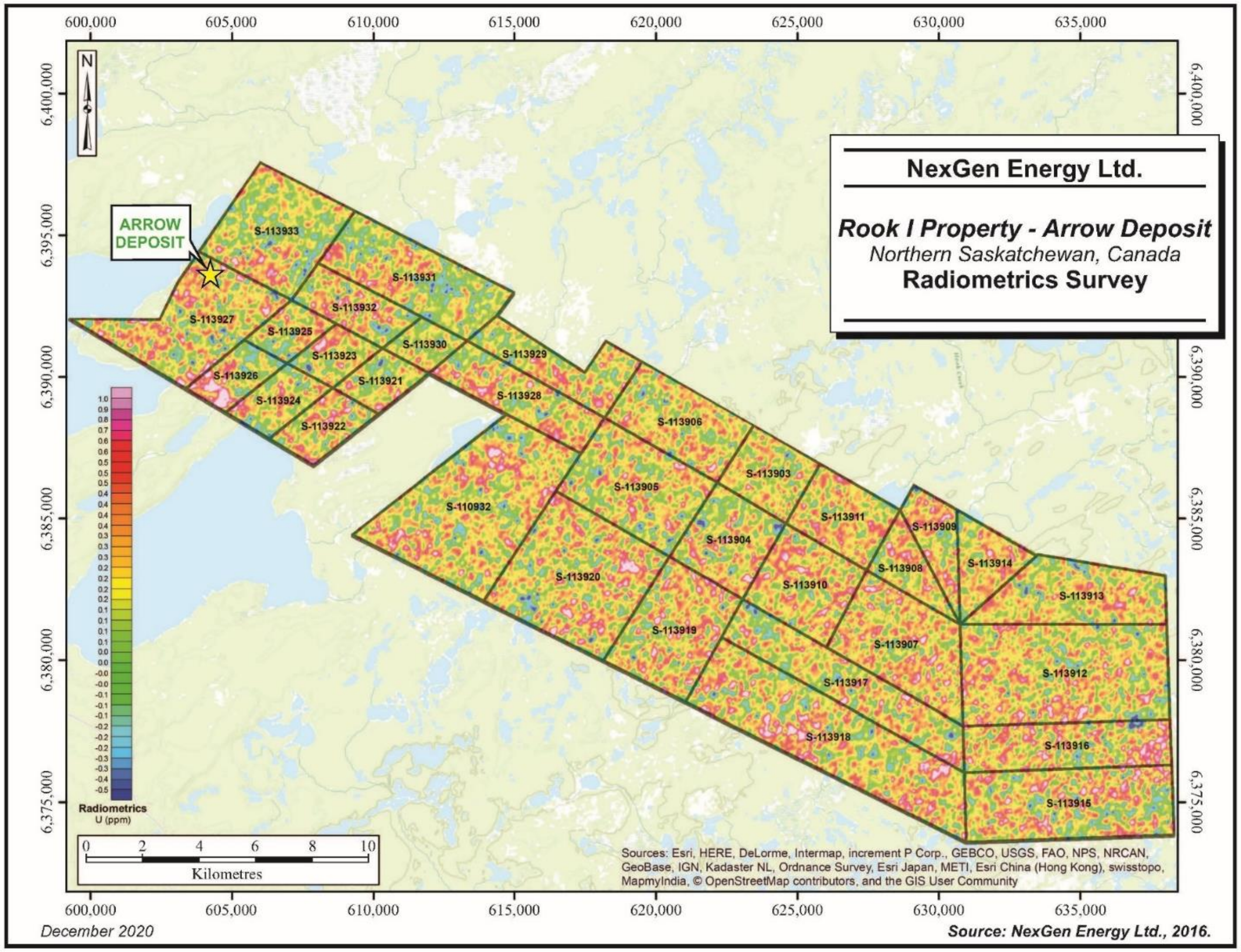
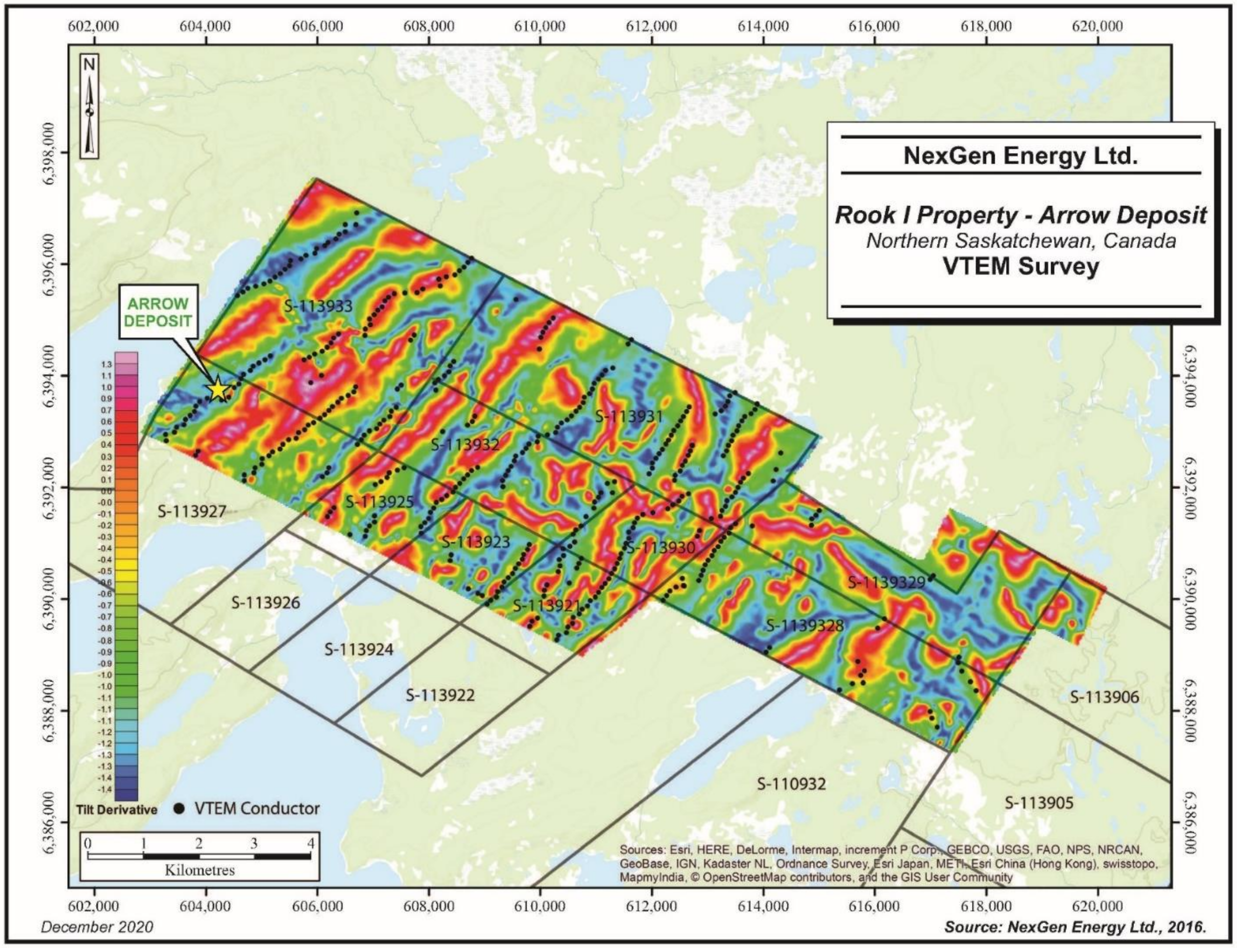


Figure 9-7: VTEM Survey



The results showed a number of northeast-trending EM conductors, most of which remain untested by drilling at the time of this report. Additionally, the acquired EM data allowed for more precise interpretation of the conductors that host the Arrow Deposit, as this survey was both higher powered and flown at closer line spacing than any previous airborne EM survey completed in the area by past operators.

ZTEM

In 2016, Geotech was retained by NexGen to carry out a ZTEM survey of a portion of the Project property (Pendrigh and Witherly, 2017). The survey was flown parallel to the Patterson conductive corridor, and it included 584 line-km on lines spaced 100 m apart.

Due to the position of the survey area of interest along the corridor, a non-standard flight orientation parallel to the primary geological strike was chosen. This is normally not advised for active source technologies such as VTEM; however, with ZTEM, recording the two orthogonal components allows for effective mapping of fields along both survey lines and tie lines.

The results of the survey showed that a broad corridor of low resistivity traverses the property from the southwest to the northeast (see Figure 9-8). The Arrow Deposit occurs within this corridor.

Gravity

In 2016, CGG Canada Services Ltd. was retained by NexGen to fly a HeliFalcon gravity survey of the Patterson conductive trend (Pendrigh and Witherly, 2017). The survey included 255 line-km on lines spaced 200 m apart, and oriented northeast-southwest.

Similar to the ground gravity survey, features identified by the survey results were interpreted to be larger regional trends upon which smaller, more localized features occur (see Figure 9-9). These smaller features, which show both relatively high- and low-gravity responses, could be the result of hydrothermal alteration in both sandstones and basement rocks.

The 2016 airborne survey positively identified the gravity anomaly associated with the Arrow Deposit and correlated very well with the ground gravity survey previously completed by NexGen. This indicates that airborne gravity will be an effective regional exploration tool when searching for basement-hosted uranium mineralization in the Athabasca Basin.

Figure 9-8: ZTEM Survey

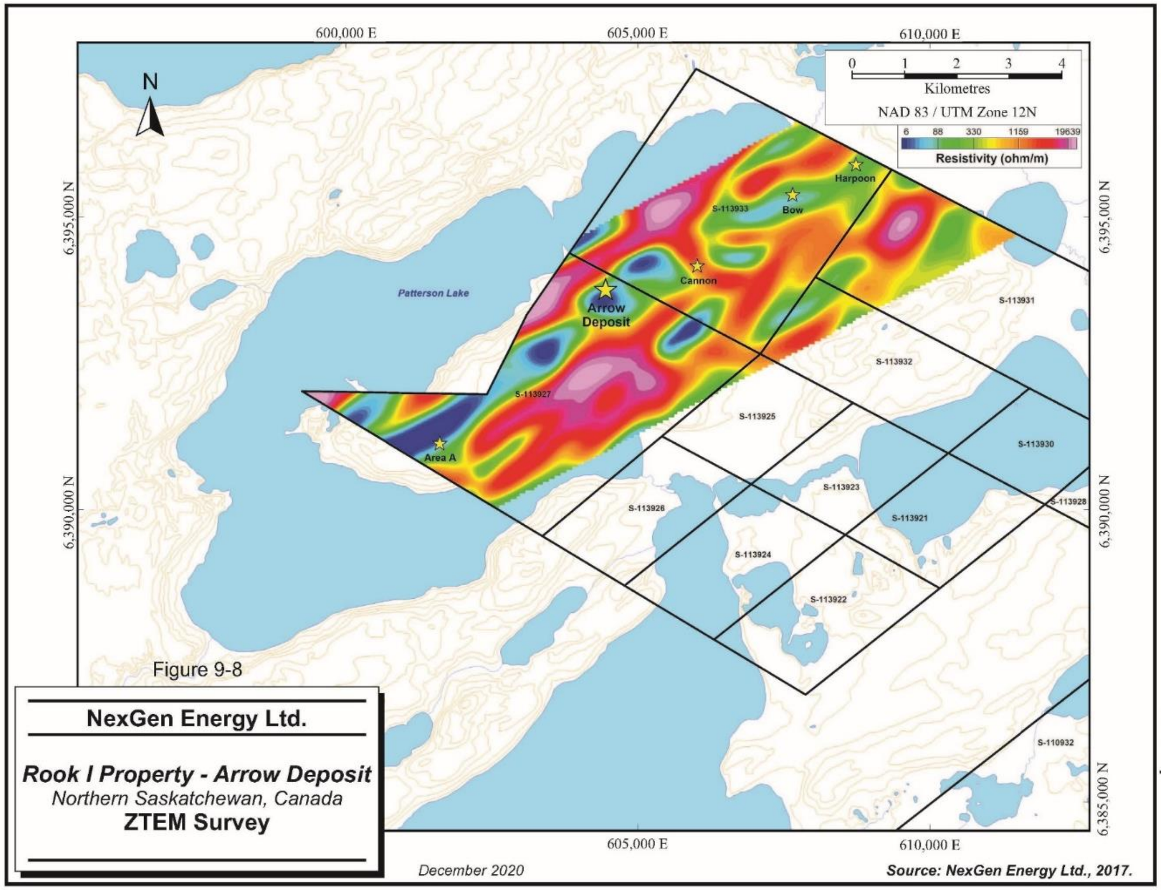


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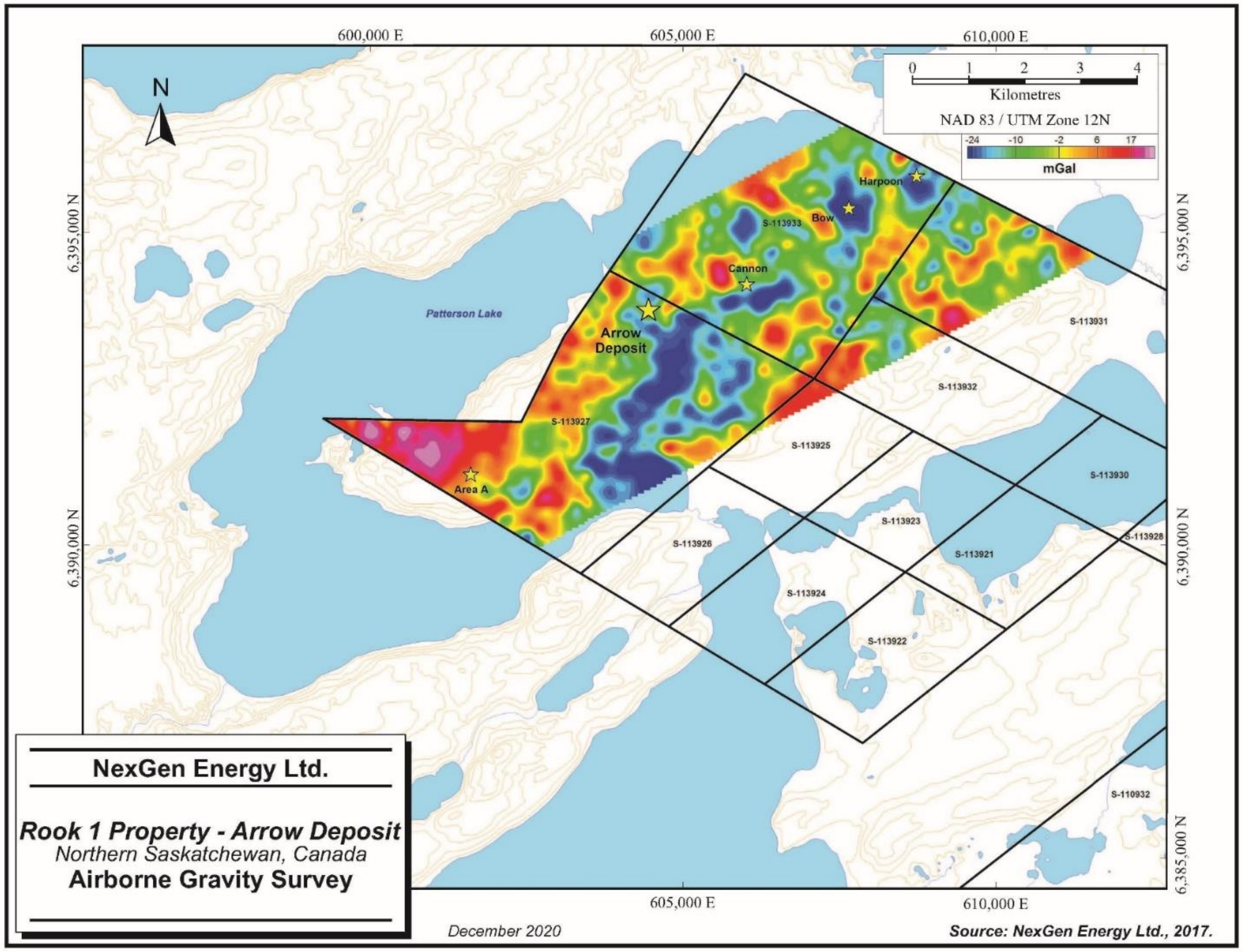
Figure 9-9: Airborne Gravity Survey



9.5 Exploration Potential

Several uranium-anomalous occurrences have been identified within the project area, as discussed in Section 7.0. Geophysical surveys have indicated several geophysical anomalies that may warrant further exploration.

At the Arrow Deposit, the A4 and A5 Mineral Resource delineation has mostly been achieved through secondary and tertiary targeting. This has left the shear zones completely untested down-dip of several deep mineralized intersections such as the following.

- **AR-15-048c3** – 10 m at 0.20% U_3O_8 from 981.5 to 991.5 m.
- **AR-16-081c2** – 1 m at 11.04% U_3O_8 from 901.5 to 902.5 m, and 1.5 m at 2.24% U_3O_8 from 905.5 to 907.0 m.
- **AR-16-092c2** – 12.5 m at 0.53% U_3O_8 from 997.5 to 1010.0 m.

Intersections are situated along an apparent cross-cutting north to north-east trend of extended mineralization across the deposit that contains the widest intercepts at the Arrow Deposit, the HG domains, and some of the deepest intersections.

This area consists of the northeast-southwest and north-northeast-south-southwest oblique shear bends and flexures that connect the major shear zones, as detailed in the structural analysis of the Arrow Deposit. RPA is of the opinion that there is potential to expand upon the Arrow Deposit at depth.

10.0 DRILLING

As of the effective date of this report, NexGen and previous owners of the Rook I property have completed 754 holes totalling 380,051 m. From 2013 to the effective date of this report, NexGen has completed 716 holes, totalling 374,917 m of drilling.

One of the holes drilled by NexGen during the company's Summer 2018 program was a designated training hole located 2.34 km southwest of the Arrow Deposit. The purpose of this hole was to use it to train onsite personnel regarding the safety and protocols required for working on and around drill rigs. This hole has not been considered in the Mineral Resource estimate.

All drilling completed in the project area is summarized in Table 10-1. Locations of drill collars for the NexGen programs that took place between 2013 and 2019 are shown in Figure 10-1.

Diamond drilling on the property is the principal method of exploration and delineation of uranium mineralization after initial geophysical surveys. Drilling can generally be conducted year-round.

Sample acquisition, preparation, security, and analysis remained relatively unchanged throughout all drill programs, as is discussed in greater detail in Section 11.0. Deposit dimensions are listed in Section 7.0.

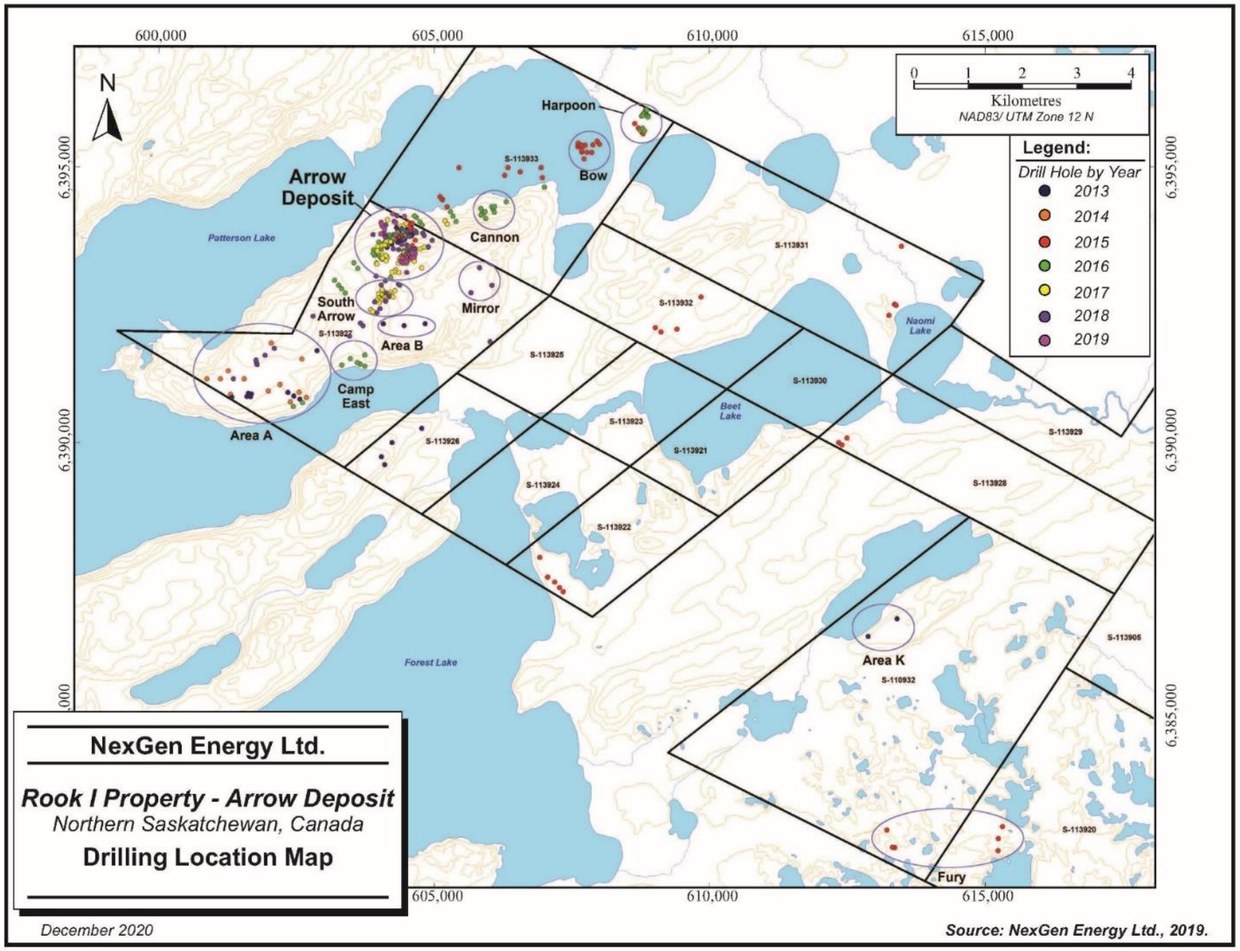
Table 10-1: Drilling Programs

Year	Season	Target Area	Company	Contractor	No. of Holes	Metres Drilled (m)
1977	Winter	SW-2	Kerr – SMDC JV	Bradley Bros.	1	124
1977 Total	–	–	–	–	1	124
1978	Winter	SW-2	Canadian Occidental Petroleum Ltd.	Canadian Longyear	2	290
			Hudson Bay Exploration and Development Co. Ltd.	Midwest Drilling	1	91
1978 Total	–	–	–	–	3	381
1979	Winter	SW-2	Canadian Occidental Petroleum Ltd.	Canadian Longyear	7	800
1979 Total	–	–	–	–	7	800
1980	Winter	PAT	Saskatchewan Mining Development Corporation	DW Coates Enterprises	6	746
		SW-2	Canadian Occidental Petroleum Ltd.	Canadian Longyear	11	1,764
1980 Total	–	–	–		17	2,510
1982	Winter	PAT	Saskatchewan Mining Development Corporation	Midwest Drilling	8	1,070
		SW-2	Hudson Bay Exploration and Development Co. Ltd.	Midwest Drilling	2	248
1982 Total	–	–	–	–	10	1,319
2013	Fall	Area A	NexGen Energy Ltd.	Guardian Drilling	13	3,029
2013 Total	–	–	–	–	13	3,029
2014	Winter	Arrow	NexGen Energy Ltd.	Aggressive Drilling	8	4,642
		Area A	NexGen Energy Ltd.	Aggressive Drilling	6	1,837
		Dagger	NexGen Energy Ltd.	Aggressive Drilling	3	963
	Summer	Arrow	NexGen Energy Ltd.	Aggressive Drilling	26	16,094

Year	Season	Target Area	Company	Contractor	No. of Holes	Metres Drilled (m)
		Area A	NexGen Energy Ltd.	Aggressive Drilling	3	885
		Area B	NexGen Energy Ltd.	Aggressive Drilling	3	936
		Dagger	NexGen Energy Ltd.	Aggressive Drilling	1	413
		K	NexGen Energy Ltd.	Aggressive Drilling	2	558
2014 Total	–	-	–	–	52	26,328
2015	Winter	Arrow	NexGen Energy Ltd.	Aggressive Drilling	24	12,550
		Bow	NexGen Energy Ltd.	Aggressive Drilling	14	5,185
		Fury	NexGen Energy Ltd.	Aggressive Drilling	6	1,357
		North Patterson	NexGen Energy Ltd.	Aggressive Drilling	10	2,473
	Summer	Arrow	NexGen Energy Ltd.	Aggressive Drilling	40	26,366
		Derkson Trend	NexGen Energy Ltd.	Aggressive Drilling	16	4,670
		NE Bow	NexGen Energy Ltd.	Aggressive Drilling	5	1,974
2015 Total	–	–	–	–	115	54,574
2016	Winter/Spring	Arrow	NexGen Energy Ltd.	Aggressive Drilling	71	37,240
		Cannon	NexGen Energy Ltd.	Aggressive Drilling	11	4,229
		NE Extension	NexGen Energy Ltd.	Aggressive Drilling	7	2,721
		North Patterson	NexGen Energy Ltd.	Aggressive Drilling	1	408
	Summer	Arrow	NexGen Energy Ltd.	Aggressive Drilling	53	37,598
		Arrow Trend	NexGen Energy Ltd.	Aggressive Drilling	4	3,546
		Camp East	NexGen Energy Ltd.	Aggressive Drilling	6	3,116
		Camp West	NexGen Energy Ltd.	Aggressive Drilling	2	850
		Harpoon	NexGen Energy Ltd.	Aggressive Drilling	20	7,285
2016 Total	–	–	–	–	175	96,993
2017	Winter	Arrow	NexGen Energy Ltd.	Aggressive Drilling	56	34,271

Year	Season	Target Area	Company	Contractor	No. of Holes	Metres Drilled (m)
		South Arrow	NexGen Energy Ltd.	Aggressive Drilling	2	1,792
		Arrow Trend	NexGen Energy Ltd.	Aggressive Drilling	1	994
		NE Extension	NexGen Energy Ltd.	Aggressive Drilling	2	1,628
		SE Extension	NexGen Energy Ltd.	Aggressive Drilling	3	2,085
	Summer	Arrow	NexGen Energy Ltd.	Aggressive Drilling	51	31,758
		South Arrow	NexGen Energy Ltd.	Aggressive Drilling	31	13,023
2017 Total	–	–	–	–	146	85,549
2018	Winter	Arrow	NexGen Energy Ltd.	Aggressive Drilling	32	19,089
		South Arrow	NexGen Energy Ltd.	Aggressive Drilling	12	5,912
		Area A	NexGen Energy Ltd.	Aggressive Drilling	7	3,437
		Mirror	NexGen Energy Ltd.	Aggressive Drilling	3	1,770
	Summer	Arrow	NexGen Energy Ltd.	Aggressive Drilling	29	20,482
		Training	NexGen Energy Ltd.	Aggressive Drilling	1	475
2018 Total	–	–	–	–	84	51,165
2019	Winter	Arrow	NexGen Energy Ltd.	Aggressive Drilling	131	57,279
2019 Total	–	–	–	–	131	57,279
Grand Total	–	–	–	–	754	380,051

Figure 10-1: Drilling Location Map



10.1 Drilling Methods

All previous NexGen drilling relied on core methods. NexGen has historically retained two contractors to complete drilling: Guardian Drilling Corp. (Guardian) in 2013, and Aggressive Drilling Ltd. (Aggressive) from 2014 to 2019. Core has been drilled predominantly at an NQ diameter (i.e., 47.6 mm), except for geotechnical holes which were drilled at an HQ diameter (i.e., 63.6 mm), and AQ (i.e., 27 mm) and BQ (i.e., 36.5 mm) diameters when directional drilling technology was being used.

Directional core drilling technology was used from 2015 to 2019 to allow for precise controlled deviation of drill holes, and to make it possible to drill multiple branches from one main pilot hole. This drilling method allows for precise pierce point control (within three metres) and it saves drilling metres. Directional drilling was completed by Tech Directional Services Ltd. (Tech) from 2015 to 2018, and International Directional Services LLC (International) in 2019.

All holes that have been drilled within the project area have been cemented from the bottom of the hole to approximately thirty metres below the bottom of the casing.

10.2 NexGen Drill Programs

The following sections discuss the methods and results of the drill programs completed up to the effective date of this report.

All drill holes are named according to naming conventions. All drill hole names begin with one of the following prefixes that describes where the drill hole is located.

- "AR", denoting "Arrow"
- "GAR", denoting "Geotechnical Arrow"
- "BO", denoting "Bow"
- "CN", denoting "Cannon"
- "HP", denoting "Harpoon"
- "RK", denoting "Rook I"

These prefixes are followed by two digits representing the year and the number of the drill hole. For example, RK-13-01 is the first hole that was drilled on the Rook I property in 2013.

10.2.1 Fall 2013 Drill Program

From August 2013 to October 2013, Guardian completed 3,029 m of diamond drilling in 13 drill holes. Guardian used two rigs to complete this work, and their drilling was supported by a helicopter for most of the program. The purpose of drilling these holes was to test targets that were identified in the 2013 DC resistivity survey in Area A.

Drill holes RK-13-01, RK-13-02, and RK-13-03 targeted a narrow resistivity low on the eastern portion of the grid. The low was interpreted to be caused by a graphitic quartz-feldspar gneiss horizon.

Drill holes RK-13-04, RK-13-05, RK-13-07, RK-13-09, RK-13-11, and RK-13-13 targeted the east side of a broad resistivity low; drill holes RK-13-06, RK-13-08, RK-13-10, and RK-13-12 tested the west side of the same low. The broad low is interpreted as a thick sequence of quartz-feldspar-garnet-biotite gneisses, with variable graphite content.

Anomalous radioactivity was intersected in RK-13-05; it returned as 330 ppm U_3O_8 over four metres. Visible uraninite was identified within a strongly hematite-altered breccia. Mineralization was observed to occur within a 29 m wide shear zone marked by faults, fractures, a variety of veins, and breccias. The host rocks in RK-13-05 are garnetiferous quartz-plagioclase-biotite gneiss with minor graphite. Follow-up drilling failed to intersect mineralization.

10.2.2 Winter 2014 Drill Program

From January 2014 to March 2014, Aggressive completed 7,442 m of diamond drilling in 17 drill holes. The purpose of the drill program was to follow up on previously intersected uranium mineralization in hole RK-13-05, and test a combination of airborne magnetic, electromagnetic (EM), and ground gravity geophysical anomalies that were considered priority targets for uranium mineralization.

Three areas were targeted during the Winter 2014 exploration drill season: Area A, Dagger (Area D), and Arrow (see Figure 10-1). Anomalous radioactivity was intersected in drill holes AR-14-01 (formerly RK-14-21) through AR-14-08 (formerly RK-14-30) at the Arrow Deposit.

Subsequent assay results confirmed the presence of significant uranium concentrations. These drill holes represent the first discovery of significant mineralization at the Arrow Deposit.

10.2.3 Summer 2014 Drill Program

From May 2014 to September 2014, Aggressive completed 18,886 m of diamond drilling in 35 drill holes. Aggressive used three drill rigs to complete this work. The drill holes were primarily designed to follow up on uranium mineralization intersected at the Arrow Deposit during the Winter 2014 drill program. In addition, regional holes tested a combination of magnetic, EM, and gravity targets in four areas on the property that included Area A, Area B, Area D (Dagger), and Area K (see Figure 10-1).

The Summer 2014 program was successful, and extensive uranium mineralization was intersected at the Arrow Deposit in several holes including AR-14-15 (3.42% U_3O_8 over 22.35 m and 1.52% U_3O_8 over 32.0 m), and AR-14-30 (10.17% U_3O_8 over 20.0 m and 7.54% U_3O_8 over 63.5 m).

A reinterpretation of the structural setting helped to identify three main mineralized shear zones: the A1, A2, and A3 shears. Both AR-14-15 and AR-14-30 were the first holes drilled through what is now the HG domain of the A2 shear.

10.2.4 Winter 2015 Drill Program

From January 2015 through April 2015, Aggressive completed 21,565 m of diamond drilling in 54 drill holes. Aggressive used four drill rigs to complete this work. The holes were primarily designed to expand the mineralization at the Arrow Deposit.

Regional holes were used to continue testing on a combination of magnetic, EM, and gravity targets at the Bow and Fury areas (see Figure 10-1). At the Arrow Deposit, drilling continued to intersect strong mineralization. Results were highlighted in AR-15-44b, which intersected 11.55% U_3O_8 over 56.5 m, including 20.0 m at 20.68% U_3O_8 and 1.0 m at 70.0% U_3O_8 in the HG domain of the A2 shear.

A new zone of uranium mineralization was discovered in the Bow area (now referred to as the Bow occurrence). The hole with the most uranium mineralization in this area to date has been BO-15-10. This hole intersected 0.20% U_3O_8 over 9.5 m. To date, 14 holes have been drilled at Bow.

10.2.5 Summer 2015 Drill Program

From June 2015 to October 2015, Aggressive completed 33,010 m of diamond drilling in 61 drill holes on the property. Aggressive used five drill rigs to complete this work.

Directional core drilling technology was used to allow for precise controlled deviation of drill holes, and to make it possible to drill multiple branches from one main pilot hole. This drilling method allows for both precise pierce point control (within three metres) and it saves drilling metres. Directional drilling was completed by Tech.

The purpose of the holes drilled during the Summer 2015 program was to follow up on uranium mineralization intersected at the Arrow Deposit since the Winter 2014 program (see Figure 10-1). All holes at the Arrow Deposit intersected significant and often intense uranium mineralization. Results were highlighted in AR-15-62, which intersected 6.35% U_3O_8 over 124.0 m, including 10.00% U_3O_8 over 78.0 m. In addition, AR-15-49c2 intersected 12.01% U_3O_8 over 50.0 m, including 18.0 m at 20.55% U_3O_8.

Regional holes were drilled during the Summer 2015 program to test a combination of magnetic, EM, and gravity targets on the property. The targets included an on-land target area 750 m northeast of the Bow occurrence, and five on-land target areas within the Derkson conductor corridor in the area of Beet Lake. Highly anomalous uranium concentrations were intersected in one hole in the Bow occurrence area.

RK-15-69 encountered 0.05% U_3O_8 over 2.5 m. Drill hole RK-15-69 was subsequently renamed HP-16-03 in concert after the discovery of the Harpoon occurrence during the Summer 2016 drill program, as described in greater detail in Section 10.2.7 of this report.

10.2.6 Winter / Spring 2016 Drill Program

From January 2016 to 26 June 2016, Aggressive completed 44,598 m of diamond drilling in 90 drill holes on the property. Aggressive used six drill rigs to complete this work, as well as directional core drilling technology to delineate and expand on the Arrow Deposit. During the Winter / Spring 2016 drill program, RPA published an initial Inferred Mineral Resource estimate for the Arrow Deposit (RPA, 2016).

Drill holes of the Winter / Spring 2016 program were primarily designed to infill the Arrow Deposit in support of an Indicated Mineral Resource classification in the A2-HG domain, and to materially expand the footprint of mineralization in support of an expanded Inferred Mineral Resource (see Figure 10-1).

Prior to the Winter / Spring 2016 program, drilling at the Arrow Deposit was largely completed from northwest to southeast. However, during the Winter / Spring 2016 program, seven infill holes were drilled in a scissor orientation from southeast to northwest to verify the near vertical dip of the mineralization. Scissor drilling verified both the near vertical dip of the mineralization and the varying thicknesses of the Arrow Deposit resource domains.

Arrow Deposit results for the Winter / Spring 2016 program were highlighted by AR-16-63c2, which intersected 15.20% U_3O_8 over 42 m, and 12.99% U_3O_8 over 46.5 m. In addition, AR-16-76c1 intersected 11.29% U_3O_8 over 67.5 m, including 9.0 m at 51.35% U_3O_8.

Step-out drilling at the Arrow Deposit during the Winter / Spring 2016 program was successful, with two significant new areas of mineralization discovered. Firstly, HG uranium mineralization was identified in the A1 shear for the first time where scissor hole AR-16-84c1 intersected 2.13% U_3O_8 over 28.5 m, including 3.99% U_3O_8 over 11.0 m.

Secondly, uranium mineralization was intersected 180 m southwest of the Arrow Deposit where drill hole AR-16-90c3 intersected 8.09% U_3O_8 over 13.0 m, including 10.33% U_3O_8 over 10.0 m. Mineralization in this area occurs in the southwest extensions of the Arrow Deposit shears.

The highlight of regional drilling during the Winter / Spring 2016 drilling program was the discovery of a new area of uranium mineralization which has been named the Cannon occurrence. The Cannon occurrence was tested with eleven drill holes, three of which intersected narrow zones of LG uranium mineralization. The best hole, CN-16-06, intersected 0.06% U_3O_8 over one metre.

Continued regional drilling during the Winter / Spring 2016 program largely tested the interpreted extensions of the conductor hosting the Arrow Deposit (the Arrow conductor) to the northeast. Firstly, a four-hole fence tested the Arrow conductor 200 m northeast of the Arrow Deposit. Although no mineralization was intersected, prospective hydrothermal alteration and geological structures were encountered.

A three-hole fence was subsequently drilled 750 m northeast of the Arrow Deposit, targeting a break in the Arrow conductor. Again, no mineralization was intersected, however, prospective hydrothermal alteration and geological structures were identified. Additionally, one hole was drilled 2.5 km northeast of the Arrow Deposit to test another interpreted break in the Arrow conductor. No mineralization was intersected.

Two additional holes were drilled 650 m southwest of the Arrow Deposit to test a subtle gravity anomaly that is coincident with the Arrow conductor. Both holes intersected Arrow-like lithologies and prospective graphitic shear zones, however no mineralization was intersected.

10.2.7 Summer 2016 Drill Program

From 26 June 2020 to November 2016, Aggressive completed 52,395 m of diamond drilling in 85 drill holes on the property. Aggressive used seven drill rigs to complete this work, as well as directional core drilling technology to delineate and expand the Arrow Deposit.

Drill holes of the Summer 2016 program were primarily designed to both infill the Arrow Deposit in support of an Indicated Mineral Resource classification in the A2-HG domain and materially expand the footprint of mineralization in support of an expanded Inferred Mineral Resource.

During the Summer 2016 program, 35 of the 53 holes drilled at the Arrow Deposit were drilled in a scissor orientation from southeast to northwest. Scissor oriented drilling again verified both the near vertical dip of the mineralization and the thicknesses of the Arrow Deposit resource domains. Results from the Arrow Deposit for the Summer 2016 program are highlighted by scissor hole AR-16-98c2, which intersected 7.59% U_3O_8 over 73.5 m, including 51.40% U_3O_8 over 10.0 m. In addition, scissor hole AR-16-91c2 intersected 12.69% U_3O_8 over 40.5 m, including 25.0 m at 19.97% U_3O_8.

During the Summer 2016 program, the highlight of regional exploration drilling was the discovery of the Harpoon occurrence with drill hole HP-16-08. The hole intersected 3.89% U_3O_8 over 17.5 m, which continues to be the best intersection of mineralization to date at the Harpoon occurrence. In total, 20 holes were drilled at the Harpoon occurrence, to within 27 m of the northeast boundary of the property, during the Summer

2016 program. It is likely that the Harpoon occurrence crosses the property boundary to the northeast, where it may be continuous with the Spitfire occurrence, owned by a joint venture between Cameco (40%), Areva Inc. (40%), and Purepoint Uranium Group Inc. (Purepoint) (20%).

Regional exploration drilling was also conducted at three other target areas during the Summer 2016 program. Firstly, a large airborne ZTEM resistivity anomaly 1.1 km southwest of the Arrow Deposit was tested with a four-hole fence where encouraging clay alteration and graphitic shear zones were intersected.

Secondly, coincident gravity and VTEM anomalies were tested with two holes approximately three kilometres southwest of the Arrow Deposit. Finally, coincident gravity and VTEM anomalies were tested with six holes approximately 2.3 km south-southwest of the Arrow Deposit. In this area, informally referred to as the Camp East area due to the proximity to the Project camp, narrow intersections of weakly anomalous radioactivity were intersected in two drill holes. In addition, all six drill holes intersected extensive hydrothermal alteration.

10.2.8 Winter 2017 Drill Program

From January 2017 to May 2017, Aggressive completed 40,770 m of diamond drilling in 64 drill holes on the property. Aggressive used seven drill rigs to complete this work, as well as directional core drilling technology to delineate and expand the Arrow Deposit.

Objectives for the Winter 2017 drill program included expansion and delineation of the Arrow Deposit, as well as the testing of high priority regional exploration targets on the property. In total, 34,271 m of drilling was completed in 56 drill holes at the Arrow Deposit; with the remainder of drilling completed on regional drill targets.

Significant uranium mineralization was intersected in most of the Arrow Deposit holes, extensively expanding the footprint of uranium mineralization throughout the Arrow Deposit.

The Winter 2017 drill program resulted in several major developments at the Arrow Deposit. Most importantly, additional growth and infill of the A2-HG domain was accomplished. Eight drill holes were successfully drilled to either infill or expand the A2-HG domain during the Winter 2017 drill season. Drill hole AR-17-114c2 intersected 4.58% U_3O_8 over 33.0 m in the A2-HG domain. Additionally, the Winter 2017 program resulted in new mineralization being identified in the A1 through A5 shears.

A primary objective of the Winter 2017 program was to further delineate the A3 shear, including the A3-HG domain. Eleven drill holes from the Winter 2017 drill program were successfully drilled to test the A3-HG domain. Drill hole AR-17-136c2 intersected 9.58% U_3O_8 over 13.5 m in the A3-HG domain. This resulted in NexGen greatly expanding the extent of uranium mineralization within the target area.

NexGen continued to target the areas immediately southwest and northeast of the Arrow Deposit. In the southwest, NexGen intersected significant mineralization between the Arrow Deposit and the zone of mineralization 180 m southwest of the Arrow Deposit, resulting in the 180 m southwest zone being incorporated into the A3 and A4 shear zone models. The target areas northeast of the Arrow Deposit also returned favourable results, expanding the footprint of the mineralization in the A1 and A2 shear zones.

Two drill holes were completed within a regional target area—later named the South Arrow Discovery—located approximately 400 m south-southwest of the Arrow Deposit. These holes were collared to test an Arrow-parallel VTEM conductor associated with a 3D resistivity anomaly similar to that observed at the Arrow Deposit. Drill hole AR-17-150w1 (previously named RK-17-118w1) intersected 0.25 m of mineralization at 0.09% U_3O_8. A total of 1,792 m of drilling was completed in the South Arrow Discovery area.

Three holes totalling 2,085 m were drilled 450 m southeast of the Arrow Deposit in a regional target area (the Southeast Extension). These holes were collared to test a prominent VTEM conductor parallel to the Arrow Deposit. No uranium mineralization was intersected.

Two drill holes were completed within a regional target area, named the Northeast Extension, approximately 650 m northeast along strike of the Arrow Deposit for a total of 1,627.5 m. These holes were collared to test a VTEM conductor within a prominent ZTEM corridor along strike with the Arrow Deposit. An additional drill hole totalling 993.5 m was completed along the Arrow Trend, approximately 150 m northwest of the Arrow Deposit, designed to test a regional 3D DCIP resistivity target. No uranium mineralization was intersected.

10.2.9 Summer 2017 Drill Program

From July 2017 to November 2017, Aggressive completed 44,781 m of diamond drilling in 82 drill holes on the property. Aggressive used eight drill rigs to complete this work, as well as directional core drilling technology to delineate and expand the Arrow Deposit.

Objectives for the Summer 2017 drill program included expansion and delineation of the rapidly developing Arrow Deposit, as well as expansion of the recent South Arrow Discovery. NexGen also commenced analysis of geotechnical characteristics on the Arrow Deposit for the completion of a PFS.

The Summer 2017 drill program resulted in several major developments at the Arrow Deposit. Most importantly, the growth of the A3-HG domain was accomplished. The best uranium intersection drilled in the A3 shear at the Arrow Deposit during the Summer 2017 season was in AR-17-159c1, which intersected 26.5 m of mineralization at 10.6% U_3O_8. Additionally, the Summer 2017 program resulted in new mineralization being identified throughout the A1 to A5 shears.

A primary objective of the Summer 2017 program was to test the extent of mineralization northeast and southwest of the HG domains, which resulted in NexGen greatly expanding the extent of uranium mineralization within these target areas.

Thirty-one drill holes were completed at the South Arrow Discovery, located approximately 400 m south-southwest of the Arrow Deposit. The best continuous mineralized interval drilled at the South Arrow Discovery was in AR-17-166c1, which intersected 24.5 m of mineralization at 1.46% U_3O_8.

The mineralized footprint at the South Arrow Discovery has been traced over a strike length of approximately 290 m that extends from 110 m from surface to a depth of 550 m. A total of 13,023 m of drilling was completed at the South Arrow Discovery where NexGen believes there is high potential for the discovery of additional mineralization.

10.2.10 Winter 2018 Drill Program

From January 2018 to April 2018, Aggressive completed 30,208 m of diamond drilling in 54 drill holes on the property. Aggressive used eight drill rigs to complete this work.

The holes were primarily designed to expand the mineralization at the Arrow Deposit and in the South Arrow Discovery. Regional holes were designed to test a combination of magnetic, EM, and gravity targets along the Arrow conductor to the southwest of the Arrow Deposit (Area A) and the Mirror area to the southeast of the Arrow Deposit (see Figure 10-1).

At the Arrow Deposit, 19,089 m in 32 drill holes were completed, with drilling at the Arrow Deposit continuing to intersect strong mineralization. Regional drilling on the Mirror target area totalled 1,770 m in three drill holes. The Mirror target area is a conductor located 1.5 km southeast of the Arrow Deposit; it runs parallel to the Arrow conductor. Drilling at Mirror successfully intersected the targeted VTEM conductor, but did not encounter significant uranium mineralization.

All drilling along the Arrow conductor was situated approximately 2.5 km southwest and along strike from the Arrow Deposit, hosted within the same VTEM conductor. Arrow-type silicified quartz-feldspar-garnet-biotite gneiss was intersected throughout in all of the six holes drilled in the area for a total of 3,437 m. Moderate to intense sericitic alteration, similar to Arrow-type alteration found proximal to the Arrow Deposit was intersected in several of the drill holes.

Expansion drilling in the South Arrow Discovery totalled 5,912 m in 12 drill holes, and several holes intersected mineralization including intervals of HG uraninite, which was 175 m southwest of the main zone of mineralization in the South Arrow Discovery area.

10.2.11 Summer 2018 Drill Program

From June 2018 to November 2018, Aggressive completed 20,957 m of diamond drilling in 30 drill holes. Aggressive used four drill rigs to complete this work, as well as directional core drilling technology to delineate and expand the Arrow Deposit and assist with geotechnical drilling.

The purpose of the holes was to test for mineralization along the southwest and northeast peripheries of the Arrow Deposit, and analyze the geotechnical characteristics of the Arrow Deposit and associated areas of planned mine development. One regional hole was drilled to test a magnetic, EM, and gravity coincident target along an Arrow-parallel conductor.

At the Arrow Deposit, 20,482 m in 29 drill holes were completed where drilling continued to intersect mineralization. The Summer 2018 program was highlighted by the completion of three shaft pilot holes designed for the analysis of geotechnical characteristics in potential shaft development areas. One regional hole for a total of 474.5 m was drilled along an Arrow-parallel VTEM conductor approximately 2.3 km southeast of the Arrow Deposit. No uranium mineralization was intersected.

10.2.12 Winter 2019 Drill Program

From December 2018 to May 2019, Aggressive completed 57,279 m of diamond drilling in 131 drill holes. Aggressive used ten drill rigs to complete this work, as well as directional core drilling technology to delineate the Arrow Deposit and assist with geotechnical drilling. The rigs were operated by Aggressive Drilling, and the directional drilling was performed by International Directional Services LLC.

The holes were designed to increase confidence in mineralization continuity and upgrade a portion of the Arrow Deposit mineralization from Indicated classification to a Measured classification. Several holes were also designed to analyze the geotechnical characteristics of the Arrow Deposit and areas of planned mine development.

The Winter 2019 program resulted in the resource classification upgrade of large portions of Arrow Deposit mineralization from Indicated to Measured. Drill holes pierced the Arrow Deposit at relatively shallow angles (i.e., generally between -55° and -60° dip) with high precision. The continuity of HG mineralization was demonstrated, highlighted by AR-19-225c1, which intersected 11.36% U_3O_8 over 36.0 m, including 33.78% U_3O_8 over 12.0 m.

10.3 Drill Hole Surveying

The collar locations of drill holes were spotted and surveyed by differential base station GPS using the UTM Zone 12N NAD83 reference datum. Drilling was predominantly completed in both northwest and southeast directions, with drill holes at the Arrow

Deposit spaced approximately 12.5 m to 50 m apart based on directional drilling orientation.

The trajectory of all drill holes was determined during drilling with a Reflex instrument in single point mode, which measures the dip and azimuth at 30-m intervals. In more recent programs, an Axis Mining Technology north-seeking Champ Gyro was used to determine dip and azimuth at three-metre intervals through directional drilling intervals; this allowed for greater accuracy of the trajectory of the drill hole, particularly the vertical shaft pilot holes drilled in 2018.

Both immediately below casing and after completion, all holes at the Arrow Deposit were surveyed using a Stockholm Precision Tools north-seeking gyro, which measures the dip and azimuth continuously downhole. All holes on the property were cemented from the bottom of the hole to approximately 30 m below the drill casing, which was typically seated in the basement.

10.4 Drill Core Handling and Logging Procedures

At each drill site, core was removed from the core tube by the drill contractors, and placed directly into three-row NQ wooden core boxes in standard 1.5 m lengths (4.5 m total). Individual drill runs were identified with small wooden blocks, onto which the depth in metres was recorded. Diamond drill core was transported at the end of each drill shift to an enclosed core handling facility at NexGen's camp. The diamond drill core boxes were surveyed with a Radiation Solutions RS-120 scintillometer to determine if any boxes contained mineralization.

A threshold of 500 cps was used to determine mineralization for Arrow core, and 300 cps for any core that was from elsewhere on the property. All mineralized core boxes above the threshold, plus a box before and after the box containing mineralized core, were taken to designated areas for mineralized material for logging and sampling. All other core was moved to be processed in the logging areas designated for non-mineralized core.

Before the core was split for sampling, depth markers were checked, and core was carefully reconstructed, washed, and geotechnically logged for lithologies, alteration, structures, mineralization, and rock mass rating (RMR); resurveyed in detail with the scintillometer; marked for sampling; and photographed wet. Drill hole sampling for assay was guided by the observed geology and readings from a hand-held scintillometer.

Logging and sampling information was entered into a proprietary acQuire 4 database. Prior to December 2018, a Microsoft Access database template on a laptop computer was used, which was then integrated into the project master digital database on a daily basis.

10.5 Drill Core Recovery

Core recovery at the Arrow Deposit is excellent, allowing for representative samples to be taken and accurate analyses to be performed.

Mineralization in the Arrow Deposit is sub-vertical, and the true width is estimated to be from 30% to 50% of reported core lengths, based on information available at the time of this report.

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

In RPA's opinion the drilling, core handling, logging, and sampling procedures currently used meet or exceed industry standards, and are adequate for the purpose of Mineral Resource estimation.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Methods

Three types of drill core samples have been collected at the project site for geochemical analysis and uranium assay.

- One metre and 0.5-metre samples taken over intervals of elevated radioactivity, and one metre or two metres beyond radioactivity.
- Point samples taken at nominal spacings of five metres—or 50 m for infill holes—which is meant to be representative of the interval or of a particular rock unit.
- Composite samples in the Devonian and Athabasca sandstone units where one-centimetre-long pieces are taken and spaced throughout sample intervals ranging from one metre to 10 m long.

On-site sample preparation consists of geological technicians splitting cores under the supervision of geologists. One half of the core is placed in plastic sample bags pre-marked with the sample number, along with a sample number tag. The other half is returned to the core box and stored at the core storage area located near the logging facility on the project site.

The bags containing the split samples are then placed in lidded buckets to be transported by NexGen personnel to SRC Geoanalytical Laboratories in Saskatoon, Saskatchewan.

11.2 Density Determinations

NexGen personnel perform full core bulk density measurements on-site using standard laboratory techniques. In mineralized zones, average bulk density is measured using samples taken from mineralized zones at 2.5 m intervals, where possible (i.e., approximately 20% of all mineralized samples).

For density to be correlated with uranium grades across the data set, each density sample directly correlates with a sample sent to SRC for assay (i.e., downhole intervals are the same for density samples and assay samples).

Bulk density is used globally to convert volume to tonnage and—where bulk density is highly variable—may be used to weight block grade estimates. For instance, HG uranium deposits in the Athabasca Basin have bulk densities that commonly correlate with grade.

Bulk density also varies with clay alteration and in situ rock porosity, which can result in low bulk density values. When modelling HG uranium deposits, it is common to estimate bulk density values throughout the deposit and to weight grades by density, since small volumes of HG material contain large quantities of uranium oxide.

Bulk density is determined by NexGen with specific gravity (SG) measurements on drill core using the water immersion method according to the Archimedes principle, after the core has been sealed and shrink-wrapped in cellophane or dipped in wax. SG is calculated as follows.

$$weight\ in\ air/(weight\ in\ air - weight\ in\ water)$$

Under normal atmospheric conditions, SG (a unitless ratio) is equivalent to density in t/m^3.

A total of 5,850 bulk density measurements have been completed using drill core samples from the main mineralized zones within the Arrow Deposit and South Arrow Discovery. These samples represent different local major lithologic units, mineralization styles, and alteration types. Samples were collected from full core, which had been retained in the core box prior to splitting for sampling.

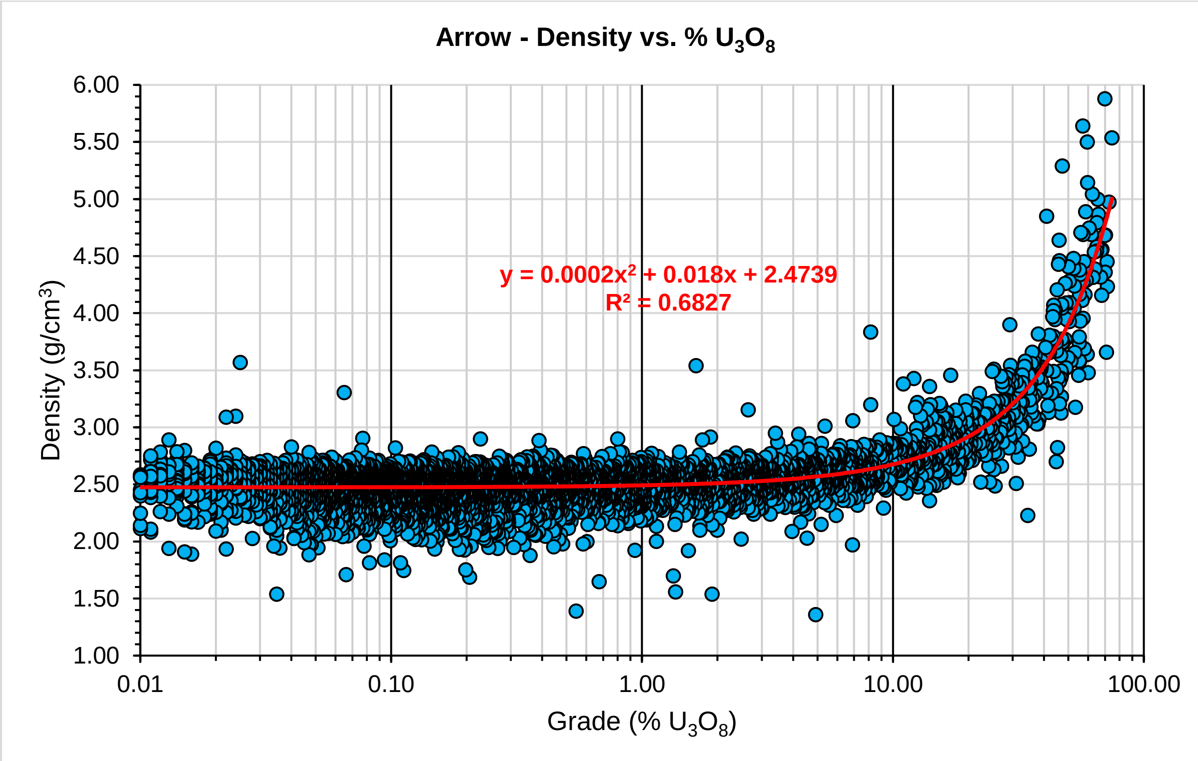
NexGen conducted correlation analyses of the bulk density values against uranium grades. The analyses indicated that a strong relationship exists between density and uranium grade (%U_3O_8), as shown in Figure 11-1. The relationship for the Arrow Deposit can be represented by the following polynomial formula which is based on a regression fit.

$$y = 0.0002x^2 + 0.018x + 2.4739$$

where y is dry bulk density (g/cm^3 which is equivalent to t/m^3) and x is the uranium grade in %U_3O_8.

The uranium grade was used to estimate the density of each sample with the aforementioned polynomial formula. Densities were then interpolated into the block model to convert mineralized volumes to tonnage and used to weight the uranium grades interpolated into each block.

Figure 11-1: Arrow Deposit – Plot of Bulk Density versus Log of Uranium Grade



The regression curve in Figure 11-1 is relatively flat at a grade less than 10% U_3O_8, with density relatively constant at 2.4739 g/cm^3. At grades greater than 10% U_3O_8, dry bulk density increases with higher uranium grades.

There are several strongly mineralized samples that have low dry bulk densities, and LG samples that have high dry bulk density. This has resulted in mild scatter in dry bulk density values.

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.

11.3 Analytical and Test Laboratories

All uranium analyses were carried out at SRC Geoanalytical Laboratories. SRC operates in accordance with ISO/IEC 17025:2005 (CAN-P-4E), General Requirements for the Competence of Mineral Testing and Calibration Laboratories.

SRC is independent of NexGen and RPA.

11.4 Sample Preparation and Analyses

11.4.1 Sample Preparation

SRC crushes each sample until 60% is capable of passing -10 mesh. It is then riffle-split to a 200 g sample, with the remainder retained as coarse reject. The 200 g sample is then milled to 90% passing -140 mesh.

11.4.2 Geochemical Analyses and Assay

All samples are analyzed at SRC by ICP-OES or ICP-MS for 64 elements including uranium. Samples with low radioactivity are analyzed using ICP-MS. Samples with anomalous radioactivity are analyzed using ICP-OES.

Partial and total digestion runs are completed for most samples. For partial digestion, an aliquot of each sample is digested in HNO_3/HCl for one hour at 95 °C, and then diluted using de-ionized water. For the total digestion, an aliquot of each sample is heated in a mixture of HF/HNO_3/$HClO_4$ until completely dried, and the residue dissolved in dilute HNO_3.

For uranium assays, an aliquot of sample pulp is completely digested in concentrated HCl:HNO_3, and then dissolved in dilute HNO_3 before being analyzed using ICP-OES. For boron, an aliquot of pulp is fused in a mixture of NaO_2/$NaCO_3$ in a muffle oven. The fused melt is dissolved in de-ionized water before being analyzed using ICP-OES.

Selected samples are also analyzed for gold, platinum, and palladium using traditional fire assay methods.

11.4.3 Portable Infrared Mineral Analyzer Analyses

Samples are also collected for clay mineral identification using infrared spectroscopy in areas of clay alteration. These samples are typically collected at 5 m intervals; they are typically centimetre-sized pieces of core selected by a geologist.

These samples are transported to Rekasa Rocks Inc. (Rekasa) of Saskatoon, Saskatchewan, by NexGen staff for analysis. Rekasa performs clay analyses using a portable infrared mineral analyzer (PIMA).

11.5 Quality Assurance and Quality Control

NexGen's quality assurance and quality control (QA/QC) program includes the following.

- Standard reference materials (SRM) to determine accuracy.
- Duplicate samples to determine precision / repeatability.

- Blank samples to screen for cross-contamination between samples during preparation and analyses.

The QA/QC program used at the Arrow Deposit included the insertion of SRMs, blanks, and duplicates into the sample stream at the frequency summarized in Table 11-1.

Table 11-1: Laboratory QA/QC Protocols

QA/QC Type	Insertion Frequency	Acceptance Criteria
Blank	1 in 50	Assay > 10% detection limit
Field Duplicate	1 in 50	Relative Difference ≤ ±20%
SRM	1 in 50	95% of samples ≤ ±2 Std. Dev ≤ 1% of samples ≥ ±3 Std. Dev

Results from the QA/QC samples are continually tracked by NexGen as certificates for each sample batch are received. If QA/QC samples of a sample batch pass within acceptable limits, the results of the sample batch are imported into the master database.

11.5.1 Standard Reference Material

SRMs were obtained from the Canadian Centre for Mineral and Energy Technology (CANMET). They included BL2A (0.502 +/- 0.002% U_3O_8), BL4A (0.1472 +/- 0.008% U_3O_8), and BL5 (8.36 +/- 0.04% U_3O_8). The individual SRM inserted into the sample stream were selected based on the core scintillometer measurements.

In zones of drill core radioactivity between 500 cps and 5,000 cps, BL4a is used. In zones of drill core radioactivity between 5,000 cps and 10,000 cps, BL-2a is used. In zones of drill core radioactivity in excess of 10,000 cps, BL-5 is used. SRMs are inserted into the sample stream prior to the first mineralized sample of the drill hole, and systematically thereafter so that they fall on samples XXXX20 and XXXX60. At least one SRM is inserted for each mineralized drill hole.

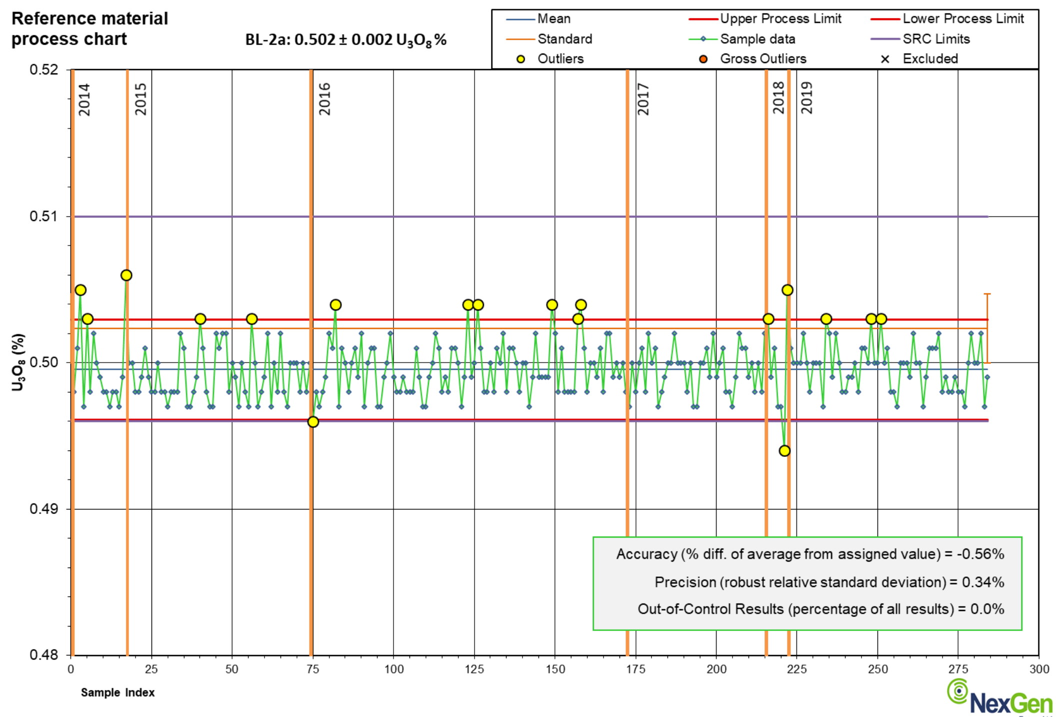
The precision and performance over time of the laboratory is displayed graphically in Figure 11-2, Figure 11-3, and Figure 11-4. The variation from the SRM's mean value in standard deviation (SD) defines the QA/QC variance and is used to determine acceptability of the SRM sample assay. Results within ±2SD are considered acceptable. SRMs fail when more than ±3SD from the mean of the measured values for each type of material is returned.

Z-Score calculations from 2013 to 2018 (see Figure 11-5) show a small but negative bias for BL5 and BL2A and suggest that the SRC declared value of BL4a is incorrect. On investigation, it was found that the BL4a material was certified 30 years previously, using analytical methods that are currently rarely used, and without round robin testing. Nevertheless, over time RPA concluded that the BL4a reference material is extremely homogeneous with repeatable results.

On average, less than 1% of samples were outside the precision limits. One sample from BL-4a returned values in excess of ±3SD from the respective mean, however, because the one sample plotted just above the ±3SD threshold, the decision was made to pass the respective batch.

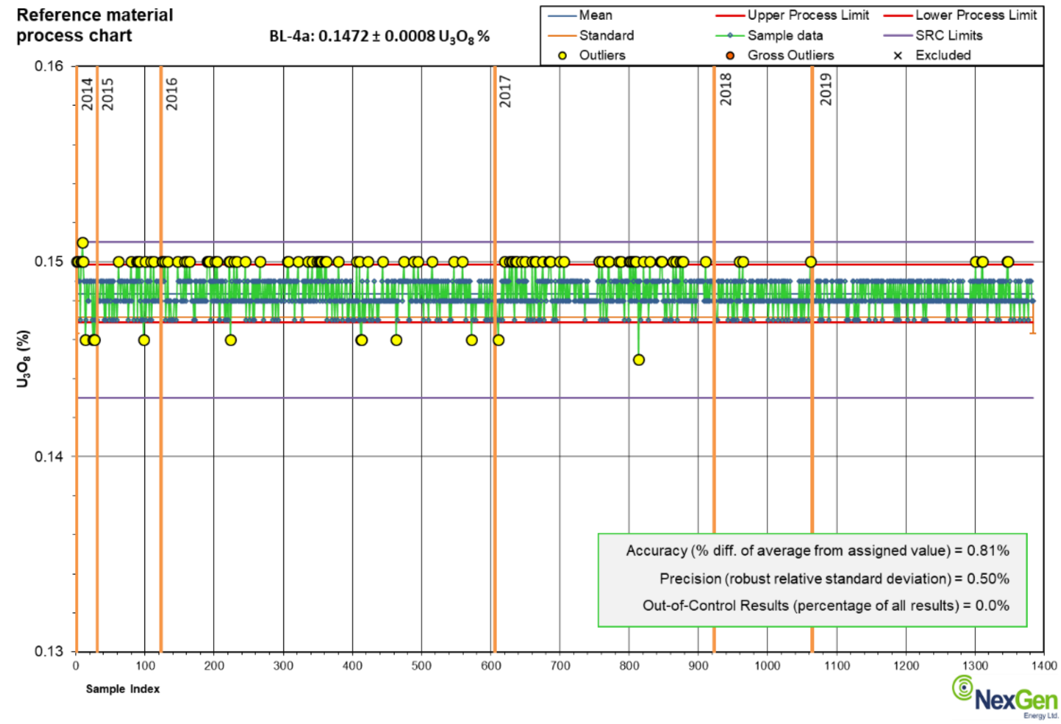
RPA considers there to be a good correlation between the SRMs used and the average economic metal concentration in the drill samples. RPA is of the opinion that the results of the SRM samples from 2014 to 2019 support the use of samples assayed at the SRC laboratory during this period in Mineral Resource estimation.

Figure 11-2: Reference Material Control Chart – BL-2A (Low Grade Standard)



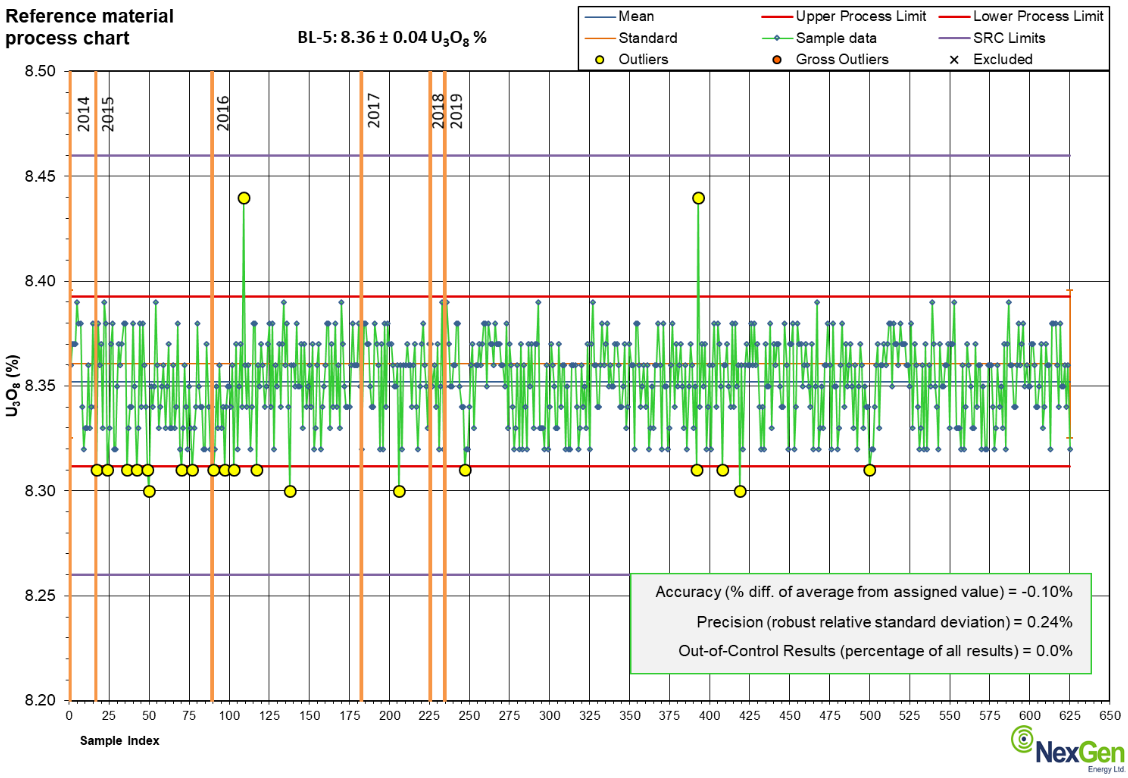
Source: NexGen 2019

Figure 11-3: Reference Material Control Chart – BL-4A (Medium Grade Standard)



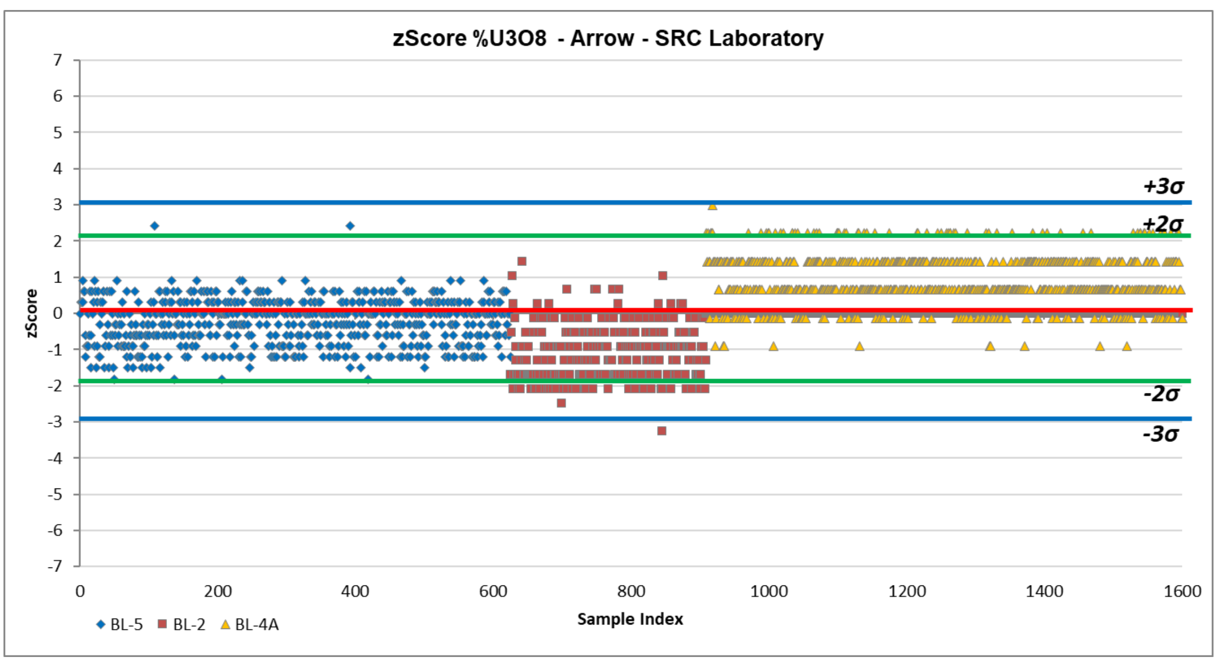
Source: NexGen 2019

Figure 11-4: Reference Material Control Chart – BL-5 (High Grade Standard)



Source: NexGen 2019

Figure 11-5: SRM Z-Scores Over Time for the 2013 to 2019 Period



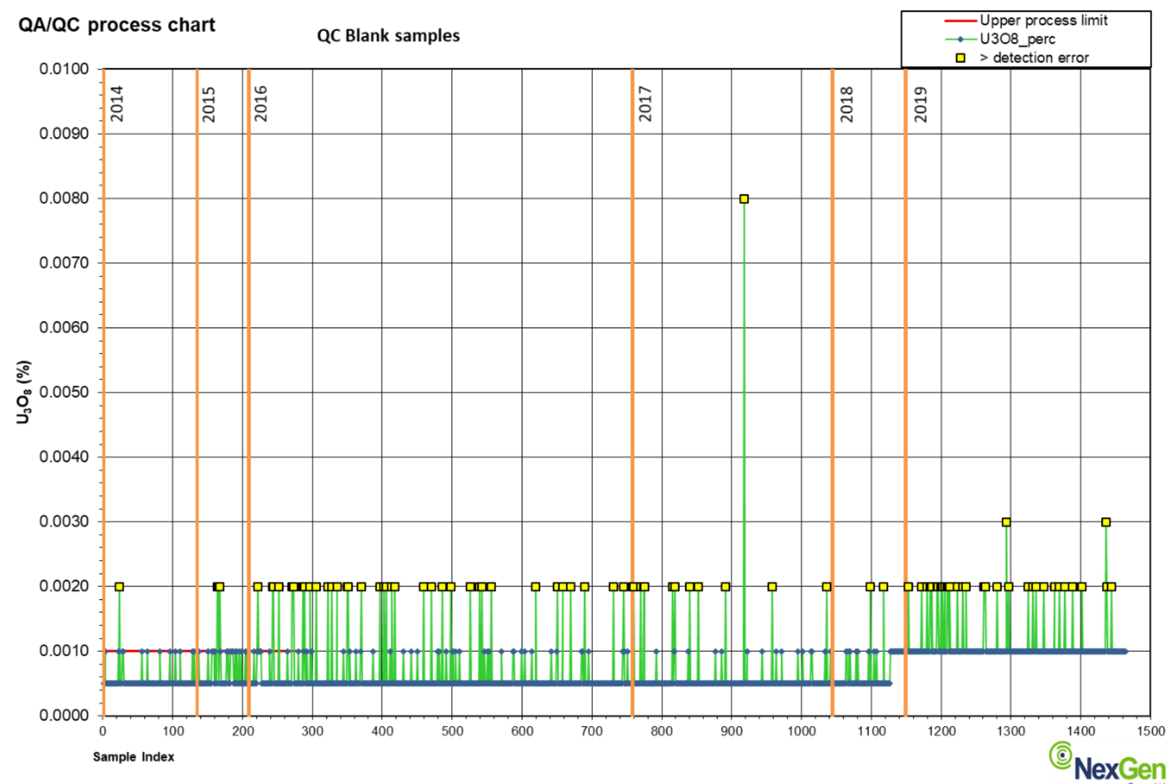
Source: RPA 2019

11.5.2 Blanks

Blank samples are inserted into the sample stream so that they fall on samples XXXX40 and XXXX80. At least one blank sample is inserted into the sample stream for each mineralized drill hole. Blank material samples consist of pieces of rose quartz obtained from Deptuck's Landscaping & Supplies from Saskatoon, Saskatchewan.

Details of the performance of blanks are provided in Figure 11-6. Blanks are considered to have failed when results are greater than 10 times the lower detection limit (LDL). In the case of uranium assays completed at SRC, the pass/fail threshold is 0.001% U_3O_8. Two sample failures occurred. Sample 25604 returned 0.036% U_3O_8 and Sample 104740 returned 0.008% U_3O_8. However, as all other QA/QC samples from those sample batches passed, NexGen chose not to take corrective steps and the batches were passed.

Figure 11-6: Blank Material Control Chart



Source: NexGen 2019

11.5.3 Duplicate Samples

Field duplicates, pulp duplicates, or crush duplicates are submitted to SRC at every 50th even-numbered mineralized sample sent for analysis with the original sample on XXXX48 or XXXX98, the field duplicate on XXXX49 or XXXX99, and alternating pulp and crush laboratory duplicates with pulp duplicates on XXXX50 and crush duplicates on XXXX00. These samples are split into quarter cores at the Project's core processing facility. A minimum of one field duplicate is submitted for each mineralized hole.

SRC also completes laboratory duplicate analysis on 1 in every 10 in-house bulk density measurements completed by NexGen before the respective samples are crushed prior to geochemical analyses. Bulk density measurements at SRC are completed on half cores of entire samples via wax methods.

Figure 11-7 presents the results from field duplicate samples, while Figure 11-8 presents the results from bulk density duplicate samples. RPA is of the opinion that the results are as expected, with acceptable repeatability for both data sets.

Figure 11-7: Field Duplicate Control Chart

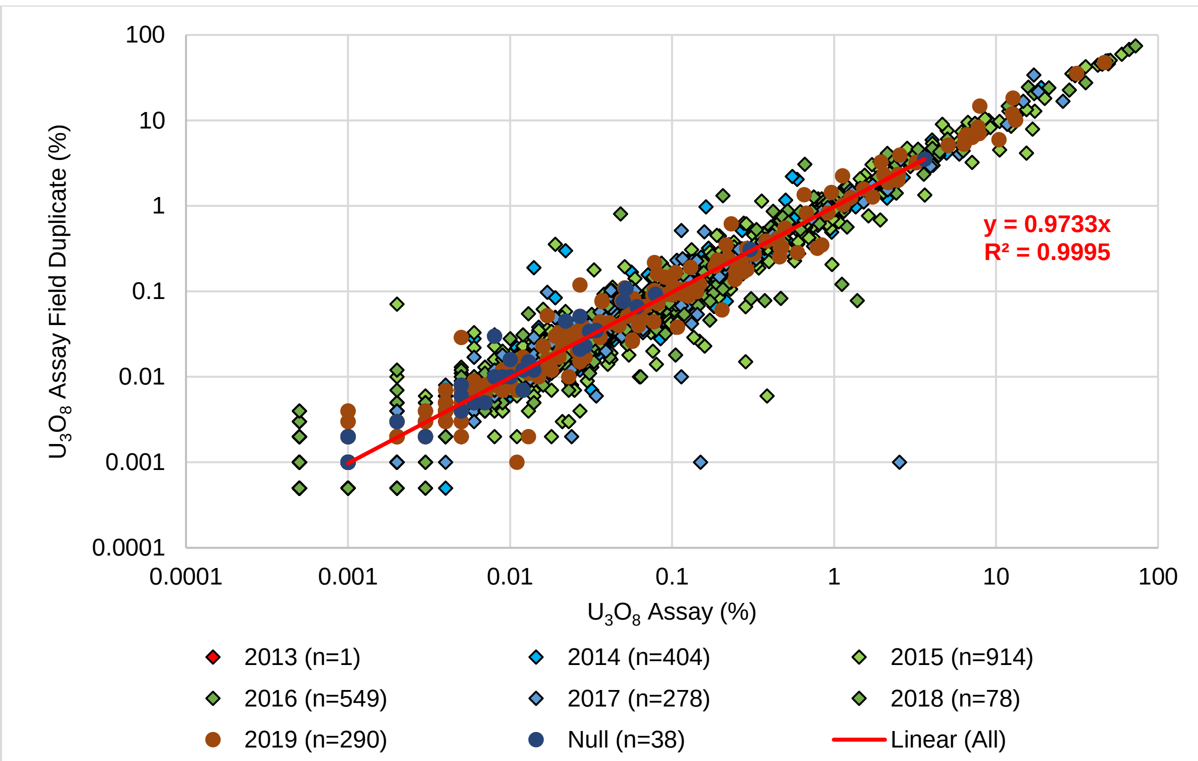
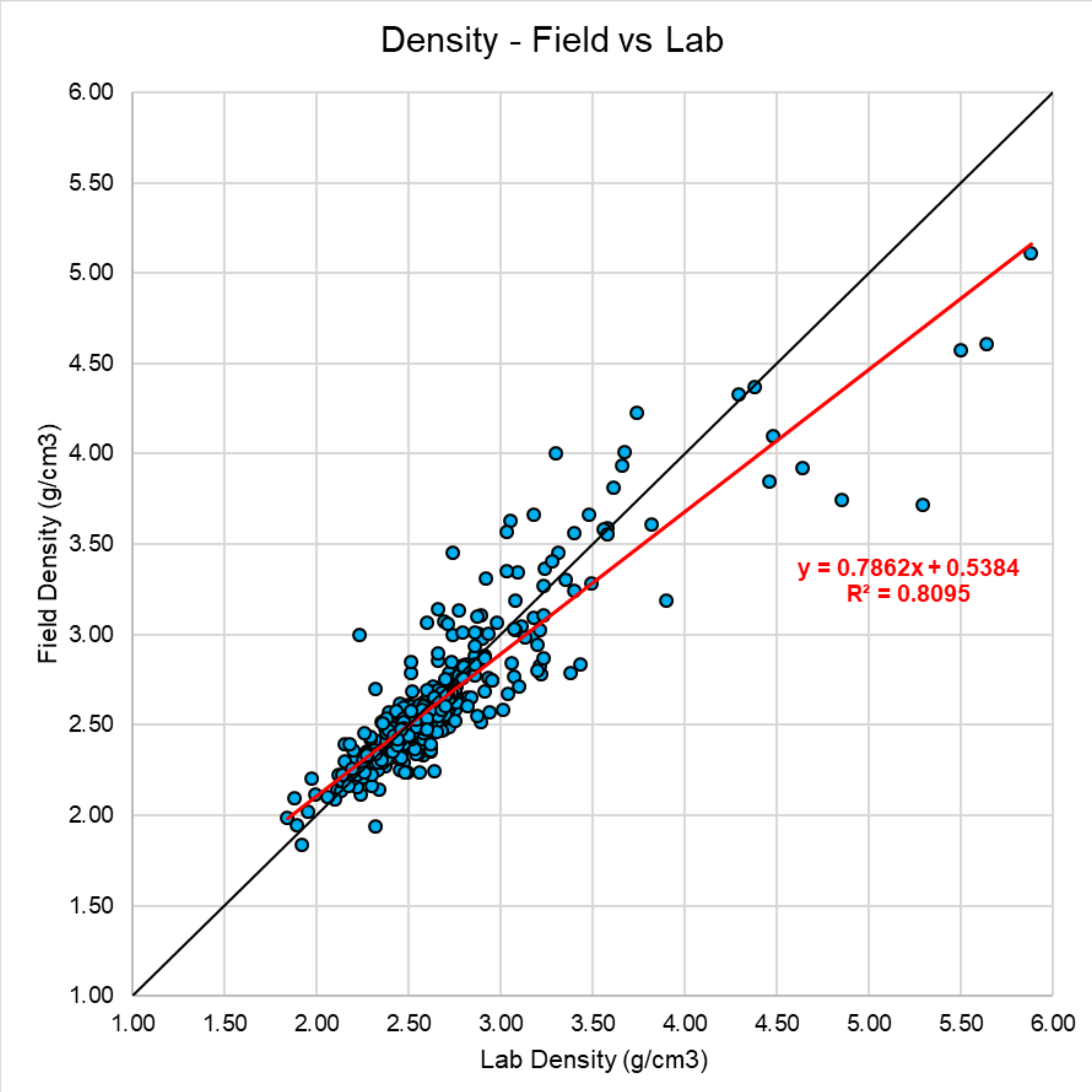


Figure 11-8: Bulk Density Comparison Chart



Source: NexGen 2019

11.5.4 SRC Internal QA/QC Program

Quality control is maintained for all analytical apparatus at SRC with certified reference material used to track analytical drift, and data accuracy and precision. Independently of NexGen's QA/QC samples, standards were inserted into sample batches at regular intervals by SRC. Standards used include BL-2a, BL-4a, BL-5, and SRCUO2 (1.59% U_3O_8), a standard produced in-house at the laboratory. In addition, samples are regularly analyzed in duplicate. All quality control results must be within specified limits, otherwise corrective action is taken. If there is a failure in a QA/QC analysis, the entire batch is reanalyzed.

All processes performed at the SRC laboratory are subject to a strict audit program, which is performed by approved trained professionals.

Results of the QA/QC program have been well-documented by NexGen. RPA has relied on documentation provided by NexGen in addition to an audit completed by RPA of the QA/QC data. RPA considers the QA/QC protocols in place at the Arrow Deposit to be acceptable and in line with standard industry practice.

Based on the data validation and the results of the standard, blank, and duplicate analyses, RPA is of the opinion that the assay and bulk density databases are of sufficient quality for Mineral Resource estimation at the Arrow Deposit.

11.6 Security

As each hole is being drilled, drilling contractor personnel place the core in wooden boxes at the drill site and seal core boxes with screwed-on wooden lids. Core is then delivered to the Project core processing facility by the contractor twice daily. Only the contractor and NexGen geological staff are authorized to be at drill sites and in the core processing facility. After logging, sampling, and shipment preparation, samples are transported directly from the project site to SRC by NexGen staff.

SRC places a large emphasis on confidentiality and data security. Appropriate steps are taken to protect the integrity of samples at all processing stages. Access to the SRC premises is restricted by an electronic security system and patrolled by security guards 24 hours a day.

After the completion of analyses, data is sent securely via electronic transmission to NexGen. These results are provided as a series of PDFs and an Excel spreadsheet.

In RPA's opinion, the sample preparation, analysis, and security procedures at the Arrow Deposit are adequate for use in the estimation of Mineral Resources.

12.0 DATA VERIFICATION

12.1 Site Visit and Core Review

Mr. David A. Ross, M.Sc., P.Geo (formerly Principal Geologist with RPA) visited the Project property on 12 June 2019. Mr. Mark Mathisen, CPG, visited the property on 19–20 January 2016, and 22–25 January 2017 during the winter drill programs in connection with the previous Arrow Deposit Mineral Resource estimates. RPA visited several active drill sites and targets.

During the 2016, 2017, and 2019 site visits, RPA reviewed core handling, logging, sample preparation and analytical protocols, density measurement systems, and storage procedures. RPA examined cores from the following six drill holes.

- AR-14-30
- AR-15-57c3
- AR-15-62
- AR-16-98c1
- AR-16-106c1
- AR-16-111c1

RPA compared their observations with assay results and descriptive log records created by NexGen geologists. As part of this review, RPA verified the mineralization occurrences visually and by way of a hand-held scintillometer.

As part of the data verification process, RPA also completed the following.

- Reviewed the Leapfrog model parameters and geological interpretation.
- Reviewed how drill hole collar locations were defined.
- Inspected the use of directional drilling procedures and operations.
- Observed data management systems and reviewed the master database.
- Obtained SRC laboratory certificates for the 2019 drilling assays.

12.2 Database Validation

RPA performed the following digital queries.

- Header table
  - Searched for incorrect or duplicate collar coordinates and duplicate hole identification numbers (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 lengths, and data points past the specified maximum depth in the collar table
- Lithology
  - Searched for duplicate entries, intervals past the specified maximum depth in the collar table, overlapping intervals, negative lengths, 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 lengths, overlapping intervals, sampling lengths exceeding tolerance levels, missing collar data, missing intervals, and duplicated sample IDs
- Exported the data from an acQuire database and imported it into a Vulcan database
  - The 2019 Vulcan database utilized a similar design as the acQuire database
  - Quality control was completed in acQuire, and validation was completed in Vulcan and Leapfrog
- Implemented the following density hierarchy:
  1. SRC density values (laboratory results)
  2. NexGen density values (field results)
  3. Calculated values (polynomial regression)

Validation files, quality control files (e.g., duplicates, blanks, standards), third-party metallurgical work, and an internal check list (e.g., survey datum, equipment used, estimation parameters) were all available in the provided Vulcan workspace.

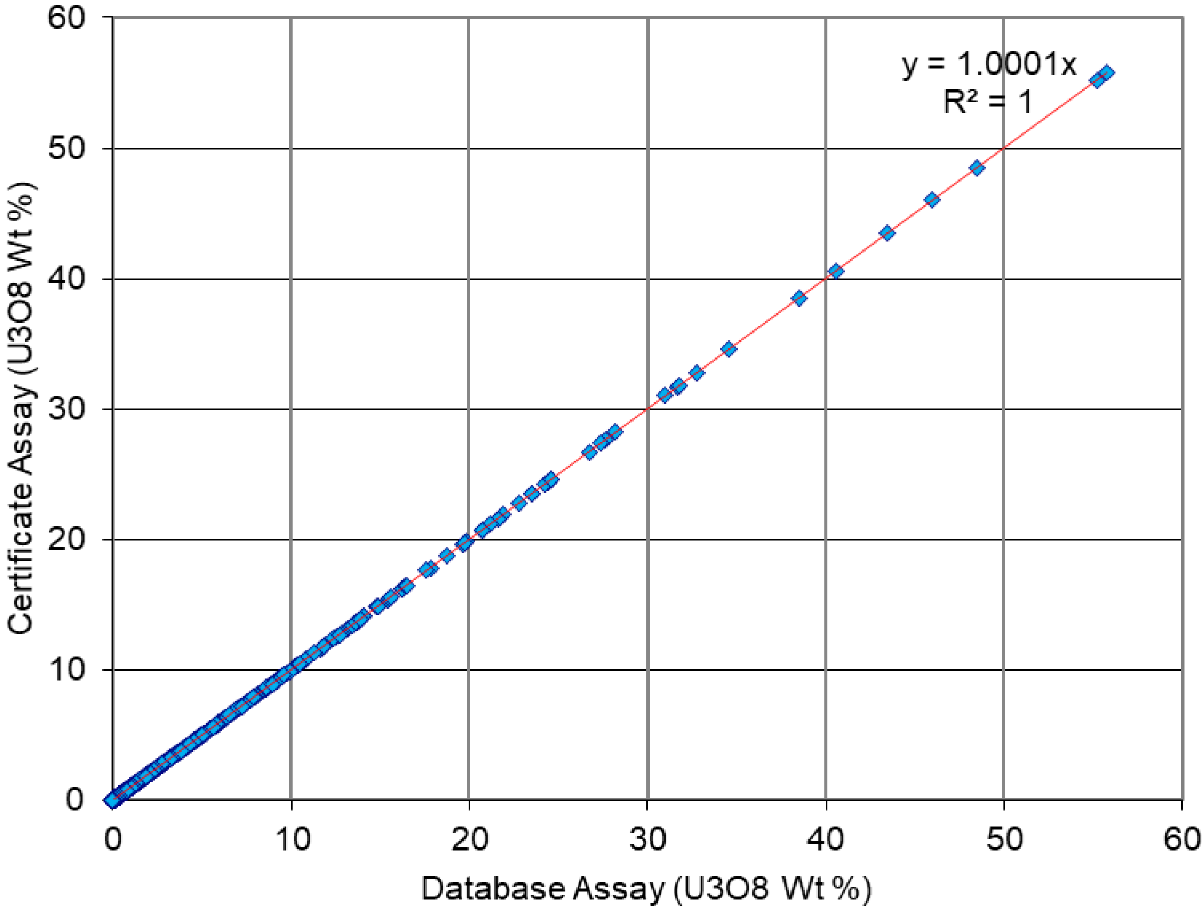
RPA is of the opinion that data collection and entry, and database verification procedures for the Arrow Deposit comply with industry standards and found the database to be sufficiently reliable for Mineral Resource estimation.

12.3 Independent Verification of Assay Table

The drilling database contains a total of 83,483 assays (i.e., 80,659 samples with uranium values and 2,825 samples that have an assigned grade of 0.0 due to not being sampled or below detection limit) used for estimating the Mineral Resource. RPA conducted checks on assays within the database against corresponding laboratory assay certificates in search of any errors occurring during data transfer and importation. RPA randomly checked approximately 13% (10,852 samples) of the drilling database with minimal errors found and all most likely due to rounding.

In RPA's opinion, the integrity of the database is acceptable for a Mineral Resource estimate. Figure 12-1 illustrates the consistency between the database and original certificates.

Figure 12-1: Database versus Laboratory Certificates U_3O_8 Weighted Percentage



RPA reviewed and verified the resource database used to estimate the Mineral Resources for the Arrow Deposit. The verification included a review of the QA/QC methods and results, comparison of the database assay table against assay certificates, standard database validation tests, and a site visit including drill core review. No limitations were placed on RPA's data verification process.

RPA considers the resource database to be reliable and appropriate to prepare a Mineral Resource estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

The SRC was contracted to do a metallurgical test program using samples from the Arrow deposit. The metallurgical test program included a bench test program (March 2018), a pilot plant program (July 2018) and paste backfill testing (July 2018). All test programs were developed and performed under Wood's supervision. During 2019 a series of tests were carried out to refine the process design. These tests were carried out at the SRC facilities. Tests included the following.

- Bench scale testing to recover uranium from gypsum (June 2019) – the gypsum testing was developed and performed under Wood's supervision.
- Trade-off study / test work of dewatering and washing technologies using belt filters (July 2019) – the belt filter testing was developed and performed under Hasler Group's supervision.
- Trade-off study / test work of dewatering and washing technologies using centrifuges (August 2019) – the centrifuge dewatering testing was developed and performed under Wood's supervision.

An advanced phase of paste backfill testing (2019) was also commenced using these project samples, which was developed and managed under Paterson & Cooke's supervision and was conducted at the SRC facilities.

13.1 2018 Metallurgical Test Work

13.1.1 Bench Testing

The bench tests were undertaken on three composite samples.

- High grade: 3.00% U_3O_8
- Medium grade: 2.03% U_3O_8
- Low grade: 0.87% U_3O_8

In addition to these three samples, ten additional samples of localized deposit areas were also tested.

- Five individual zones, A1 to A5
- One very high grade (VHG) zone
- One HG Mo/U zone
- One gangue sample
- One high rare earth element (REE) sample
- One low REE sample

A total of thirteen composite samples were prepared and used for QEMSCAN, leaching, PAG and tailings preparation for paste backfill tests.

QEMSCAN was used to characterize uranium mineralization and to identify gangue and deleterious minerals such as molybdenum (Mo), as well as gold (Au) / silver (Ag) deportment. QEMSCAN modal mineralogy analysis was performed on the MG composite and each of the A1 to A5 shear zone samples.

Five composite samples were subject to SAGDesign™ and Bond ball mill index test work.

One large batch leach test was completed on a MG composite sample (50 kg), at baseline conditions (pH 1.1 for 8 hours, 100% -300 µm). This test was to generate adequate quantity of baseline residue and pregnant leach solution (PLS) for the downstream tests.

A total of 21 optimization leaching tests were performed on the HG, MG, and LG composite samples. A total of three confirmation tests and eight variability tests were performed using the same parameters as the optimization leaching tests.

Settling tests included flocculant screening and dosage optimization, tests on the discharge from the HG, MG, and LG optimization leaching tests, and tests on the discharge from ore variability tests at optimized leaching conditions. Beaker settling tests on the large batch leach discharge of the MG composite at baseline conditions were performed to select the most applicable flocculant.

A total of eight settling tests were performed on the five individual zones and on the two extra-HG and one high molybdenum / uranium ratio composites' leach discharges using optimized leaching conditions.

SX tests included SX variable optimization tests, tests on the settling filtrate from optimization leaching tests, and tests on the settling filtrate from variability leaching tests.

Seven separating funnel shakeout tests were performed on the large batch settling filtrate of the MG composite at baseline conditions to assess four different organic / aqueous ratios, one test of fresh organic without pre-protonation of the organic, and two tests on pre-protonated organic with modified amine / isodecanol levels. Separating funnel shakeout tests using standard conditions were performed on the 21 settling filtrates of the HG, MG, and LG samples. Separating funnel shakeout tests using standard conditions were performed on the eight settling filtrates of zones A1 to A5, the VHG and high molybdenum / uranium samples.

Using bench optimization test results a flowchart for treating large batch PLS was developed and relevant large bench scale experiments were carried out, including the following.

- Organic protonation
- Extraction
- Acid scrubbing

- Strong acid stripping
- Gypsum precipitation
- YC precipitation
- Effluent treatment

Organic protonation (SX8-1) was conducted prior to the extraction tests. Four pails of the bulk PLS, collected from the previous tests from leaching 50 kg of ore, were remixed and concentrated to 11.57 g/L U_3O_8. Twenty-five liters of the concentrated solution, denoted as "fresh bulk PLS", was used in the first stage of the extraction test (SX8-2). In the second-stage test (SX9), 11.25 L of fresh bulk PLS was used to contact the loaded organic generated from the first stage extraction. The organic / aqueous ratio in both stage extraction was 1:1 and 2:1, respectively. The rise in the organic / aqueous ratio in the second stage extraction resulted from the lack of the fresh bulk PLS. The U_3O_8 concentration in the loaded organic for both stages was estimated based on uranium assay in each stage's raffinate.

A total of 22.25 L of loaded organic was mixed with 7.42 L of diluted acidic solution (20 g/L H_2SO_4) at the organic to aqueous phase (O/A) ratio of 3:1 in ambient temperature (SX10) for removing arsenic from the organic phase to the aqueous phase.

Three sets of stripping tests were completed.

- Five separating funnel shakeout tests were performed to determine uranium loading to strong acid acidic solution in five different organic / aqueous ratios (10:1, 15:1, 18:1, 20:1, and 25:1). All tests followed a standard SX shakeout extraction test procedure.
- Eight separating funnel shakeout tests (SX18) were performed to determine the lowest U_3O_8 concentration achievable in the barren organic at organic / aqueous ratios of 20:1. All tests followed a standard SX shakeout extraction test procedure.
- Five sets of three-stage of stripping tests (SX19) were carried out to generate adequate bulk loaded strip solution use in downstream gypsum and YC precipitation tests.

One gypsum precipitation test (SX20) was carried out with a diluted loaded strip solution, which was obtained by combined the first, second, and third loaded strip solutions from SX19 and was used for gypsum precipitation and filtration tests.

Two YC precipitation tests were performed (SX21) at a small bench scale and large bench scale. The reagent addition sequence for H_2O_2 and MgO was slightly different in each test.

Two tailings neutralization tests (L1-NT) were performed on the bulk leach residue after two more-time washes with pH2 deionized water (47.5% solids and 50 ppm U in wash). Test-1 was a small bench-scale test, and aimed to verify how much lime might be consumed for generating minimum tailings. Test-2 was a larger bench-scale test to confirm the results from Test-1.

One preliminary sulfide flotation test was performed on the bulk leaching residue to investigate the efficiency to recover molybdenum and copper (Cu).

Approximately 1.6 kg wet residue was neutralized and screened into different size fractions (+212-300, +106-212 and +45-106 µm) for diagnostic gravity separation (heavy media separation [HMS]). Each size fraction was tested with five specific gravities (3.18, 3.1, 2.8, 2.6, and 2.4).

One effluent treatment test (BNT-Raff&YC BS) was performed at ambient temperature.

13.1.2 Pilot Plant Testing

Two pilot leaching tests were performed using two different feed samples.

The feed of the first pilot leaching test (MG pilot) was the MG composite sample. The MG composite sample represented mineralization studied in the 2017 PEA. The sample contained 2.03% U_3O_8, 315 ppm molybdenum and 37 ppm arsenic. The total weight was 409.3 kg. The feed was ground to 100% passing 300 µm using a 1 ft by 3 ft pilot size ball mill. Particle sizing was P85 = 170 µm.

The feed of the second pilot test (2C pilot) was the combined composite samples other than the MG sample. The calculated grade of the combined sample was 4.89% U_3O_8, based on the assays of the individual composite samples. This composite sample represents a wide range in the deposit mineralization and reflects an overall higher uranium grade. The combined sample of 466.6 kg was ground to 100% passing 300 µm using the 1 ft by 3 ft pilot size ball mill. The 2C sample was only used in the pilot leach and tailings neutralization tests to generate enough sample for paste testing. Only the MG sample was used for all the milling processing circuits.

The pilot test program was set up as a series of pilot-sized tests to represent the following unit operations.

- Leaching and solid / liquid separation
- Tailings neutralization
- SX
- Gypsum precipitation, settling, and leaching
- YC precipitation and settling
- Effluent treatment and settling

13.1.3 Test Work Results

Mineralogy

Uranium is present as fresh and altered uraninite. Samples are dominated by clay minerals: muscovite / illite and chlorite. All other mineralization is strongly associated with these clay minerals. There are few free uraninite particles but the uraninite is exposed to leaching. No primary molybdenum-bearing minerals were identified in any samples. Only two gold grains were identified in one the MG composite sample. There were no other gold grains in any other samples.

Grinding

Grinding test results indicate a medium hardness deposit. Average semi-autogenous grinding (SAG) mill pinion energy was 9.94 kWh/t (57th percentile in Starkey's database). The ball mill index ranked in the database 47th percentile at 14.69 kWh/t.

Leaching

Tests indicated that good uranium extraction was generally achieved within an 8-hour residence time. In bench tests, a few samples benefited from longer residence times between 8–12 hours.

Settling Testing – Leach Precipitates

Settling tests of leached solids indicated that the solids separated relatively quickly to give a high-density slurry. Average density achieved in the pilot test was very good achieving 59.3% solids with the MG sample. In the bench tests, only samples with relatively high clay mineralogy required more time and resulted in reduced densities (still over 48% solids).

Solvent Extraction

Bench SX tests yielded good uranium extraction from the leach pregnant aqueous solution. Extractions ranged from 98.0% to 99.9%. One of the seven HG samples had an extraction of 93.5% (the other six tests all had extractions of over 99.5%) and one of the seven MG samples had an extraction of 96.5% (the other six tests all had extractions over 98.4%).

Arsenic Scrubbing

As wash efficiency was low (27.8%). However, the arsenic concentration in the organic is very low and is only 9.0 ppm compared to 12.7 g U/L, indicating that washing is not an issue.

Gypsum Precipitation and Washing

The gypsum precipitation and washing testing produced gypsum solids that contained 296 ppm U. This represents a 0.2% recovery loss.

Yellowcake Precipitation

Bench testing of YC precipitation / washing and drying methods yielded two samples that met product specifications without being rejected. However, both samples had some level of impurities above the penalty limits (calcium [Ca], magnesium [Mg], and phosphorus [P]). Most of the contamination in the YC was likely from reagent additions in the precipitation process. With better control in a mill circuit, calcium, magnesium, and phosphorus parameters will not likely be an issue.

Froth Flotation to Recover Molybdenum / Copper

Recovery of molybdenum was 45% and recovery of copper was 89%. No further work was done on these elements.

Gravity Separation to Recover Gold

The HMS was hard to perform on the fine-grained material. No significant gold can be recovered by gravity and no further test work was conducted.

Effluent Treatment

Final effluent quality from the bench test achieved results that were measured below the Canadian Metal and Diamond Mining Effluent Regulations (MDMER) – Schedule 4 limits.

Pilot Testing

Pilot leaching of the MG sample resulted in uranium leaching rates at 8-hour retention that ranged from 98.8% to 99.4% with an average of 99.2%. Pilot leaching of the 2C sample resulted in uranium leaching rates at 8-hour retention that ranged from 98.2% to 99.7% with an average of 99.5%.

In the SX pilot testing the uranium was almost completely extracted with more than 99.999% transferred to the organic phase. The SX was very selective for uranium. Most of the impurity metals (e.g., iron [Fe], aluminum [Al], calcium, magnesium, sodium [Na], potassium [K], manganese [Mn], vanadium [V], copper, zinc [Zn], cobalt, nickel, and arsenic) were left in the raffinate. Most molybdenum was extracted along with the extraction of uranium. Other impurities are typical for uranium raffinate. The strip solution contained 133.1 g/L U_3O_8.

Methods of adding lime slurry to precipitate gypsum from the loaded strip were tested. Tests were also done to determine the concentration of uranium in the gypsum particles as the gypsum is precipitated out of the loaded strip solution. It was found that the gypsum could be washed with acid rinses to bring the level of uranium in the precipitated gypsum to between 200–300 ppm U. This represents a loss of about 0.23% U that is locked in the gypsum precipitate. Washing of the concentrated uranium strip solution from the gypsum must be done thoroughly to ensure uranium losses to the surface of the gypsum particles are low. Gypsum washing performance in the pilot testing represents an additional 0.25% U loss.

Settled gypsum was only 10% to 22% solids in settling tests. When centrifuging, the gypsum cake was dewatered to an average of 60% solids.

In the pilot testing, when comparing to American Society of Testing and Materials (ASTM) C967-13 and client specification requirements, YC assays indicated that iron, magnesium, and silica content per uranium exceeded the penalty limit but were substantially lower than the reject limits. Parameters, other than mentioned as follows, were below penalty concentration limits.

- Iron (Fe) was 0.23 to 0.37%:U, penalty level 0.15%:U, reject level 1.0%:U (ASTM) and 0.5%:U (client).
- Magnesium (Mg) was 0.08 to 0.11%:U, penalty level 0.02%:U and reject level 0.50%:U.
- SiO_2 was 0.71 to 0.85%:U, penalty level 0.50%:U, reject level 2.5%:U (ASTM) and 2.0%:U (client).
- Fluorine (F) was 0.11% (not calcined) to <0.01%:U, penalty level 0.01%:U, reject level 0.10%:U. Note that as calcine temperature increased, concentration of F decreased. The YC sample calcined at 800 °C was acceptable without penalty (<0.01%). Non-calcined YC would have been rejected by this ASTM and client standard (0.11%:U assay compared to 0.10%:U reject).

Peroxide precipitation of uranyl sulphate is known to be a selective process and it is expected that the process design will be able to produce YC within the accepted product specifications.

Calciner tests were conducted on uranyl peroxided YC produced in the pilot test. Different calcining temperatures were used. X-ray diffraction (XRD) analysis indicated that most of the uranyl peroxide is U_3O_8 by 600 °C. The initial uranyl peroxide sample was off spec with respect to fluorine. The fluorine assay for YC calcined at 800 °C was below penalty concentration limit.

Similar to the bench testing, final effluent quality achieved in the pilot effluent treatment testing was below the Canadian MDMER – Schedule 4 limits.

13.1.4 Paste Backfill Testing (July 2018)

The sieve analysis results for the tailings indicate that approximately 65% of the particles are below 75 µm and approximately 35% are finer than 20 µm. A minimum of 15%–20% by weight of minus 20 µm (625 mesh) material is required for homogeneous non-settling pipeline transport. Both the results of the sieve analysis and laser particle size distribution (LPSD) show the tailings contain high amounts of fine particles (below

20 µm). The fineness of the tailings affects the water demand of the paste mix; water demand increases with the fineness of the tailings. Therefore, the presence of clays in the tailings leads to increased water demand for the mix, which is related to strength development.

Acid-base accounting (ABA) testing indicated that all the tailings samples contain sulphide minerals. They are net acid generating materials with the potential to produce sulphuric acid, which can affect cement hydration.

Index tests were conducted for paste without binder with total tailings contents ranging from 77.50% to 64.00%. Trial batches were conducted for paste mixes with no binder (0%), 4%, and 7% Portland cement in the mix to develop mix designs to meet the target slump, while maximizing the tailings content for a given binder and binder dosage rate. A minimum of 4% Portland cement / slag binder is required to meet the 28-day UCS target for low strength backfill and a minimum of 5% Portland cement / slag is required to meet the 28-day UCS target for regular strength and high strength backfill.

13.2 2019 Metallurgical Test Work

13.2.1 Bench Scale Testing to Recover Uranium from Gypsum (June 2019)

The Bench scale testing to recover uranium from gypsum was preformed to recover uranium from the gypsum in order to achieve a concentration of less than 250 ppm. The test program was designed to minimize uranium "occlusion" as gypsum is precipitated, wash dissolved uranium from the gypsum solids very thoroughly and possibly leach some uranium from gypsum solids. Washing was done initially with pH 3 acid solution. Washing was done six times to ensure a thorough wash. Another four water washes were done after the acid washes. Assays were done on the gypsum after the washing stages and on the filtrates that were removed from the dewatered (centrifuged) gypsum.

Acid solutions of pH 1.0, 1.5, and 2.0 were used to leach the clean gypsum from the above procedure. Leaching was done three times for each wash pH. Gypsum uranium assay was done for each step for the gypsum solids as well as the centrates. Moisture assays were done on the gypsum cakes and are an indication to the minimum moisture that could be expected in a centrifuge in the washing / dewatering mill process.

Acid washing of the gypsum is required to achieve a uranium concentration of less than 250 ppm. During the FS, the decision was made to store all gypsum UG which means that target of less than 250 ppm U is no longer required.

13.2.2 Trade-Off Study / Test Work of Dewatering and Washing Technologies Using Belt Filters (July 2019)

Filtration and washing tests were performed on un-neutralized leach residue (UNLR), gypsum precipitates and YC solids. Filtration testing was performed on neutralized leach residue (NLR), a mixture of tailings materials (NLR and blend of effluent treatment precipitates), and a more diverse mixture of tailings materials (NLR, gypsum, and a blend of effluent treatment precipitates).

Un-Neutralized Leach Residue Testing

The test for dewatering and washing of UNLR involves dewatering the leach residue and washing the uranium rich solution from the barren gangue solids. This test investigates the replacement of CCD thickeners with a belt filter.

Acceptable washing efficiency and water use was achieved for the UNLR material; however, the filtration rate achieved would lead to a series of three large filters. Hasler indicated that the solids particle size distribution was lower than they have experienced elsewhere for leach residues. However, due to the mineralogy of the Rook I deposit the fine particle size of the UNLR is necessary.

The conclusion for this testing was to not use belt filters for the UNLR.

Gypsum Testing

The test for dewatering and washing of gypsum precipitates involves dewatering the gypsum precipitates and washing the uranium-rich solution from the precipitates. This test investigates the replacement of centrifuges with a belt filter.

Acceptable dewatering was achieved with about 30% free moisture in cake. Acceptable washing ratio and cake thickness were achieved. The target washing efficiency of 99.75% was exceeded. The filtration rate achieved gives a reasonably sized belt filter of 10.5 m^2. Another positive factor to this test was that no flocculant was required.

The conclusion from this testing was that a belt filter is acceptable for dewatering and washing of gypsum precipitates.

Yellowcake Testing

The test for dewatering and washing of YC involves dewatering the YC and washing to reduce the dissolved solids in the cake. This test investigates the replacement of a centrifuge and a thickener with a belt filter.

The YC sample produced from the pilot testing was very fine resulting in high viscosity and poor filtering performance. Belt filters are used to dewater and wash YC at uranium operations around the world. It is believed that the YC produced in the pilot testing is not representative of the YC produced in operating plants. The procedure to produce YC will be updated to attempt to more closely replicate plant performance and the belt filter testing will be repeated during basic engineering.

Neutralized Leach Residue Dewatering

Filtration testing was performed on a sample of NLR to dewater the NLR to a level to provide a suitable moisture content for the paste backfill plant.

Filtration rates and cake moisture contents achieved make a belt filter suitable for this application.

Tailings Mixture Dewatering

Filtration testing was performed on a sample of NLR and effluent precipitates and on a sample of NLR, effluent precipitates and gypsum to dewater the slurries to a level to provide a suitable moisture content for the paste backfill plant.

Filtration rates and cake moisture contents achieved on the sample of NLR and effluent precipitates make a belt filter suitable for this application.

Filtration rates and cake moisture contents achieved on the sample of NLR, effluent precipitates and gypsum result in a very large belt filter suitable for this application. The filtered gypsum cake from the gypsum dewatering and washing should be combined to the cake of the NLR and effluent precipitates and should not be blended with the NLR and effluent precipitates in the filter feed tank.

13.2.3 Trade-Off Study / Test Work of Dewatering and Washing Technologies Using Centrifuges (August 2019)

Centrifuge dewatering and washing tests were conducted on samples of UNLR, gypsum precipitates, and YC samples. Centrifuge dewatering tests were conducted on a sample of NLR, a combined sample of NLR and effluent precipitates and a combined sample of NLR, effluent precipitates and gypsum.

The test results for the YC sample dewatered poorer than expected due to the fineness of the YC. All other samples produced results that were acceptable for those samples.

13.2.4 Paste Backfill Testing (2019)

Paste Backfill Testing Summary

A full range of paste backfill related tests were conducted to aid obtaining parameters to adequately design backfill recipes.

The following tests (including methods used) were conducted to establish the material properties of the various waste streams materials.

- Particle size distribution (PSD) using sieving and hydrometer method.

- Whole rock analysis using a lithium metaborate fusion with induced coupled plasma (ICP) analysis.
- The mineralogy using an XRD methodology.
- Static yield stress using a rotational viscometer with a vane attachment.
- Transportable moisture limit using a standard floating table equipment.
- UCS tests using 2-inch by 4-inch (51 mm by 102 mm) cylinder moulds cured in chambers with an ambient temperature of 23 °C (±2 °C) and greater than 95% relative humidity.
- Process water analysis.
- Humidity cell and leaching EA framework tailings kinetic tests.

Table 13-1 shows a summary of the tests conducted for paste (both cemented paste backfill [CPB] and cemented paste tailings [CPT]) design purposes.

Table 13-1: Paste Backfill Test Summary

Test Material	Skeletal Solids Density	Particle Size Distribution	Mineralogy	Water Analysis	Rheology	Transportable Moisture Limit
High Grade NLR	✓	✓	✓		✓	✓
Medium Grade NLR	✓	✓	✓		✓	
Clean Gypsum (U <300 ppm)	✓	✓	✓		✓	
Reject Gypsum (U >300 ppm)	✓	✓	✓		✓	✓
Effluent Precipitates (High & Low pH Stream Mixed in Process Ratio)	✓	✓	✓		✓	✓
Process Water from High Grade Leach Residue				✓		
Process Water from Medium Grade Leach Residue				✓		
Cemented Paste Tailings 1 High Grade Leach Residue and Precipitates with Low Cement Content					✓	
Cemented Paste Tailings 1 High Grade Leach Residue and Precipitates with High Cement Content					✓	
Cemented Paste Tailings 2 High Grade Leach Residue, Reject Gypsum and Precipitates with Low Cement Content					✓	

Test Material	Skeletal Solids Density	Particle Size Distribution	Mineralogy	Water Analysis	Rheology	Transportable Moisture Limit
Cemented Paste Tailings 2 High Grade Leach Residue, Reject Gypsum and Precipitates with High Cement Content					✓	
Cemented Paste Backfill High Grade Leach Residue with Low Cement Content					✓	
Cemented Paste Backfill High Grade Leach Residue with High Cement Content					✓	

Table 13-2 outlines the UCS tests conducted for this Project.

Table 13-2: UCS Test Summary

Mix	Tailings Content	Binder Type	Binder Content	Concentration on a Weight Basis of Mix	Water:Binder Ratio
1	NLR	100% Ordinary Portland Cement (OPC)	4.5%	64%m	12.5
2	NLR	100% OPC	5.5%	64%m	10.2
3	NLR	100% OPC	7.5%	64%m	7.5
4	NLR	100% OPC	11%	64%m	5.1
5	NLR	100% OPC	22.5%	64%m	2.5
6	NLR	50% Slag / 50% OPC	4.5%	64%m	12.5
7	NLR	50% Slag / 50% OPC	5.5%	64 %m	10.2
8	NLR	50% Slag / 50% OPC	7.5%	64 %m	7.5
9	NLR	50% Slag / 50% OPC	11.0%	64 %m	5.1
10	NLR	50% Slag / 50% OPC	22.5%	64 %m	2.5
11	NLR + Precipitate	100% OPC	8.0%	55.5 %m	10.0
12	NLR + Precipitate	100% OPC	11.0%	55.5 %m	7.3
13	NLR + Precipitate	100% OPC	14.0%	55.5 %m	5.7
14	NLR + Precipitate	100% OPC	20.0%	55.5 %m	4.0
15	NLR + Precipitate + Gypsum	100% OPC	8.0%	55.5 %m	10.0
16	NLR + Precipitate + Gypsum	100% OPC	11.0%	55.5 %m	7.3
17	NLR + Precipitate + Gypsum	100% OPC	14.0%	55.5 %m	5.7

Mix	Tailings Content	Binder Type	Binder Content	Concentration on a Weight Basis of Mix	Water:Binder Ratio
18	NLR + Precipitate + Gypsum	100% OPC	20.0%	55.5 %m	4.0

Paste Backfill Testing Results

All materials tested are fine with the NLR having approximately 35% passing 20 µm and the gypsum and effluent precipitates having approximately 58% and 54% passing 20 µm respectively. This is within the typical range for paste production and transportation within a pipeline. The gypsum and effluent precipitates both have approximately 35% materials passing 10 µm which increases the rheology measurement of the planned paste mixtures. A paste yield stress in excess of 200 kPa is targeted to ensure a homogeneous paste mixture. This equates to solids mass concentrations of approximately 55%m for the CPT and approximately 63%m for the CPB.

The silicate mineral quartz is present in the NLR (approximately 30%). This mineral is inert and does not participate in the hydration dynamics of the cementitious reactions for backfill purposes. As such, it is considered a good filler material for backfill. There is a portion of the mica mineral muscovite in both the NLR (approximately 50%). The micas have weak bonds between the internal sheet structure of the minerals allowing for failure planes and crack propagation paths that could lead to lower strengths of the backfill. However, the magnitude to which this happens is dependent on the weathering, and the size fraction that the mica reports to. The chlorite minerals clinochlore are present in the NLR (approximately 10%) as well as chamosite (approximately 10%). The effect the chlorites are dependent on the formation of the minerals. Generally, the internal sheet structures are held together more firmly than that of the micas and is not usually an issue for backfill.

The sulphate mineral gypsum is present in all the materials tested. Gypsum does not participate in the main hydration dynamics that produce the final strength of the backfill. However, it does participate in the early cementitious reactions by promoting early Ettringite formation. This is beneficial for long term strength and limits sulphate attack.

Backfill strength tests were conducted as in the UCS test matrix presented in Table 13-2. A cement dosage of between 7.5% and 9.5% will be required for high strength CPB over a 28-day curing time. A minimum of 4.5% binder will be required for the low strength CPB to meet the 28-day UCS strength targets. The required cement dosage for low strength CPT will be a minimum of 4.2%. (Note the definition of binder percentage being [mass dry binder / (mass total dry solids)].)

13.3 Recovery Estimates

The average recovery estimate used in the FS was determined from the pilot plant program (July 2018). Pilot leach testing had uranium extractions of 99.3%. The washing efficiency in the counter current decantation was greater than 99.6%. All other unit operations in the pilot testing had uranium recoveries of greater than 99.6%.

Section 17.3 provides additional discussion on recovery estimates within the plant. Metallurgical recovery of uranium was estimated by evaluating the recovery of the individual circuits and combining these into an overall recovery. Total net uranium metallurgical recovery is forecast to be 97.6%, from that section.

13.4 Metallurgical Variability

Eleven leaching tests were performed to test variability of the deposit. The grade of the 11 samples ranged from 0.51% U_3O_8 to 8.53% U_3O_8. The LG, MG, and HG samples resulted in leaching rates of 97.2% to 98.8%. The remaining eight tests had leaching rates ranging from 89.8% to 97.5%. Of the four samples that had low leaching rates, only the A3 zone is in the FS mine plan and this sample had a leaching rate similar to the LG sample (96.5% versus 97.2%).

13.5 Deleterious Elements

QEMSCAN analysis identified that there were no primary molybdenum-bearing minerals present; however, molybdenum may occur in chalcopyrite and galena solid solutions. Similarly, there were no arsenic-bearing minerals identified.

13.6 Comments on Section 13

Metallurgical test work conducted is appropriate to the mineralization type. Total net uranium metallurgical recovery is forecast to be 97.6%. There are no known deleterious elements in sufficient concentrations to affect marketing of the final YC product.

14.0 MINERAL RESOURCE ESTIMATE

14.1 Summary

Table 14-1 summarizes Mineral Resources at the Arrow Deposit, based on a uranium price of \$50/lb U_3O_8 at a cut-off grade of 0.25% U_3O_8. Measured Mineral Resources total 2.18 million tonnes (Mt) at an average grade of 4.35% U_3O_8 for a total of 209.6 million pounds (Mlb) of U_3O_8. Indicated Mineral Resources total 1.57 Mt at an average grade of 1.36% U_3O_8 for a total of 47.1 Mlb U_3O_8. Inferred Mineral Resources total 4.40 Mt at an average grade of 0.83% U_3O_8 for a total of 80.7 Mlb U_3O_8. The effective date of the Mineral Resource estimate is 19 July 2019.

Estimated block model grades are based on chemical assays only. The Mineral Resources were estimated by NexGen and audited by RPA. Mineral Resources are inclusive of Mineral Reserves. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated 10 May 2014 (CIM (2014) definitions) were used for Mineral Resource classification.

Table 14-1: Mineral Resource Estimate – 19 July 2019

Classification	Zone	Tonnage (t)	Grade (% U_3O_8)	Contained Metal (lb U_3O_8)
Measured	A2-LG	920,000	0.79	16,000,000
	A2-HG	441,000	16.65	161,900,000
	A3-LG	821,000	1.75	31,700,000
Measured Total	**–**	**2,183,000**	**4.35**	**209,600,000**
Indicated	A2-LG	700,000	0.79	12,200,000
	A2-HG	56,000	9.92	12,300,000
	A3-LG	815,000	1.26	22,700,000
Indicated Total	**–**	**1,572,000**	**1.36**	**47,100,000**
Measured + Indicated	A2-LG	1,620,000	0.79	28,100,000
	A2-HG	497,000	15.90	174,200,000
	A3-LG	1,637,000	1.51	54,400,000
Measured + Indicated Total	**–**	**3,754,000**	**3.10**	**256,700,000**
Inferred	A1	1,557,000	0.69	23,700,000
	A2-LG	863,000	0.61	11,500,000
	A2-HG	3,000	10.95	600,000
	A3-LG	1,207,000	1.12	29,800,000
	A4	769,000	0.89	15,000,000
Inferred Total	**–**	**4,399,000**	**0.83**	**80,700,000**

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.

2. Mineral Resources are reported at a cut-off grade of 0.25% U_3O_8.
3. Mineral Resources are estimated using a long-term uranium price of US$50/lb U_3O_8 and estimated mining, operating, and processing costs.
4. A minimum thickness width of one metre was used.
5. Tonnes are based on bulk density weighting.
6. Mineral Resources are inclusive of Mineral Reserves.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. Numbers may not sum due to rounding.

Estimated block model grades are based on chemical assays only. The Mineral Resources were estimated by NexGen and audited by RPA. Mineral Resources are inclusive of Mineral Reserves. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definition standards for Mineral Resources and Mineral Reserves dated 10 May 2014 (CIM [2014] definitions) were used for Mineral Resource classification.

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 other than what has been described in this report.

14.2 Resource Database

NexGen maintains a complete set of drill hole data plus other exploration data for the entire Property in an acQuire database. RPA was supplied with a drill hole database for the Arrow Deposit on the Property by NexGen. The Arrow Deposit resource database, dated 19 July 2019, includes drill hole collar locations (including dip and azimuth), assay, and lithology data from 566 drill holes totalling 318,096 m of drilling. Of the 566 holes completed, 45 drill holes were drilled on the South Arrow Discovery and were not used in the Mineral Resource estimate. The wireframe models representing the Arrow Deposit mineralized zones are intersected in 418 of 566 drill holes. A summary of the database records is provided in Table 14-2.

Table 14-2: Vulcan Database Record Count

Table Name	Number of Records
Collar	566
Survey	103,629
U_3O_8 Chemical Assays	83,483
Lithology	5,355
Density	5,850
One-metre Composites	20,137

14.3 Geological Interpretation and 3D Solids

Uranium mineralization at the Arrow Deposit occurs within, and proximal to, altered basement rocks that show varying degrees of clay, chlorite, and hematite alteration. Structures have been reactivated, and six main structural shear zones named A0, A1, A2, A3, A4, and A5 have been recognized, with the A2 and A3 shears hosting higher grade, thicker, and more continuous mineralization than the others as defined by current drilling. The mineralized area is 315 m wide with an overall strike of 980 m. Mineralization occurs 100 m below surface and extends to a depth of 950 m. The individual shear zones vary in thickness from two metres to 60 m.

Mineralization consists predominantly of uraninite / pitchblende that occurs as massive to semi-massive accumulations, and foliation controlled mineral replacements, and disseminations. A continuous zone of higher-grade mineralization in the A2 shear is known as the higher-grade A2 sub-zone (A2-HG).

Previous resource estimates also included a higher-grade sub-zone of mineralization within the A3 shear, however, based on 2019 infill drilling results and re-examination by NexGen resource geologists, the HG sub-zone within the A3 shear has been removed from this Mineral Resource estimate.

Geological interpretations supporting the Mineral Resource estimate were generated by NexGen personnel and audited for completeness and accuracy by RPA. Topographical surfaces, solids, and mineralized wireframes were modelled in Leapfrog Geo version 4.0, and then refined in Vulcan software. The extension distance for the mineralized wireframes was halfway to the next hole, or approximately 25 m vertically and horizontally past the last drill intercept.

High-grade domain models were created using a grade intercept limit equal to or greater than one metre, with a minimum grade of 5% U_3O_8, although lower grades were incorporated in places to maintain continuity and a minimum thickness of one metre.

Low-grade domain models were created using a lower-grade intercept limit equal to or greater than one metre, with a minimum grade-thickness product of one meter of 0.1% U_3O_8, or two metres at 0.05%. U_3O_8. RPA considers the selection of 0.05% U_3O_8 to be appropriate for constructing mineralized wireframe outlines, as this value well reflects the lowest cut-off grade that is expected to be applied for reporting of the Mineral Resources in an UG operating scenario and is consistent with other known deposits in the Athabasca Basin.

Sample intervals with assay results less than the nominated cut-off grade (internal dilution) were included within the mineralized wireframes if the core length was less than two metres, or if it allowed for modelling of grade continuity.

A total of 160 wireframes, of which seven HG wireframes were contained within two LG enveloping wireframes, were constructed within the A0, A1, A2, A3, A4, and A5 shear zones and were used in the Mineral Resource estimate, as shown in Table 14-3.

Table 14-3: Summary of Wireframe Models

Shear Zone	Domain Designation	Total No. Wireframes
A0	900 series	1
A1	100 series	26
A2-HG	1, 2, 3, 5, 6, 7 and 8	7
A2	200 series	53
A3	300 series	43
A4	400 series	30

Due to the limited number of drill holes, it was not possible to fully differentiate between the A4 and A5 shears. Mineralized intercepts in the A5 shear zone were therefore grouped into the A4 shear for the Mineral Resource estimate presented in this report. One wireframe was created in the A0 shear (the 900 wireframe), which has been combined with the A1 Mineral Resources for reporting purposes.

Figure 14-1 and Figure 14-2 show an isometric view of the wireframe models.

Figure 14-1: Isometric View of the Wireframe Models (Looking East)

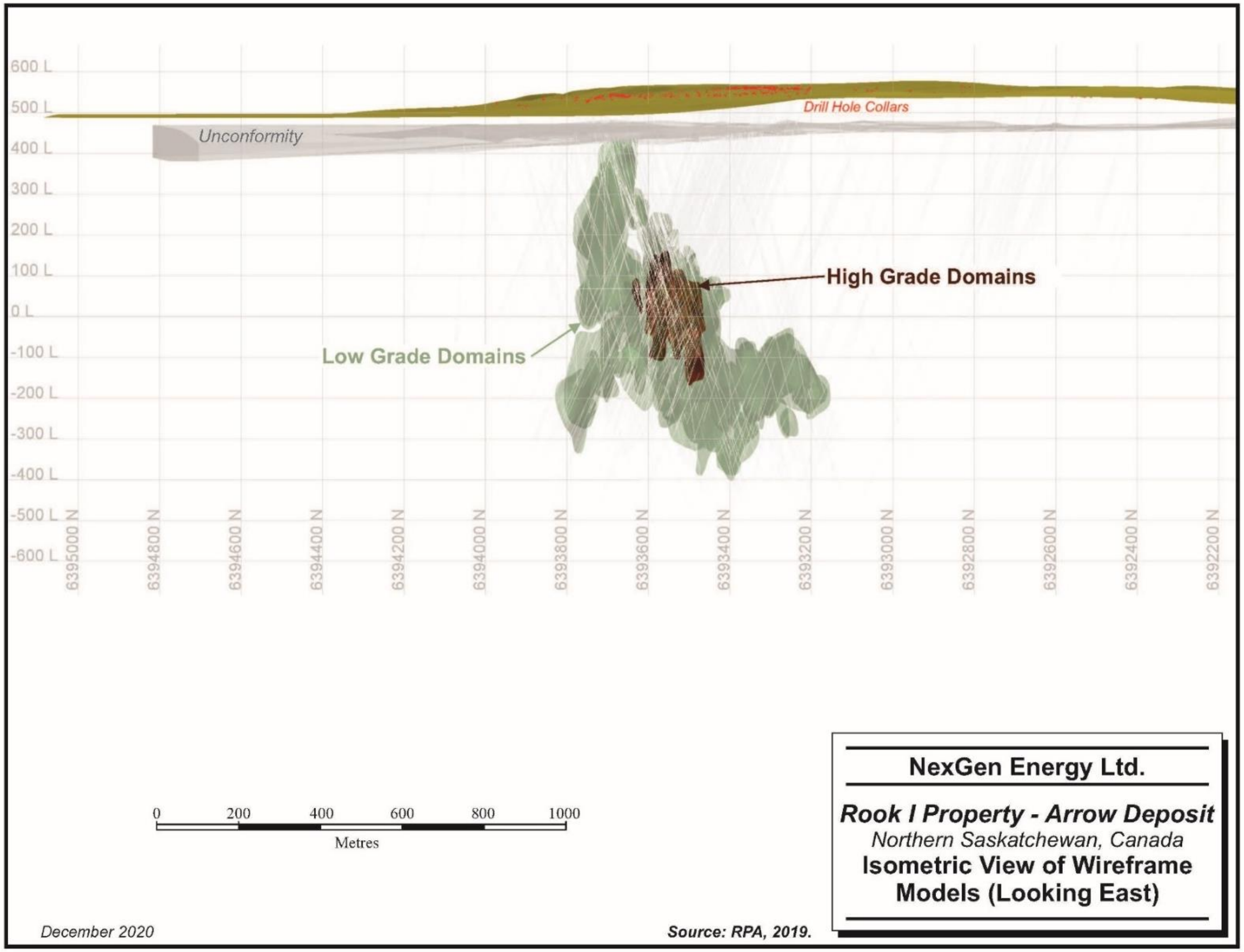
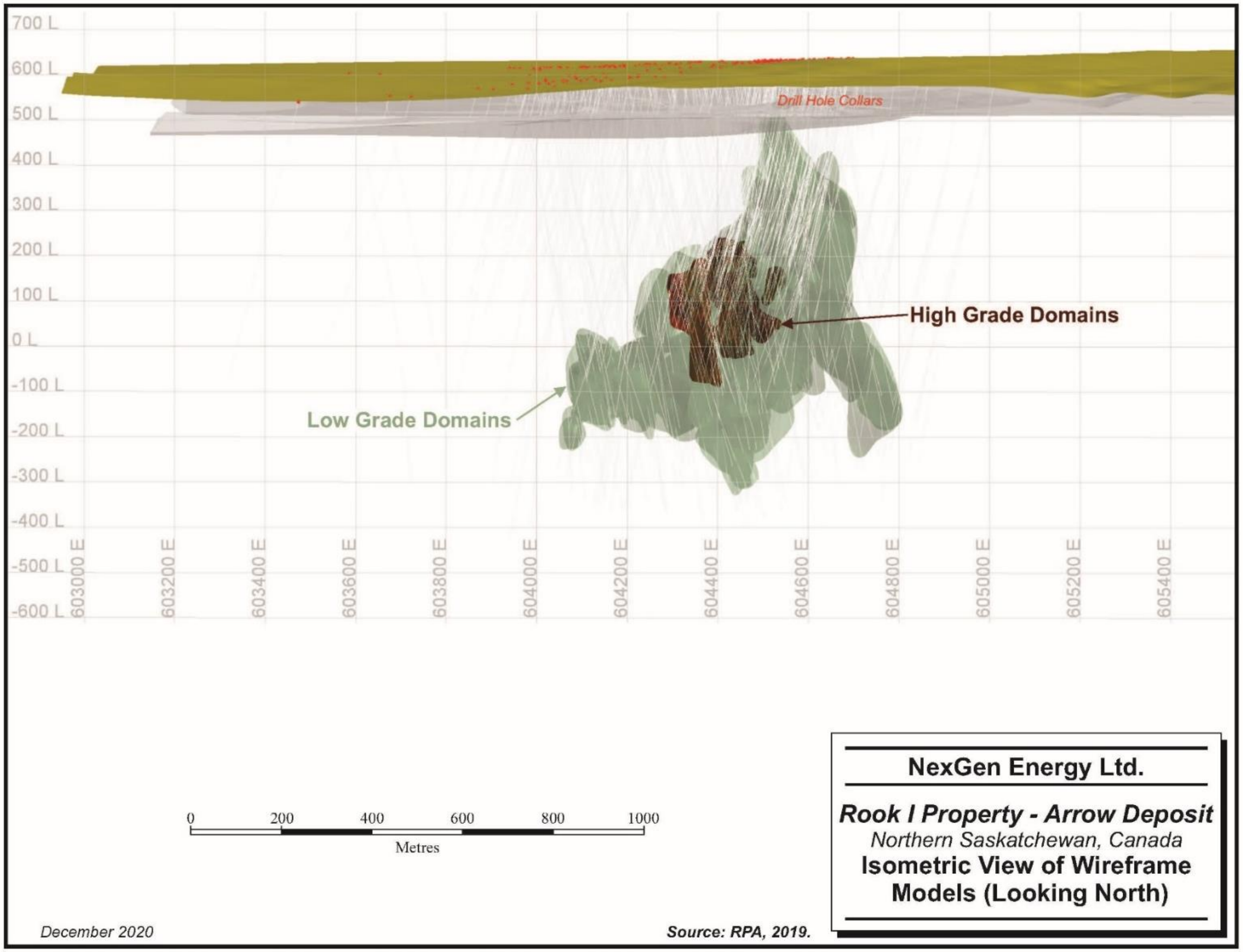


Figure 14-2: Isometric View of the Wireframe Models (Looking North)



14.4 Statistical Analysis

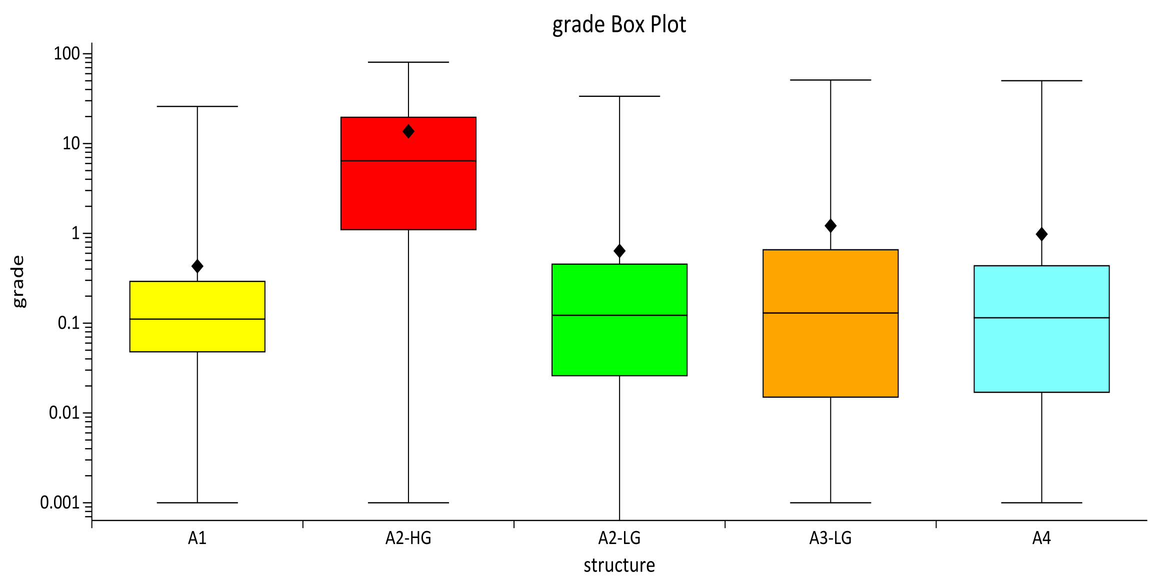
The mineralization wireframe models were used to code the drill hole database and to identify samples within the mineralized wireframes. These samples were extracted from the database on a group-by-group basis, subjected to statistical analyses for their respective domains, and then analyzed by means of histograms and probability plots.

A total of 29,232 samples are contained within the mineralized wireframes. Table 14-4 and Figure 14-3 present the descriptive and visual statistics for individual zones. SD and the coefficient of variation (CV)—which is the SD divided by the mean—is a measure of variability of the data.

Table 14-4: Summary Statistics of Uncapped % U_3O_8 Assays

Zone	Domain	Count	Min (% U_3O_8)	Max (% U_3O_8)	Mean (% U_3O_8)	Variance (% U_3O_8)	SD (% U_3O_8)	CV
A1	100 series	3,558	0.001	25.9	0.431	1.66	1.29	2.99
A2-HG	1,2,3,5,6,7,8	2,804	0.001	80.5	13.695	301.10	17.352	1.27
A2-LG	200 series	13,427	0	33.6	0.638	3.44	1.855	2.91
A3	300 series	7,409	0.001	51.0	1.216	14.19	3.767	3.10
A4	400 series	2,034	0.001	50.1	0.980	11.46	3.385	3.45
Total	–	29,232	0	80.5	2.035	49.55	7.039	3.46

Figure 14-3: Zone Box Plots



14.4.1 Grade Capping / Outlier Restrictions

Where the assay distribution is skewed positively or approaches log-normal, erratic HG 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.

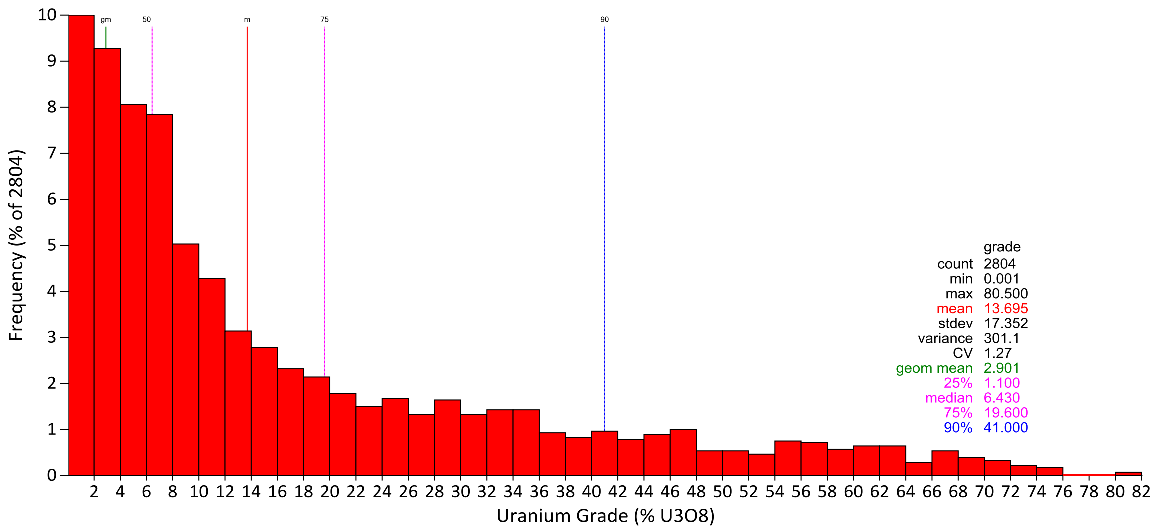
RPA is of the opinion that the influence of HG uranium assays must be reduced or controlled and uses a number of industry best practice methods to achieve this goal, including capping of HG values.

Assessing the influence of outliers involves a number of statistical analytical methods to determine an appropriate capping value. This includes preparation of frequency histograms, probability plots, decile analyses, and capping curves. Using these methodologies, NexGen geologists examined the selected capping values for each of the 160 mineralized domains in the Arrow Deposit.

Examples of the capping analysis are shown in Figure 14-4, Figure 14-5, Figure 14-6, and Figure 14-7 and applied to the data set for the mineralized zone domains.. . High grade outliers were capped at 1%, 2%, 3%, 4%, 5%, 6%, 8%, 10%, 15%, 25%, and 30% U_3O_8 in the domains, resulting in a total of 428 (1.5%) capped assay values (Table 14-5). No outlier assay values were identified in the HG domains; therefore, no capping was applied to the assays as each HG domain dataset was determined to be stationary and appropriate for interpolation, with the exclusion of the A2-HG8 which was capped at 30% U_3O_8. Capped assay statistics by zones are summarized in Table 14-6 and compared with uncapped assay statistics.

In RPA's opinion, the selected capping values are reasonable and have been correctly applied to the raw assay values for the Arrow Deposit Mineral Resource estimate.

Figure 14-4: Histogram and Log Probability of Resource Assays in A2-HG Domain



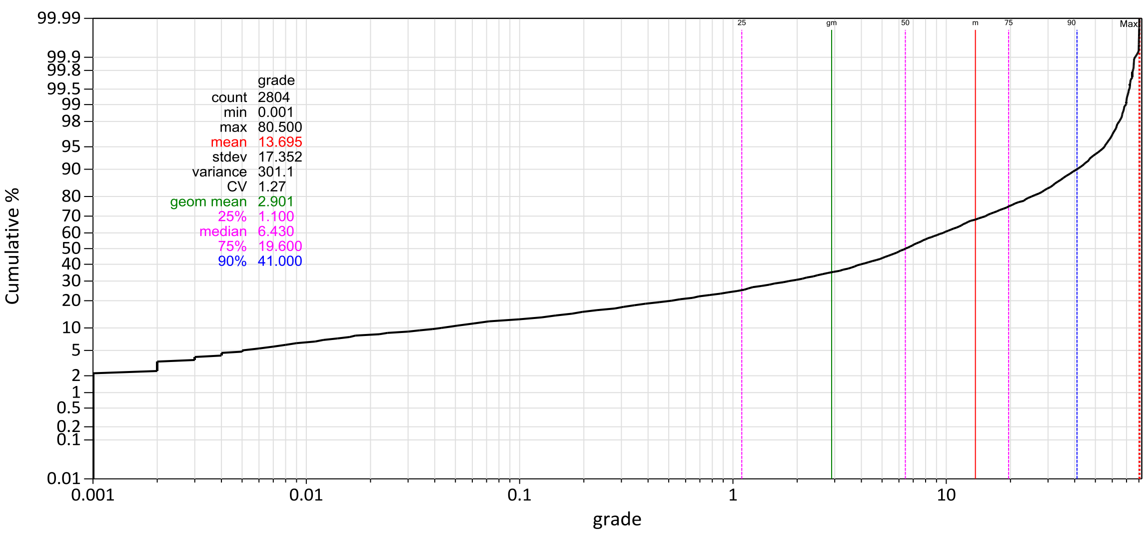
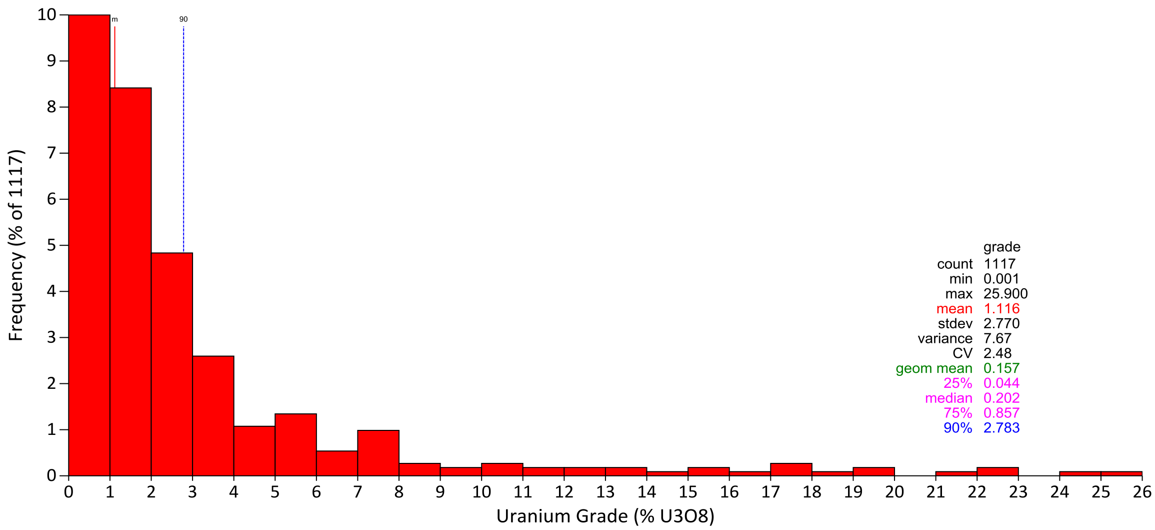


Figure 14-5: Histogram of Resource Assays in Other Domains (10% Cap)



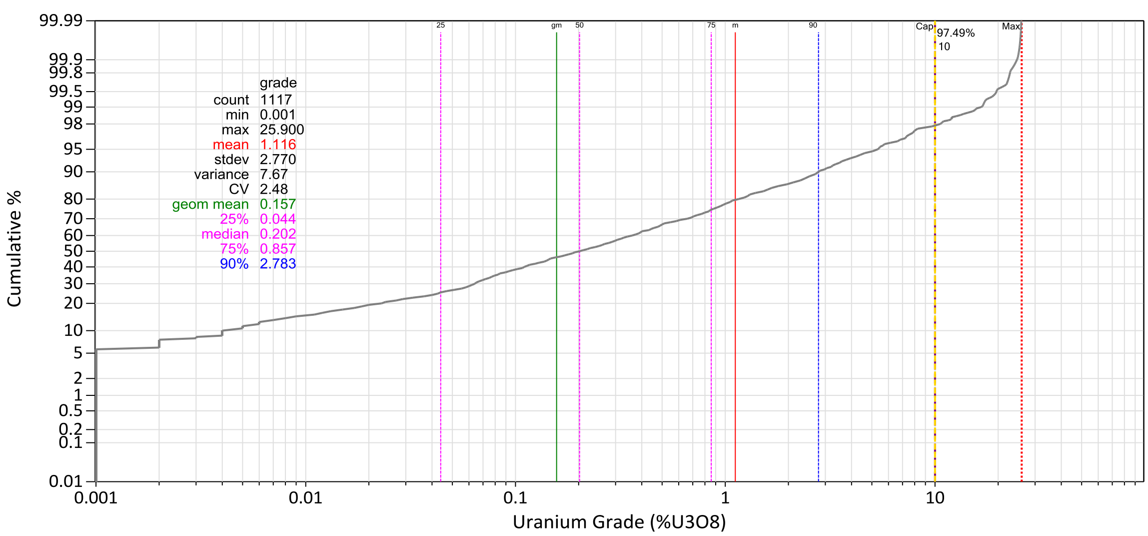
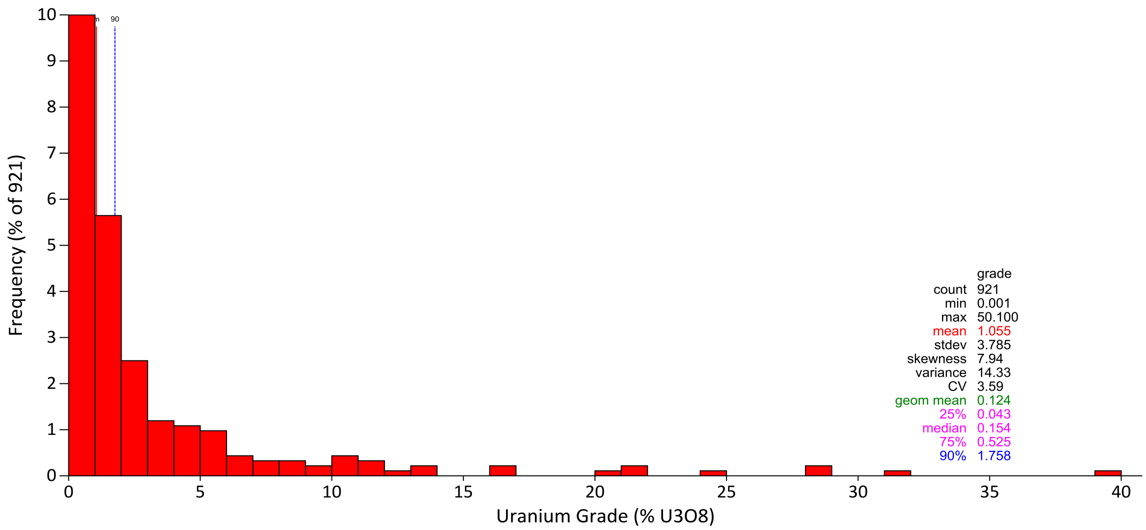


Figure 14-6: Histogram of Resource Assays in Other Domains (8% Cap)



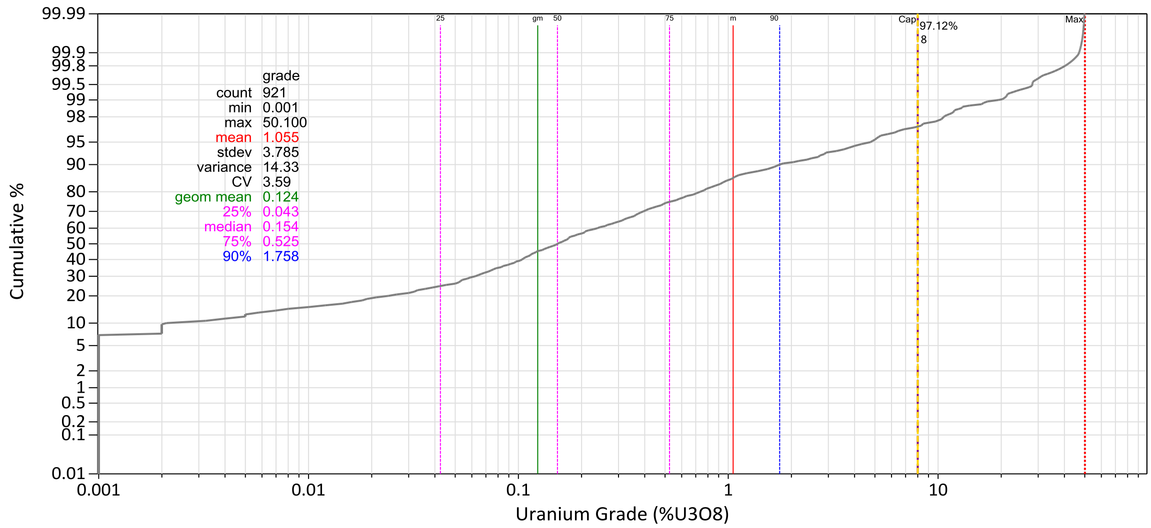


Figure 14-7: Histogram of Resource Assays in Other Domains (6% Cap)

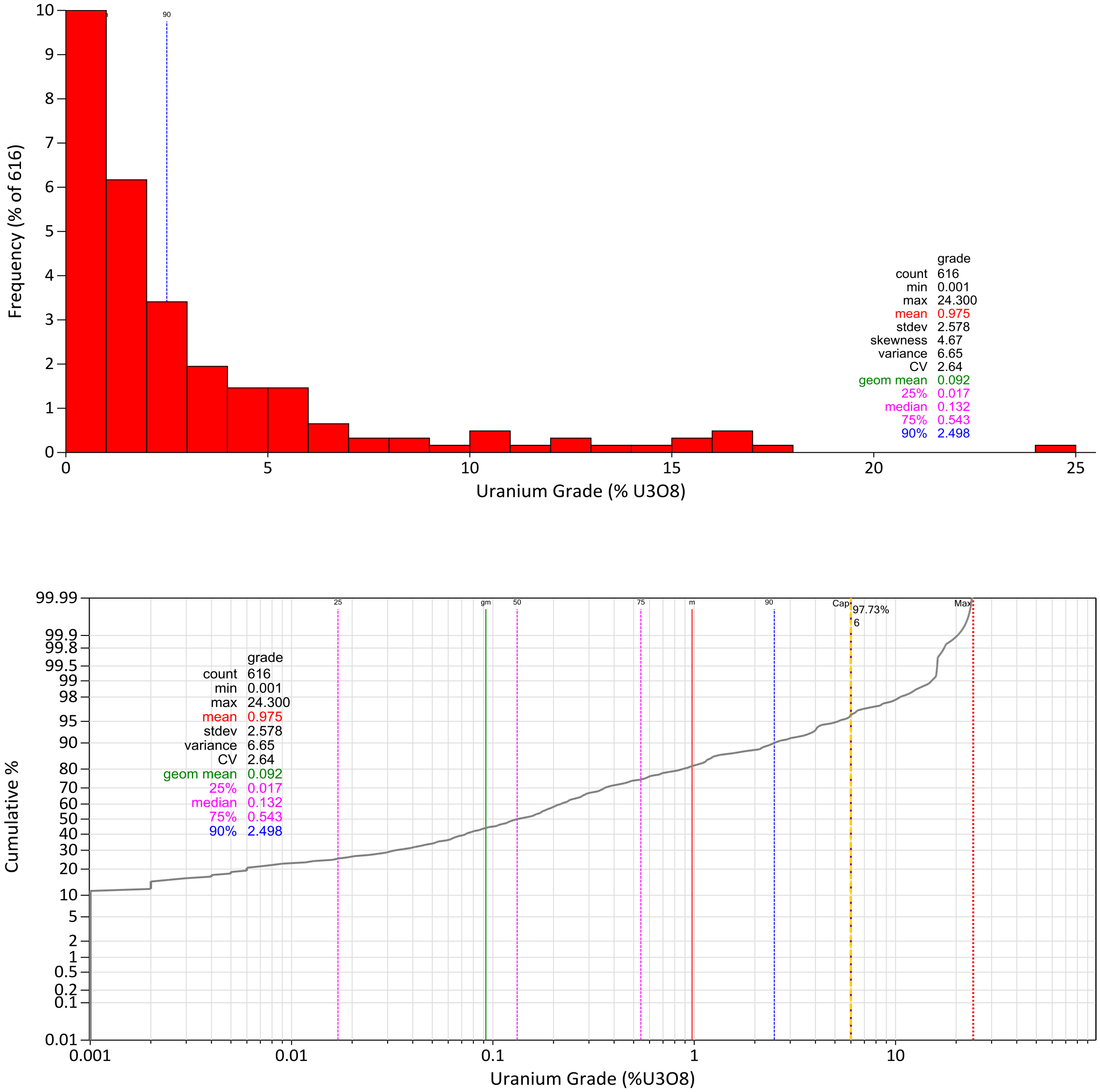


Table 14-5: Capping of Resource Assay Values by Zone

Zone	Cap Levels (% U_3O_8)	Number of Assays	Number Assays Capped	% Capped
A1	2,3,4,8, and 10	3,558	52	1.41%
A2-HG	30-HG8	2,804	6	0.21%
A2-LG	1,2,3,4,5,6,8,10, and 15	13,427	186	1.38%
A3	1,2,3,4,5,8,15,25, and 30	7,409	108	1.46%
A4	2,3,6,8, and 15	2,034	76	3.69%
Grand Total	–	29,232	428	1.45%

Table 14-6: Summary Statistics of Uncapped versus Capped Assays (A1 to A2)

Zone Descriptive Statistics	A1		A2-HG		A2-LG	
	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Number of Samples	3,558	3,558	2,804	2,804	13,427	13,427
Min (% U_3O_8)	0.00	0.00	0.00	0.00	0.00	0.00
Max(% U_3O_8)	25.90	10.00	80.50	80.50	33.60	15.00
Mean(% U_3O_8)	0.43	0.38	13.70	13.66	0.64	0.57
Variance (% U_3O_8)^2	1.66	0.76	301.10	299.30	3.44	1.90
SD (% U_3O_8)	1.29	0.87	17.35	17.30	1.86	1.38
CV	2.99	2.31	1.27	1.27	2.91	2.41
Number of Caps	0	52	0	6	0	186

Table 14-7: Summary Statistics of Uncapped versus Capped Assays (A3 to A4)

Zone Descriptive Statistics	A3		A4	
	Uncapped	Capped	Uncapped	Capped
Number of Samples	7,409	7,409	2,034	2,034
Min (% U_3O_8)	0.00	0.00	0.00	0.00
Max(% U_3O_8)	51.00	30.00	50.10	15.00
Mean(% U_3O_8)	1.22	1.12	0.98	0.72
Variance (% U_3O_8)^2	14.19	10.25	11.46	3.26
SD (% U_3O_8)	3.77	3.20	3.39	1.81
CV	3.10	2.86	3.45	2.52
Number of Caps	0	108	0	76

14.4.2 Composites

Composites were created from the capped raw assay values using the downhole compositing function of the Vulcan modelling software package. The composite lengths used during interpolation were chosen considering the predominant sampling length, the minimum mining width, style of mineralization, and continuity of grade.

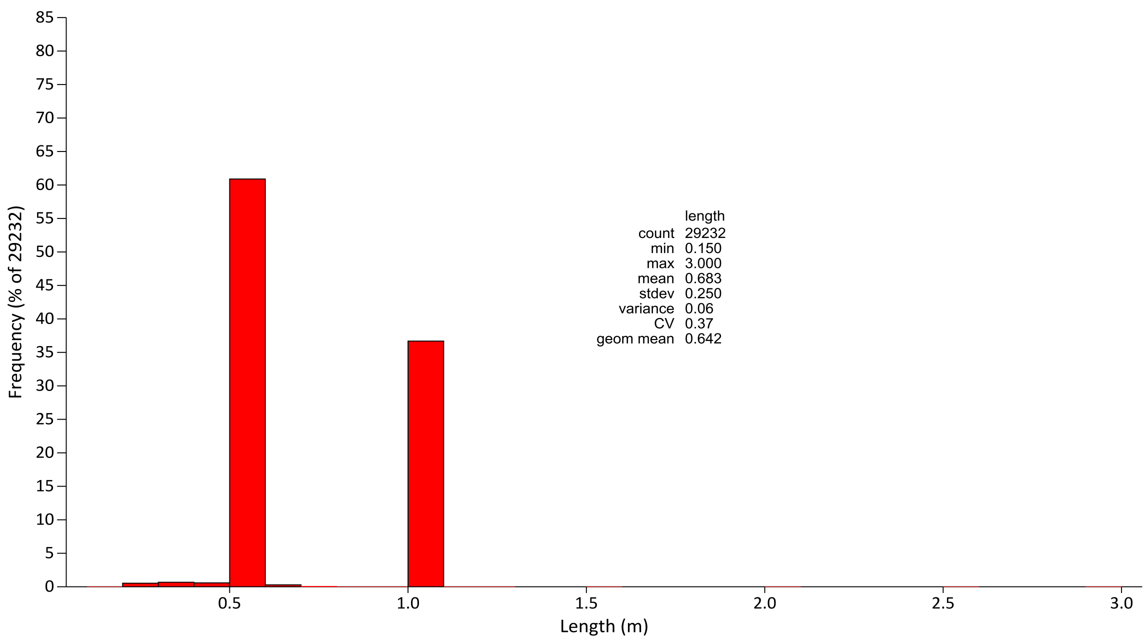
The raw assay data contains samples having irregular sample lengths. Sample lengths typically range from 15 cm to three metres within the wireframe models, with 60% of the samples taken at 0.5 m, 36% taken at one metre, and the remaining 4% taken at various other lengths, as shown in Figure 14-8.

There are 32 samples under 15 cm within the bounds of wireframes, which are not true sample lengths but rather the product of the wireframe not exactly snapping to the assay. These small samples represent approximately 0.1% of all samples taken; therefore, it is acceptable to keep the samples in the estimate due to the minimal influence the samples will have on the overall estimate. Furthermore, the estimation was completed using a length weighting, so the small samples will have a near negligible contribution to the estimation.

There are unsampled intervals within the wireframes that are considered to be internal dilution and nine of these unsampled intervals are over three metres. The maximum length of the unsampled intervals is 15.5 m from AR-16-096c1, which is between continuous mineralized intervals. The inclusion of this interval into the domain is a compromise between the mineralization and unmineralized sections of the hole within the domain alongside the mineralization observed in neighbouring holes.

Given this distribution, and considering the width of the mineralization, NexGen chose to composite to one metre lengths, which, in RPA's opinion, is appropriate for the Arrow Deposit Mineral Resource estimation.

Figure 14-8: Histogram of Sampling Length



Assays within the wireframe domains were composited starting at the first mineralized wireframe boundary from the collar, and resetting at each new wireframe boundary. Assays were capped prior to compositing. Composites less than 0.5 m, which were located at the bottom of the mineralized intercept, were excluded from the composite database. Table 14-8 shows the composite statistics by zone.

Table 14-8: Descriptive Statistics of Composite U_3O_8 Values by Domain

Zone	Domain	Count	Min (% U_3O_8)	Max (% U_3O_8)	Mean (% U_3O_8)	Variance (% U_3O_8)	SD (% U_3O_8)	CV
A1	100 series	2,300	0.001	10	0.401	0.72	0.849	2.12
A2-HG	1-8	1,761	0.001	75.7	13.665	257.80	16.056	1.17
A2-LG	200 series	8,982	0	15	0.573	1.54	1.240	2.16
A3	300 series	5,794	0	30	1.111	9.04	3.007	2.71
A4	400 series	1,300	0	15	0.670	2.14	1.463	2.18
Total	–	20,137	0	75.7	1.904	42.14	6.492	3.41

14.4.3 Variography

NexGen generated downhole, omni-directional, and directional variograms using the one-metre U_3O_8 composite values located within the A2-HG (except HG-06 and HG-08), LG-213, LG-301, and LG-312 mineralized domains. Variograms from the A2-HG-01 domain are shown in Figure 14-9, Figure 14-10, and Figure 14-11.

The variograms were used to support search ellipsoid anisotropy, linear trends observed in the data, and Mineral Resource classification decisions. The downhole variograms suggest a relative nugget effect of approximately 15%. Long range directional variograms were focused in the primary plane of mineralization, which commonly strikes northeast and dips steeply to the southeast. Most ranges were interpreted to be 10 m to 15 m.

Figure 14-9: Major Directional Variograms for A2-HG1 Domain

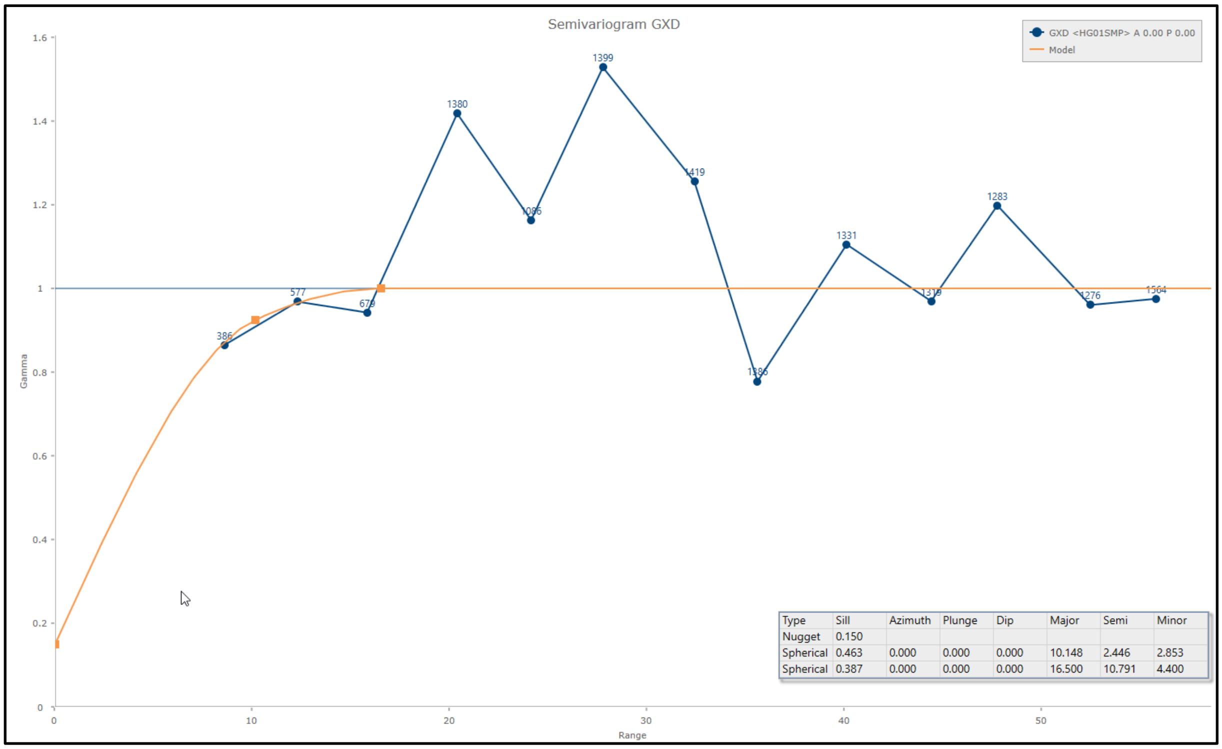


Figure 14-10: Semi-Major Directional Variograms for A2-HG1 Domain

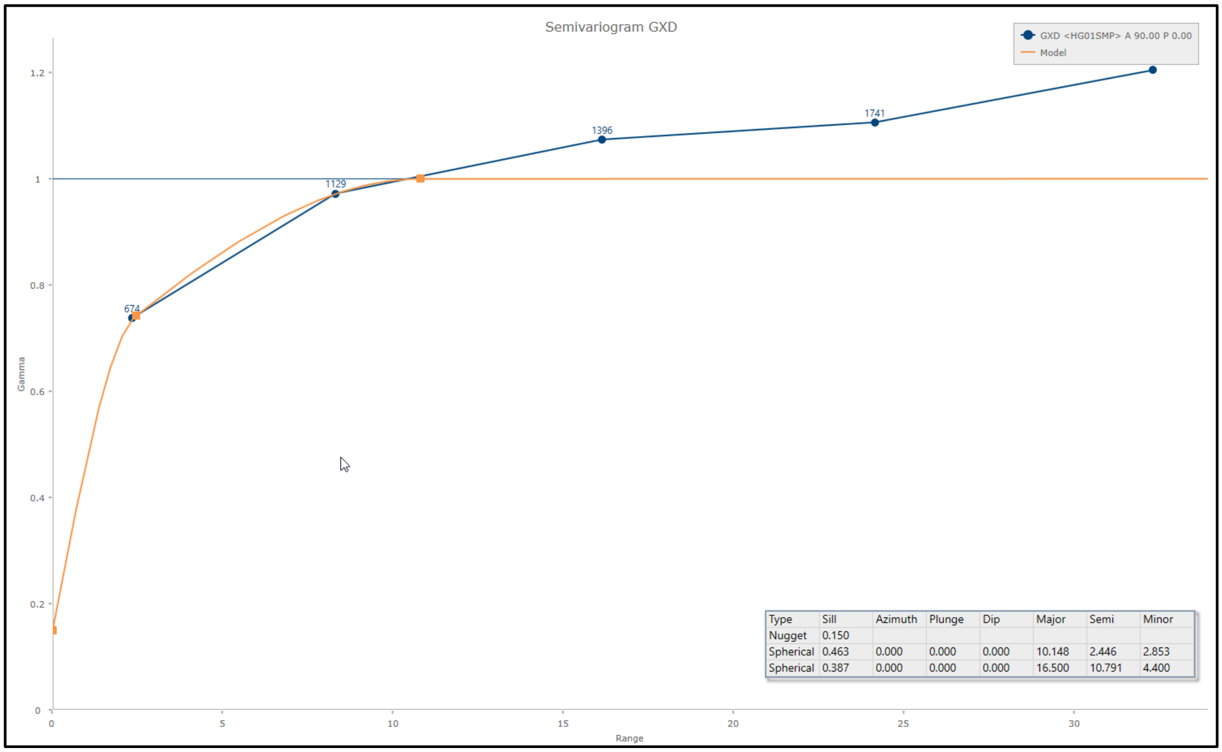
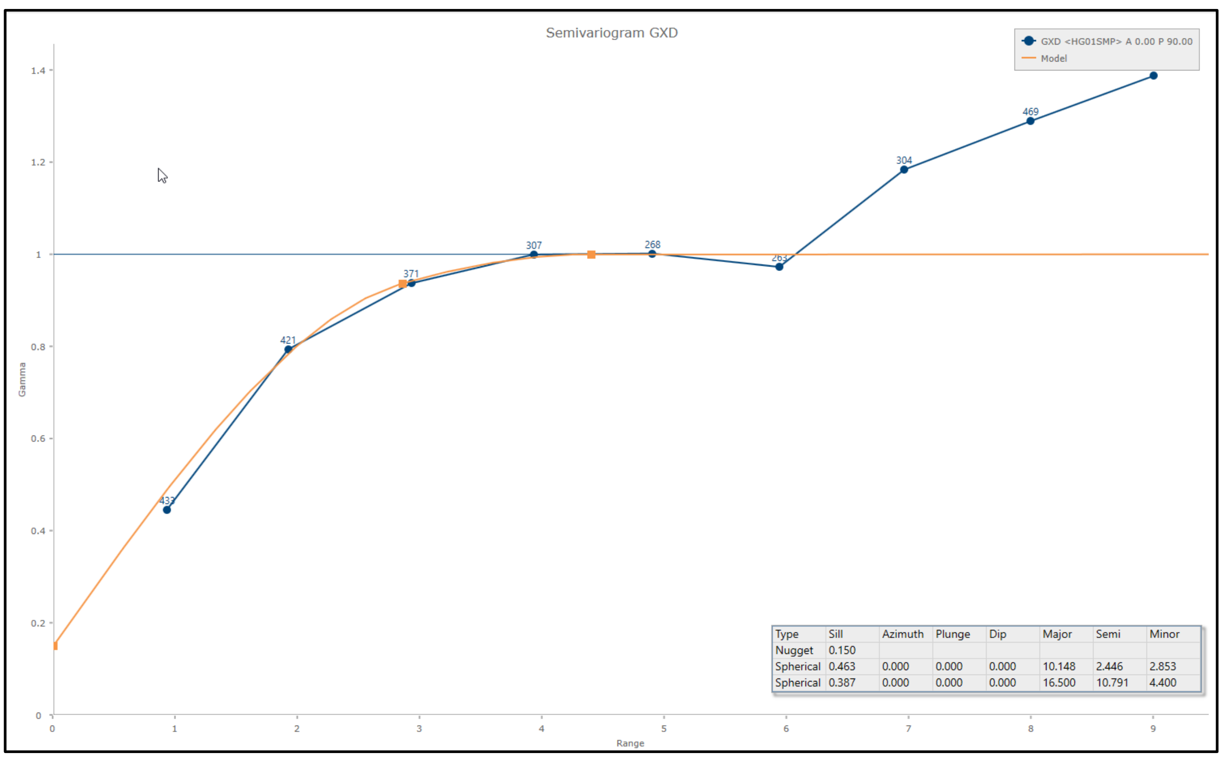


Figure 14-11: Minor Directional Variograms for A2-HG1 Domain



14.5 Block Model

Block models were created by NexGen geologists in Vulcan 12.0 to support the Mineral Resource estimate for the uranium deposits at the Property. Sub-blocking was used to give a more accurate volume representation of the wireframes using a parent block size of 4.0 m (along strike) by 4.0 m (across strike) by 4.0 m (vertical height) and sub-blocks that measured 1.0 m (along strike) by 1.0 m (across strike) by 1.0 m (bench height).

The model origin for the Arrow Deposit (lower-left corner at lowest elevation) is at UTM coordinates 604,072.0 mE, 6,393,061 mN and -500 m elevation. The model fully enclosed the modelled resource wireframes and is oriented with an azimuth of 57°, dip of 0.0°, and a plunge of 0.0° to align with the overall strike of the mineralization within the given model area. A summary of the block model extents is provided in Figure 14-12.

A number of attributes were created to store such information as bulk density, estimated uranium grades, wireframe code, Mineral Resource classification, as listed in Table 14-9.

Figure 14-12: Arrow Deposit Block Model Dimensions

Origin		**Value**
Xmin		604,072
Ymin		6,393,061
Zmin		-500
X Extents		1,100
Y Extents		500
Z Extents		1,100
Schema		**Value**
Parent		
	DX	4
	DY	4
	DZ	4
	NX	275
	NY	125
	NZ	275
Sub-Block		
	DX	1
	DY	1
	DZ	1
	NX	1,100
	NY	500
	NZ	1,100
Number of Blocks		10,172,438
Model Rotation		**Value**
Bearing		57
Plunge		0
Dip		0
Project Units		Metres
Coordinate System		NAD83 UTM Zone 12N

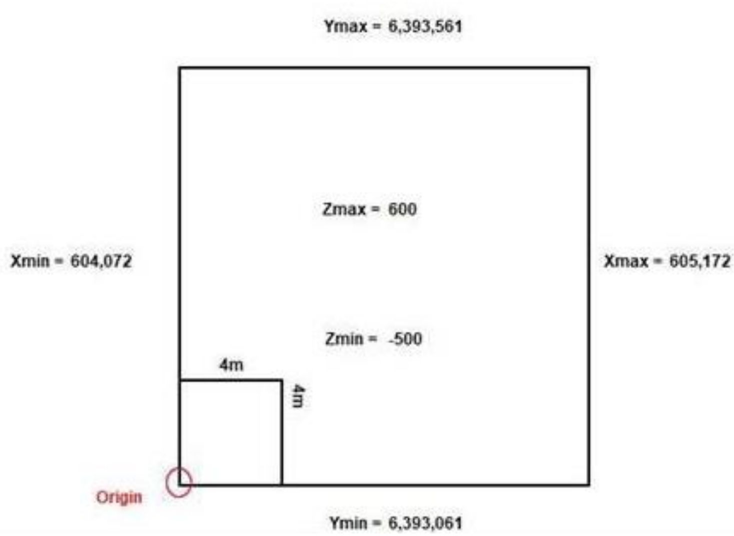


Table 14-9: Arrow Deposit Block Model Parameters and Variables

Variable	Data Type	Default Value	Description
den	Double (Real * 8)	-99	Density
gxd_d	Double (Real * 8)	-99	Equal to gxd / den
gxd	Double (Real * 8)	-99	Grade (raw) x density
grade_id2	Double (Real * 8)	-99	$\%U_3O_8$ interpolated grade ID^2
grade_id3	Double (Real * 8)	-99	$\%U_3O_8$ interpolated grade inverse distance cubed (ID^3)
grade_ok	Double (Real * 8)	-99	$\%U_3O_8$ interpolated grade OK
nsamp	Short (Integer * 2)	-99	Number of samples per estimate
nholes	Short (Integer * 2)	-99	Number of holes per estimate
est_avg_dist	Double (Real * 8)	-99	Average cartesian distance to samples per est.
est_samp_dist	Double (Real * 8)	-99	Distance to nearest sample per est.
nn	Double (Real * 8)	-99	Nearest neighbour (NN) grade
nn_distance	Double (Real * 8)	-99	Distance to NN
est_flag_id	Integer (Integer * 4)	-99	Estimation flag for ID
est_flag_ok	Integer (Integer * 4)	-99	Estimation flag for OK
ore	Integer (Integer * 4)	-99	Mineralized Domain Number
krig_var	Double (Real * 8)	-99	Kriging variance variable
blk_var	Double (Real * 8)	-99	Block variance variable
krig_eff	Double (Real * 8)	-99	Kriging efficiency variable
class	Double (Real * 8)	-99	Classification (1= Indicated)

RPA considers the Arrow Deposit block model parameters to be acceptable for a Mineral Resource estimate.

14.6 Estimation / Interpolation Parameters

For the A2-HG domains (excluding A2-HG6 and A2-HG8), search ellipsoid geometry was oriented into the structural plane of the mineralization, as indicated by the variography. The search was assisted by the use of a dynamic (unfolding) function in Vulcan, which allowed the search ellipsoid to stay subparallel to the orientation of the mineralized zone trend as it varies with location.

For the remaining domains, the interpolation strategy involved setting up search parameters in a series of two estimation runs for each individual domain. Of the 160 domains, only A1-LG grade domains 100, 101, 118, and 122 required a second pass search.

Search ellipse dimensions were chosen following a review of drill hole spacing and interpolation efficiency. First- and second-pass search ellipses maintained a 5:5:1 anisotropic ratio. The major axis of the search ellipses was oriented parallel to the dominant north-easterly trend of the domains. The semi-major axis was oriented horizontally, normal to the major axis (across strike). The minor axis was oriented with a plunge range of 0° to -53°, and a dip ranging from -76° to -90°.

For the first pass, the variables density (D) and grade multiplied by density (GxD) were interpolated using OK in the A2-HG domains (excluding A2-HG6 and A2-HG8), and A2-LG domains 206 and 213 (LG enveloping domains). ID^2 was used on all remaining mineralized domains.

Estimates used a minimum of one to three, to a maximum of 50 composites per block estimate, with the majority of the domains using a maximum of two composites per drill hole. The sample selection criteria were established through sensitivity testing that compared the estimated block means of each domain to the composited mean.

Unsampled intervals and samples below detection limit within the domains were assigned a grade of zero and considered to be internal dilution. Hard boundaries were used to limit the use of composites between domains. Block grade (GxD_D) was derived by dividing the interpolated GxD value by the interpolated density (D) value for each block.

When the first search was not enough to estimate all of the blocks in a domain, the minimum number of composites required for estimate was reduced by one. All blocks in the domains were populated by the second pass.

In order to reduce the influence of very HG composites, grades greater than a designated threshold level for the domains were restricted to a search ellipse dimension of 25 m by 25 m by 5 m (which is a high yield restriction). The threshold grade levels were chosen based on basic statistics and visual inspections of the apparent continuity of very high grades within each domain. This indicated the need to limit their influence to approximately one-quarter of the distance of the main search. Interpolation parameters are listed in Table 14-10 for the Arrow Deposit Mineral Resource domains.

Table 14-10: Block Estimate Search Strategy by Domain

Domain	Estimation Type	Cap ($\%U_3O_8$)	High Yield Restriction ($\%U_3O_8$)	Unfold File	Bearing (Azimuth)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)
1	OK	N/A	N/A	HG01_PROJECTION.tetra	0	0	0	25	15	5
2	OK	N/A	N/A	HG02_PROJECTION.tetra	0	0	0	30	15	15
3	OK	N/A	N/A	HG03_PROJECTION.tetra	0	0	0	35	30	10
5	OK	N/A	N/A	HG05_PROJECTION.tetra	30	0	0	30	30	10
6	ID^2	N/A	N/A	–	57	0	90	50	50	10
7	OK	N/A	N/A	HG07_PROJECTION.tetra	0	0	0	35	25	10
8	ID^2	30	N/A	–	57	0	90	35	35	10
206	ID^2	15	N/A	–	55	0	90	55	55	15
213	OK	5	N/A	LG213_PROJ.tetra	30	0	0	75	50	25
301	OK	30	N/A	–	55	0	0	45	45	65
312	OK	25	N/A	–	55	0	0	45	20	65
100	ID^2	2	N/A	–	233	0	90	100	100	20
101	ID^2	2	1	–	227	0	82	100	100	20
102	ID^2	3	1	–	233	0	85	100	100	20
103	ID^2	2	0.5	–	238	0	90	100	100	20
104	ID^2	4	N/A	–	238	0	90	100	100	20
105	ID^2	N/A	N/A	–	238	0	90	100	100	20
106	ID^2	N/A	N/A	–	238	0	90	100	100	20
107	ID^2	N/A	N/A	–	238	0	90	100	100	20
108	ID^2	10	N/A	–	238	0	90	100	100	20
109	ID^2	10	N/A	–	238	0	90	100	100	20
110	ID^2	2	1	–	238	0	90	100	100	20

Domain	Estimation Type	Cap ($\%U_3O_8$)	High Yield Restriction ($\%U_3O_8$)	Unfold File	Bearing (Azimuth)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)
111	ID^2	N/A	N/A	–	238	0	90	100	100	20
112	ID^2	N/A	N/A	–	238	0	90	100	100	20
113	ID^2	2	1	–	238	0	90	100	100	20
114	ID^2	N/A	1	–	238	0	90	100	100	20
115	ID^2	N/A	1	–	238	0	90	100	100	20
116	ID^2	N/A	N/A	–	238	0	90	100	100	20
117	ID^2	8	N/A	–	233	0	80	100	100	20
118	ID^2	N/A	3	–	236	0	90	100	100	20
119	ID^2	2	N/A	–	229	0	-87	100	100	20
120	ID^2	2	1	–	238	0	90	100	100	20
121	ID^2	4	3	–	238	0	90	100	100	20
122	ID^2	N/A	1	–	234	0	88	100	100	20
123	ID^2	2	1	–	223	0	76	100	100	20
124	ID^2	N/A	1	–	238	0	90	100	100	20
125	ID^2	N/A	1	–	238	0	90	100	100	20
202	ID^2	4	N/A	–	238	0	90	100	100	20
203	ID^2	3	N/A	–	238	0	90	100	100	20
204	ID^2	N/A	1	–	238	0	90	100	100	20
205	ID^2	8	N/A	–	238	0	90	100	100	20
207	ID^2	4	3	–	238	0	90	100	100	20
208	ID^2	4	N/A	–	232	0	82	100	100	20
209	ID^2	N/A	N/A	–	238	0	90	100	100	20
210	ID^2	10	N/A	–	232	0	86	100	100	20

Domain	Estimation Type	Cap ($\%U_3O_8$)	High Yield Restriction ($\%U_3O_8$)	Unfold File	Bearing (Azimuth)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)
211	ID^2	10	5	–	230	0	84	100	100	20
212	ID^2	4	N/A	–	238	0	90	100	100	20
214	ID^2	3	N/A	–	238	0	90	100	100	20
215	ID^2	N/A	N/A	–	238	0	90	100	100	20
216	ID^2	3	N/A	–	230	0	90	100	100	20
217	ID^2	N/A	N/A	–	238	0	90	100	100	20
218	ID^2	N/A	N/A	–	238	0	90	100	100	20
219	ID^2	N/A	N/A	–	238	0	90	100	100	20
220	ID^2	N/A	1	–	238	0	87	100	100	20
221	ID^2	N/A	N/A	–	235	0	85	100	100	20
222	ID^2	N/A	N/A	–	238	0	90	100	100	20
223	ID^2	3	1	–	238	0	90	100	100	20
224	ID^2	2	N/A	–	238	0	90	100	100	20
225	ID^2	10	4	–	221	0	84	100	100	20
227	ID^2	2	1	–	234	0	81	100	100	20
228	ID^2	3	N/A	–	238	0	90	100	100	20
229	ID^2	4	N/A	–	238	0	90	100	100	20
230	ID^2	N/A	N/A	–	238	0	90	100	100	20
231	ID^2	N/A	N/A	–	238	0	90	100	100	20
232	ID^2	2	1	–	238	0	90	100	100	20
233	ID^2	1	0.5	–	225	0	90	100	100	20
234	ID^2	N/A	N/A	–	238	0	90	100	100	20
235	ID^2	N/A	N/A	–	238	0	90	100	100	20

Domain	Estimation Type	Cap ($\%U_3O_8$)	High Yield Restriction ($\%U_3O_8$)	Unfold File	Bearing (Azimuth)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)
236	ID^2	3	N/A	–	234	0	83	100	100	20
237	ID^2	2	N/A	–	232	0	90	100	100	20
238	ID^2	N/A	N/A	–	238	0	-80	100	100	20
240	ID^2	2	0.5	–	238	0	90	100	100	20
241	ID^2	3	N/A	–	246	0	88	100	100	20
242	ID^2	4	N/A	–	238	0	90	100	100	20
245	ID^2	4	N/A	–	236	0	84	100	100	20
246	ID^2	N/A	N/A	–	238	0	90	100	100	20
248	ID^2	N/A	N/A	–	235	0	90	100	100	20
249	ID^2	5	2	–	238	0	90	100	100	20
250	ID^2	2	N/A	–	230	0	90	100	100	20
252	ID^2	2	N/A	–	238	0	90	100	100	20
253	ID^2	1	0.5	–	242	0	80	100	100	20
254	ID^2	2	N/A	–	238	0	90	100	100	20
255	ID^2	2	N/A	–	245	0	85	100	100	20
256	ID^2	4	N/A	–	232	0	90	100	100	20
257	ID^2	6	5	–	234	0	90	100	100	20
258	ID^2	4	2	–	233	0	88	100	100	20
259	ID^2	2	1	–	238	0	90	100	100	20
260	ID^2	N/A	N/A	–	238	0	90	100	100	20
302	ID^2	4	N/A	–	242	0	88	100	100	20
303	ID^2	N/A	1	–	238	0	90	100	100	20
304	ID^2	N/A	N/A	–	238	0	88	100	100	20

Domain	Estimation Type	Cap ($\%U_3O_8$)	High Yield Restriction ($\%U_3O_8$)	Unfold File	Bearing (Azimuth)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)
305	ID^2	N/A	N/A	–	238	0	90	100	100	20
306	ID^2	3	1	–	243	0	85	100	100	20
307	ID^2	N/A	2	–	238	0	90	100	100	20
308	ID^2	2	1	–	231	0	90	100	100	20
309	ID^2	25	10	–	241	0	86	100	100	20
310	ID^2	N/A	2	–	236	0	87	100	100	20
311	ID^2	3	N/A	–	227	0	88	100	100	20
313	ID^2	15	10	–	227	0	85	100	100	20
314	ID^2	N/A	N/A	–	238	0	90	100	100	20
315	ID^2	N/A	1	–	231	0	-85	100	100	20
316	ID^2	1	N/A	–	238	0	90	100	100	20
317	ID^2	N/A	N/A	–	231	0	-87	100	100	20
318	ID^2	15	10	–	238	0	85	100	100	20
319	ID^2	3	2	–	238	0	90	100	100	20
320	ID^2	2	N/A	–	234	0	87	100	100	20
321	ID^2	3	N/A	–	233	0	90	100	100	20
322	ID^2	8	2	–	235	0	90	100	100	20
323	ID^2	4	N/A	–	238	0	90	100	100	20
324	ID^2	N/A	N/A	–	237	0	90	100	100	20
325	ID^2	3	N/A	–	240	0	87	100	100	20
326	ID^2	4	N/A	–	229	0	90	100	100	20
327	ID^2	2	N/A	–	226	0	90	100	100	20
328	ID^2	N/A	N/A	–	242	0	85	100	100	20

Domain	Estimation Type	Cap ($\%U_3O_8$)	High Yield Restriction ($\%U_3O_8$)	Unfold File	Bearing (Azimuth)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)
329	ID^2	N/A	1	–	239	0	86	100	100	20
330	ID^2	2	1	–	229	0	83	100	100	20
331	ID^2	5	N/A	–	240	0	80	100	100	20
332	ID^2	N/A	N/A	–	230	0	85	100	100	20
333	ID^2	N/A	N/A	–	232	0	82	100	100	20
334	ID^2	1	N/A	–	222	0	87	100	100	20
335	ID^2	N/A	N/A	–	238	0	90	100	100	20
336	ID^2	N/A	N/A	–	238	0	90	100	100	20
337	ID^2	2	N/A	–	238	0	90	100	100	20
338	ID^2	N/A	N/A	–	238	0	90	100	100	20
339	ID^2	4	2	–	238	0	90	100	100	20
340	ID^2	N/A	N/A	–	238	0	90	100	100	20
341	ID^2	N/A	N/A	–	238	0	90	100	100	20
342	ID^2	2	N/A	–	238	0	90	100	100	20
343	ID^2	4	2	–	238	0	90	100	100	20
402	ID^2	6	N/A	–	235	0	88	100	100	20
403	ID^2	15	N/A	–	232	0	80	100	100	20
404	ID^2	N/A	N/A	–	242	0	80	100	100	20
405	ID^2	2	N/A	–	235	0	90	100	100	20
406	ID^2	N/A	N/A	–	230	0	90	100	100	20
407	ID^2	6	2	–	233	0	85	100	100	20
408	ID^2	8	1	–	230	0	85	100	100	20
409	ID^2	3	1	–	225	0	80	100	100	20

Domain	Estimation Type	Cap (%U_3O_8)	High Yield Restriction (%U_3O_8)	Unfold File	Bearing (Azimuth)	Plunge (°)	Dip (°)	Major (m)	Semi (m)	Minor (m)
410	ID^2	6	4	–	227	0	85	100	100	20
411	ID^2	3	N/A	–	238	0	90	100	100	20
413	ID^2	2	1	–	235	0	75	100	100	20
414	ID^2	8	2	–	233	0	85	100	100	20
415	ID^2	3	1	–	232	0	87	100	100	20
416	ID^2	3	1	–	232	0	87	100	100	20
417	ID^2	N/A	N/A	–	239	0	85	100	100	20
418	ID^2	8	6	–	231	0	85	100	100	20
419	ID^2	6	4	–	231	0	85	100	100	20
420	ID^2	N/A	1	–	235	0	85	100	100	20
421	ID^2	N/A	N/A	–	230	0	85	100	100	20
422	ID^2	N/A	N/A	–	230	0	85	100	100	20
424	ID^2	2	1	–	238	0	90	100	100	20
425	ID^2	2	1	–	228	0	90	100	100	20
426	ID^2	N/A	N/A	–	238	0	90	100	100	20
427	ID^2	N/A	N/A	–	237	0	80	100	100	20
428	ID^2	N/A	N/A	–	233	0	87	100	100	20
429	ID^2	2	1	–	235	0	85	100	100	20
430	ID^2	N/A	N/A	–	235	0	75	100	100	20
431	ID^2	N/A	N/A	–	233	0	85	100	100	20
432	ID^2	N/A	1	–	240	0	85	100	100	20
434	ID^2	N/A	1	–	240	0	85	100	100	20
900	ID^2	2	1	–	234	0	80	100	100	20

14.7 Block Model Validation

RPA validated the block model using the following methods.

- Swath plots of composite grades (arw_1m_2019_q3_COMBINED-cmp_rpa_entry) versus block model grade estimates (arw_4x4x4_id2_ok_2019Q3_rev3_bmfoutput) and NN grades in the X, Y, and Z directions (see Figure 14-13, Figure 14-14, and Figure 14-15).
- Volumetric comparison of blocks versus wireframes.
- Visual inspection of block versus composite grades on plan, vertical cross section, and longitudinal section.
- Statistical comparison of block grades with assay and composite grades.

RPA found the grade continuity to be reasonable and confirmed that the block grades were reasonably consistent with local drill hole composite grades.

Figure 14-13: East-West (X) Swath Plot of Arrow Deposit

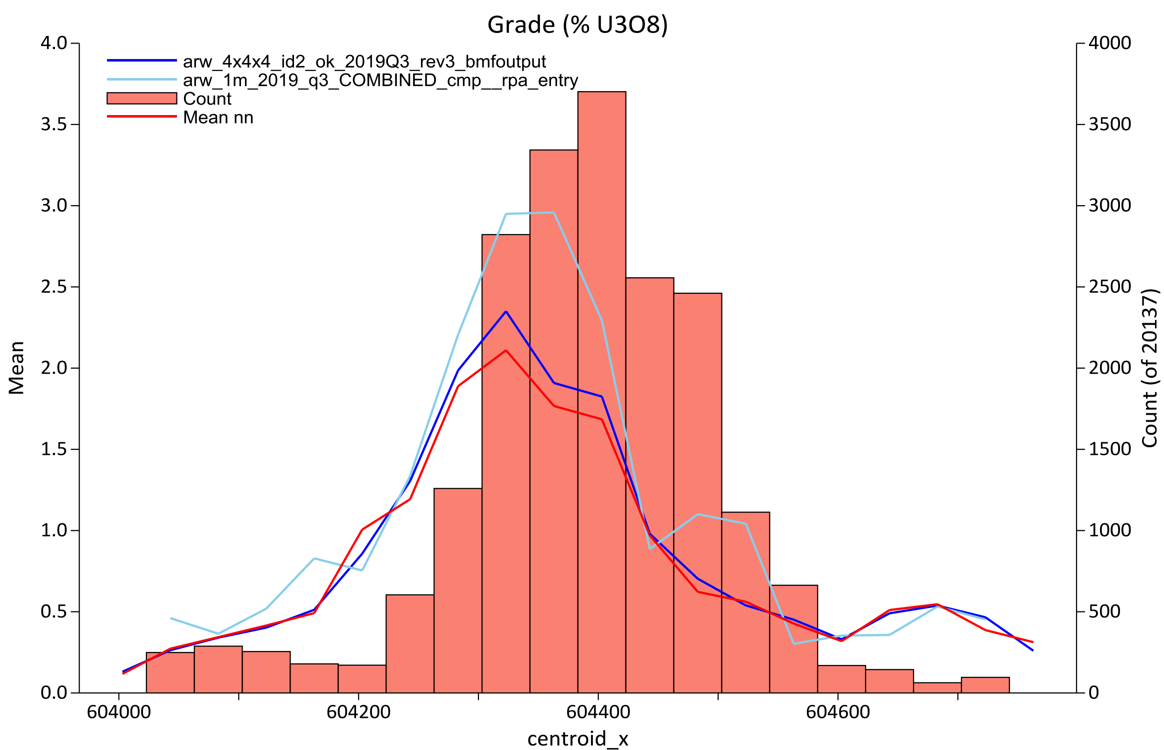


Figure 14-14: North-South (Y) Swath Plot of Arrow Deposit

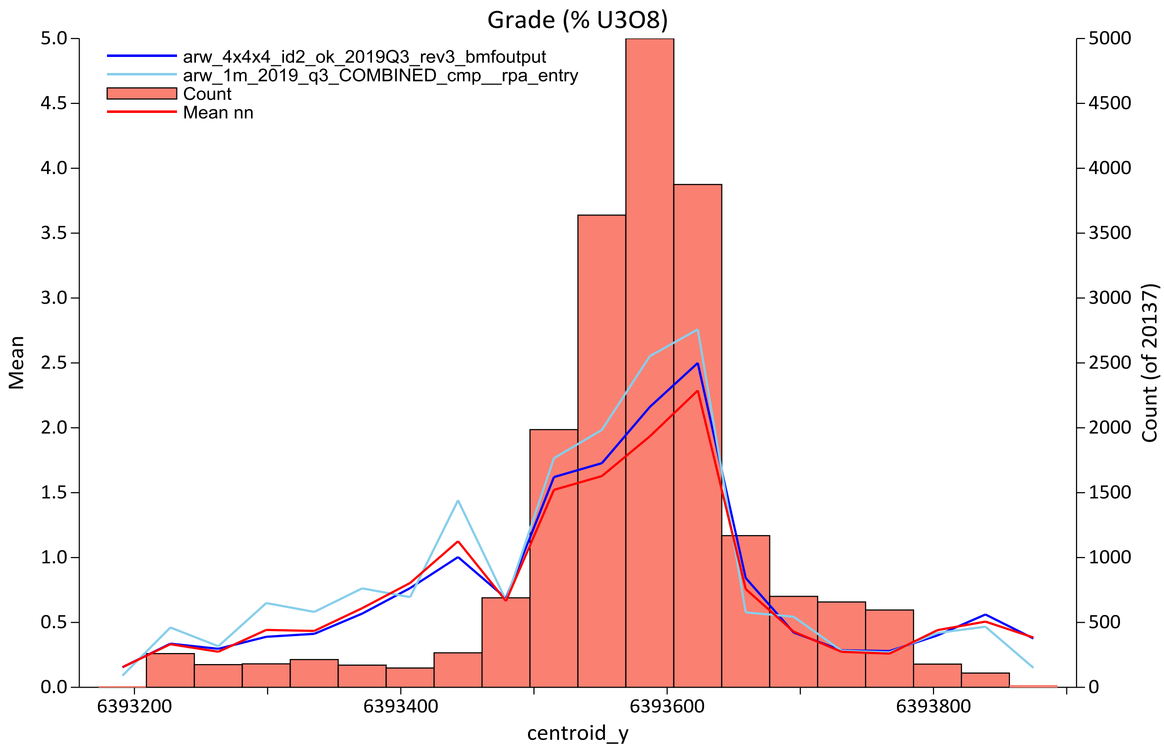
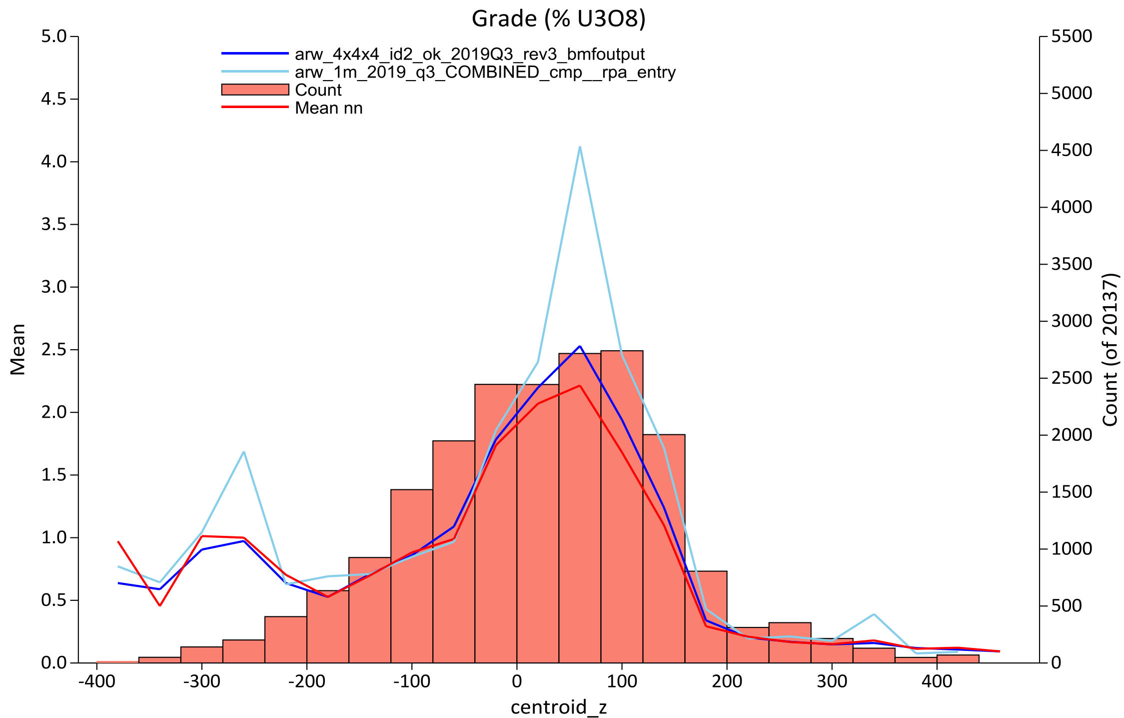


Figure 14-15: Vertical (Z) Swath Plot of Arrow Deposit



14.7.1 Volume Comparison

Wireframe volumes were compared to block volumes for each zone at the Arrow Deposit. This comparison is summarized in Table 14-11 and results demonstrate that there is good agreement between the wireframe and block model volumes with the difference being less than 1%.

Table 14-11: Volume Comparison

Zone	Wireframe Volume (m^3)	Block Model Volume (m^3)	% Difference
A1	1,521,286	1,520,946	0.02%
A2-HG	178,537	178,356	0.10%
A2-LG	1,556,711	1,556,812	-0.01%
A3-LG	1,460,039	1,458,767	0.09%
A4	524,381	524,056	0.06%
Total	5,240,955	5,238,937	0.04%

14.7.2 Visual Comparison

Block grades were visually compared with drill hole composites on cross sections, longitudinal sections, and plan views. Block and composite grades visually correlate well within the Arrow Deposit. Figure 14-16 is a vertical cross section and Figure 14-17 is a level plan showing blocks and drill hole composites; the grades within the A2-HG zone are colour-coded.

Figure 14-16: Vertical Section 4940e (5 m Window) A2-HG Domains

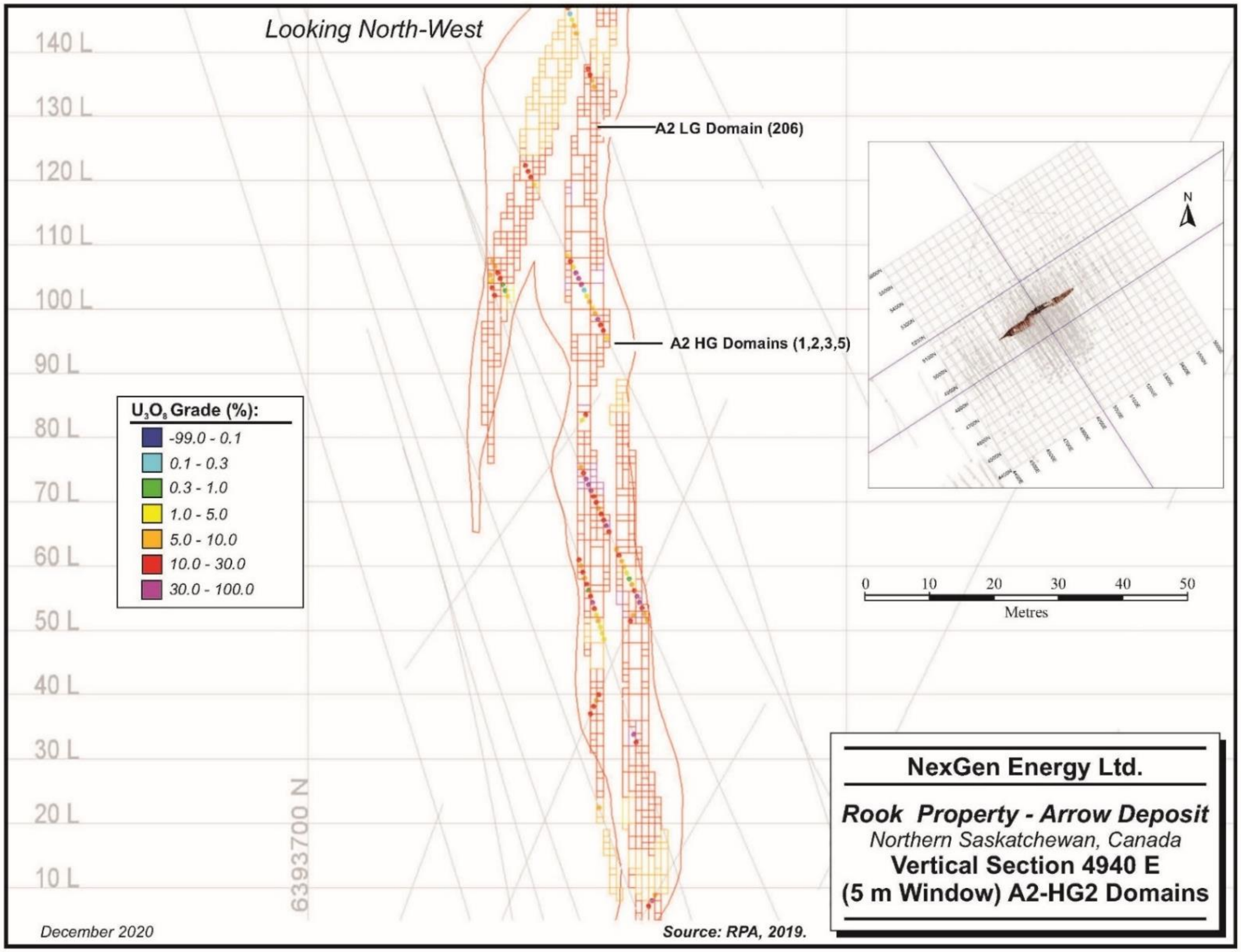
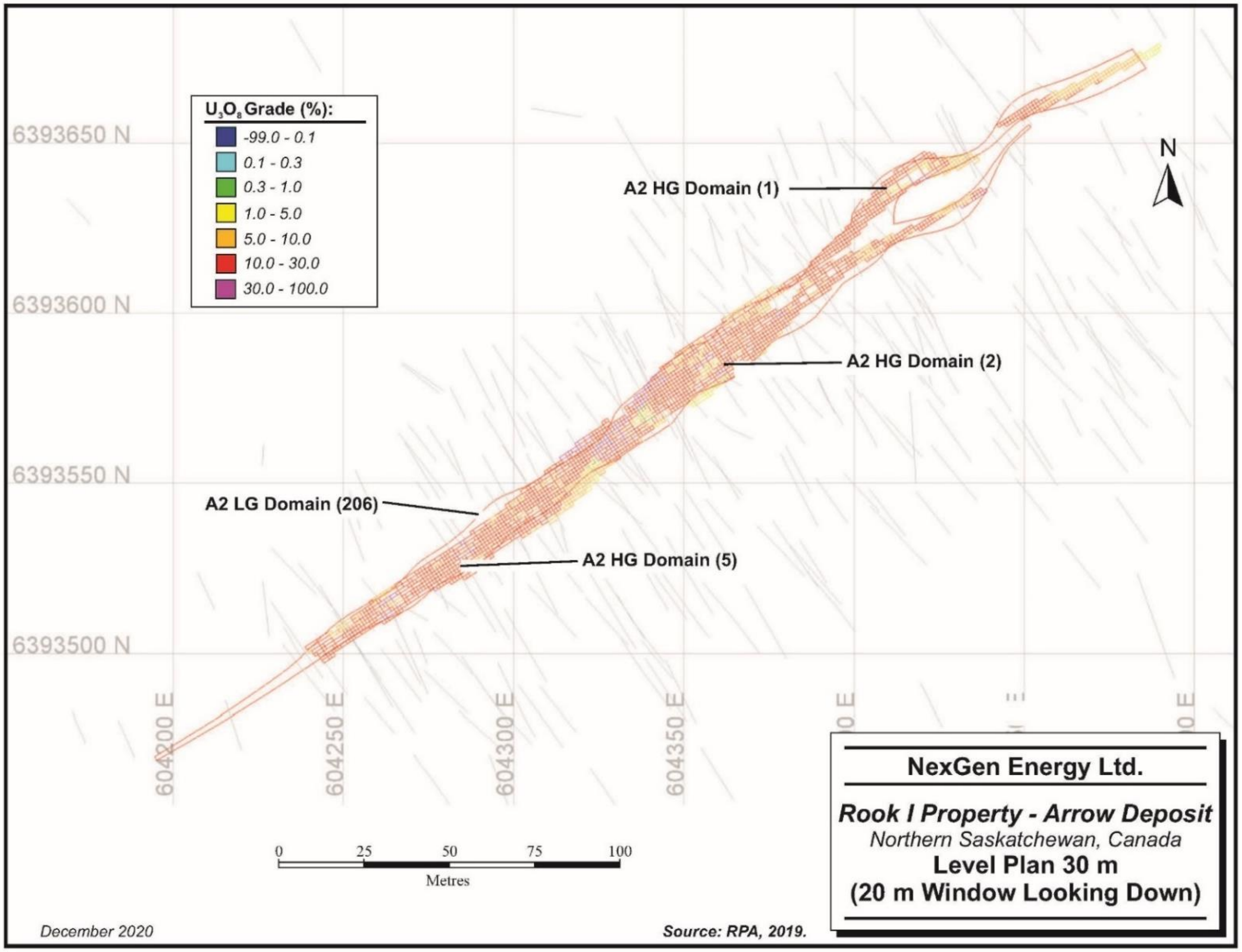


Figure 14-17: Level Plan 30 m (20 m Window Looking Down)



14.7.3 Statistical Comparison

Table 14-12 and Table 14-13 compare the statistics of the block grades with those of composite grades for all blocks, and composites within the Arrow Deposit domains. Block grades are weighted by density and tonnage for the blocks, whereas composite grades were not weighted by density.

Table 14-12: Statistics of Block Grades versus Composite Grades (A1, A2-HG, and A2-LG)

Zone Descriptive Statistics	A1		A2-HG		A2-LG	
	Comp	Block	Comp	Block	Comp	Block
Number of Samples	2,300	102,376	1,761	20,599	8,982	146,941
Min	0.00	0.00	0.00	0.04	0.00	0.00
Max	10.00	8.71	75.70	70.06	15.00	10.75
Mean	0.40	0.34	13.67	13.67	0.57	0.53
Variance	0.72	0.19	257.80	66.84	1.54	0.31
SD	0.85	0.44	16.06	8.18	1.24	0.56
CV	2.12	1.28	1.17	0.60	2.16	1.07

Table 14-13: Statistics of Block Grades versus Composite Grades (A3 and A4)

Zone Descriptive Statistics	A3		A4	
	Comp	Block	Comp	Block
Number of Samples	5,794	117,420	1,300	61,208
Min	0.00	0.00	0.00	0.00
Max	30.00	19.64	15.00	14.42
Mean	1.11	0.92	0.67	0.60
Variance	9.04	1.36	2.14	0.55
SD	3.01	1.17	1.46	0.74
CV	2.71	1.26	2.18	1.23

14.8 Cutoff Grade

To fulfill the NI 43-101 requirement of "reasonable prospects for eventual economic extraction", RPA estimated a potential UG mining cut-off grade for reporting of Mineral Resources. The cut-off grade selected uses assumptions based on historical and known operating costs for mines operating in the Athabasca Basin, and on previous studies of the Project.

In general, metal prices used for reserves are based on consensus, long term forecasts from banks, financial institutions, and other sources.

Table 14-14 summarizes RPA's cutoff grade estimate using a price of US$50/lb U_3O_8. The estimate is based on assumptions regarding process plant recovery, total operating costs, and the incremental component of operating costs.

Table 14-14: Arrow Deposit Cutoff Grade Calculation

Item	Units	Quantity
Price in US$/lb U_3O_8	US$/lb U_3O_8	50
Exchange Rate	US:CAD	1.0:0.75
Process Plant Recovery	%	97%
Revenue Royalty	%	7.25
Revenue Factor	$/%$U_3O_8$	1,330
Operating Costs		
Mining (including tailings stopes)	$/t proc	157
Processing	$/t proc	164
General and Administrative	$/t proc	67
Total Operating Cost	$/t proc	388
Incremental Operating Cost	$/t proc	357
Cutoff grade using Incremental Operating Cost	%U_3O_8	0.27
Reporting Cutoff Grade (rounded)	%U_3O_8	0.25

Table 14-15 and Figure 14-18, and Table 14-16 and Figure 14-19 demonstrate the sensitivity of the Arrow Deposit block model to various cut-off grades for Measured and Indicated (M&I) and Inferred Mineral Resources, respectively. RPA notes that, although there is some sensitivity of average grade and tonnes to cutoff grade, the contained metal is less sensitive.

Table 14-15: Arrow Deposit Measured + Indicated Mineral Resource Sensitivity to Cutoff Grade

Cut-off Grade (% U_3O_8)	Tonnes (t)	Grade (% U_3O_8)	Contained Metal (U_3O_8 lb)
0.25	**3,754,000**	**3.10**	**256,700,000**
0.3	3,580,000	3.24	255,600,000
0.5	2,867,000	3.94	249,400,000
1	1,703,000	6.15	231,000,000
2	961,000	9.82	208,200,000
2.5	798,000	11.37	200,100,000

Figure 14-18: Arrow Deposit Measured + Indicated Mineral Resource Tonnes and Grade at Various Cutoff Grades

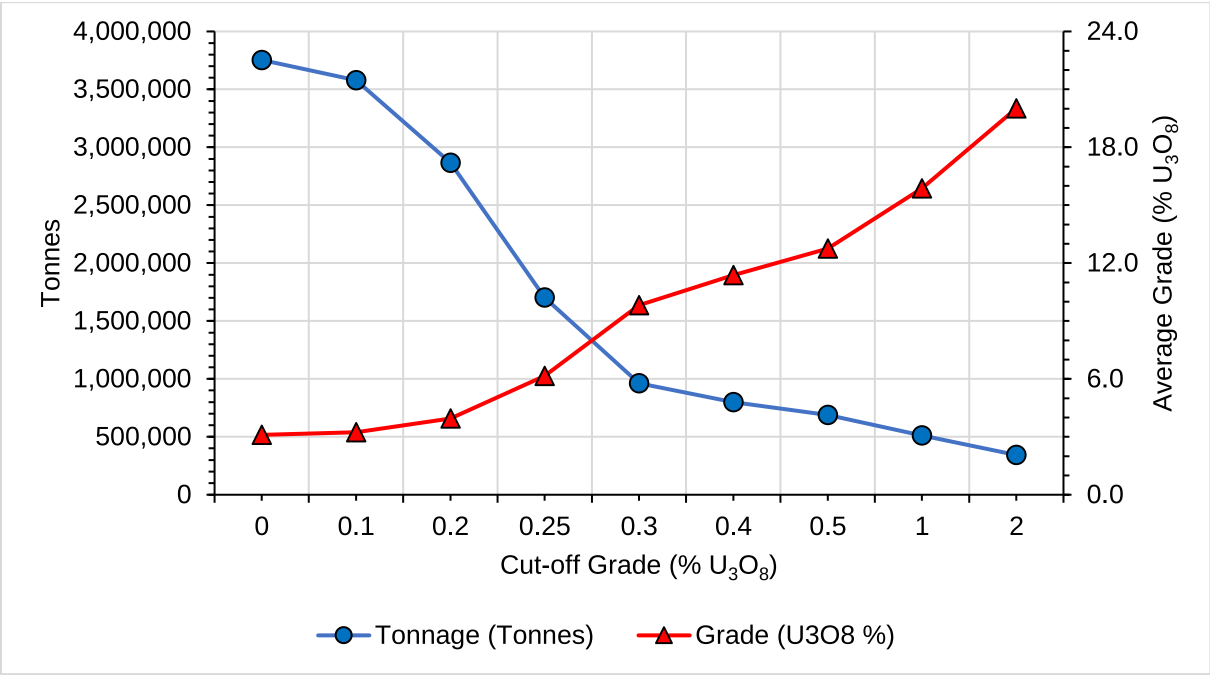
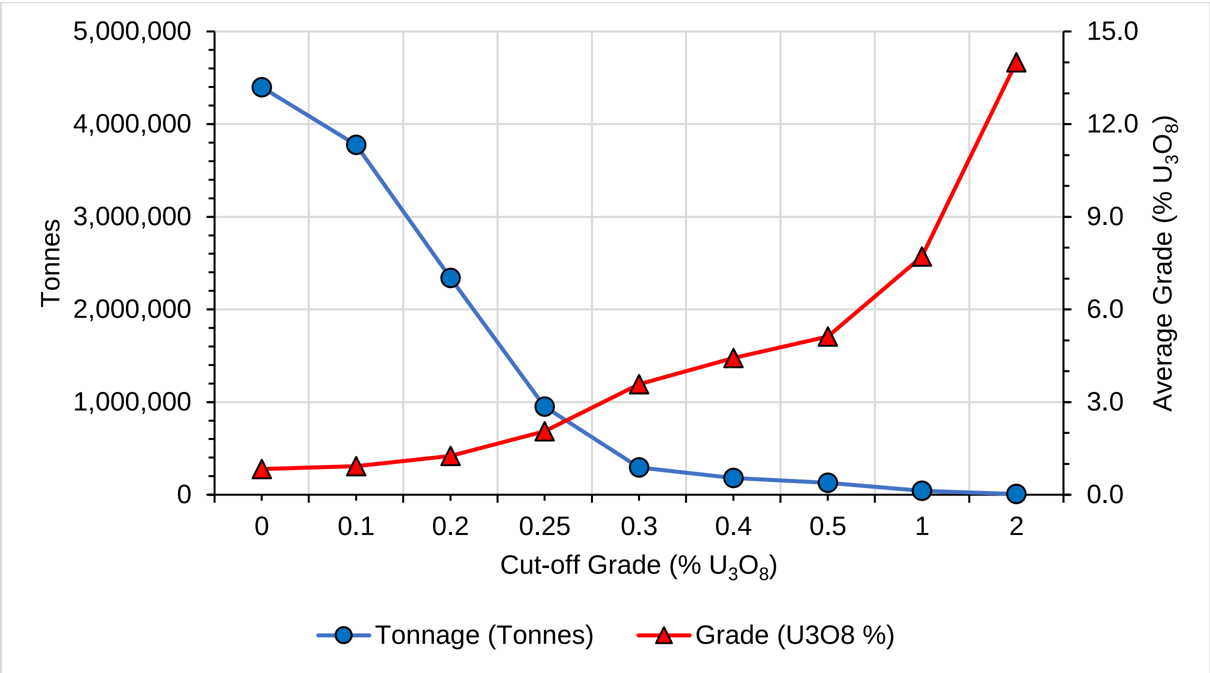


Table 14-16: Arrow Deposit Inferred Mineral Resource Sensitivity to Cutoff Grade

Cutoff Grade (% U_3O_8)	Tonnes (t)	Grade (% U_3O_8)	Contained Metal (U_3O_8 lb)
0.25	**4,399,000**	**0.83**	**80,700,000**
0.3	3,775,000	0.92	76,900,000
0.5	2,340,000	1.25	64,600,000
1	952,000	2.06	43,200,000
2	295,000	3.57	23,200,000

Figure 14-19: Arrow Deposit Inferred Mineral Resource Tonnes and Grade at Various Cutoff Grades



14.9 Classification

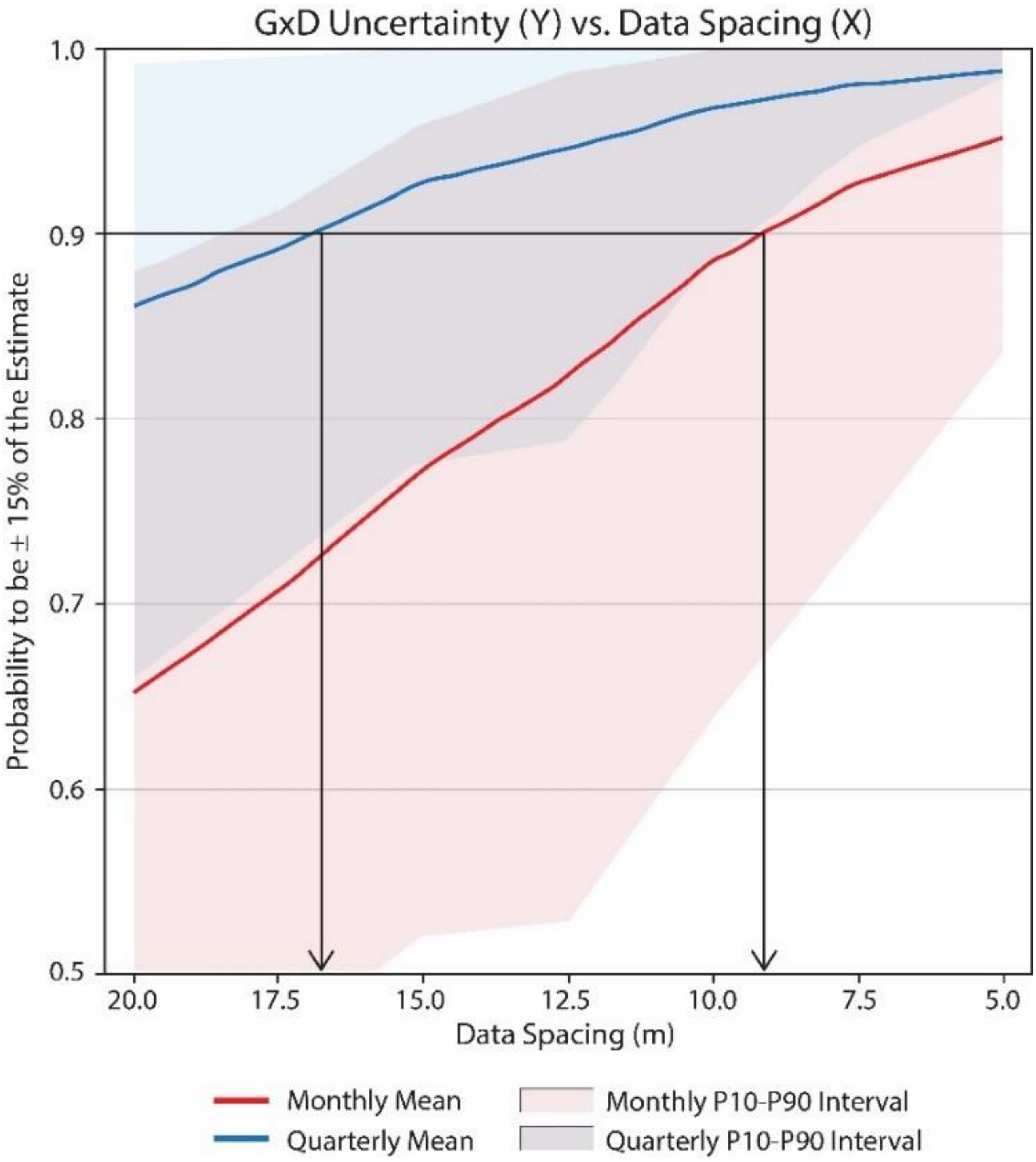
Definitions for resource categories used in this Technical Report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such 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 PFS or FS levels as appropriate. Mineral Reserves are classified into Proven and Probable categories.

Mineral Resources for the Arrow Deposit are classified into Measured, Indicated, and Inferred categories based on studies conducted for NexGen by Deutsch Consultants in 2017 (Deutsch, 2017) and Resource Modeling Solutions in 2018 (Deutsch and Barnett, 2018). The principal aim of this work was to establish a geostatistical simulation workflow for uncertainty as a function of drill hole spacing and measured resources at production scale to support decisions related to future drilling and classification.

It is common to express uncertainty as a probability of the produced metal to be within a specified tolerance. Three parameters are considered: (1) the time period for production – one month or one quarter, (2) the tolerance – 15%, and (3) the probability to be within the tolerance – > 90% for Measured and between 75% and 90% for Indicated. Figure 14-20 summarizes the drill spacing study documented in the 2018 report (Deutsch, et. al, 2018).

Figure 14-20: Grade Uncertainty versus Drill Hole Spacing Arrow Deposit



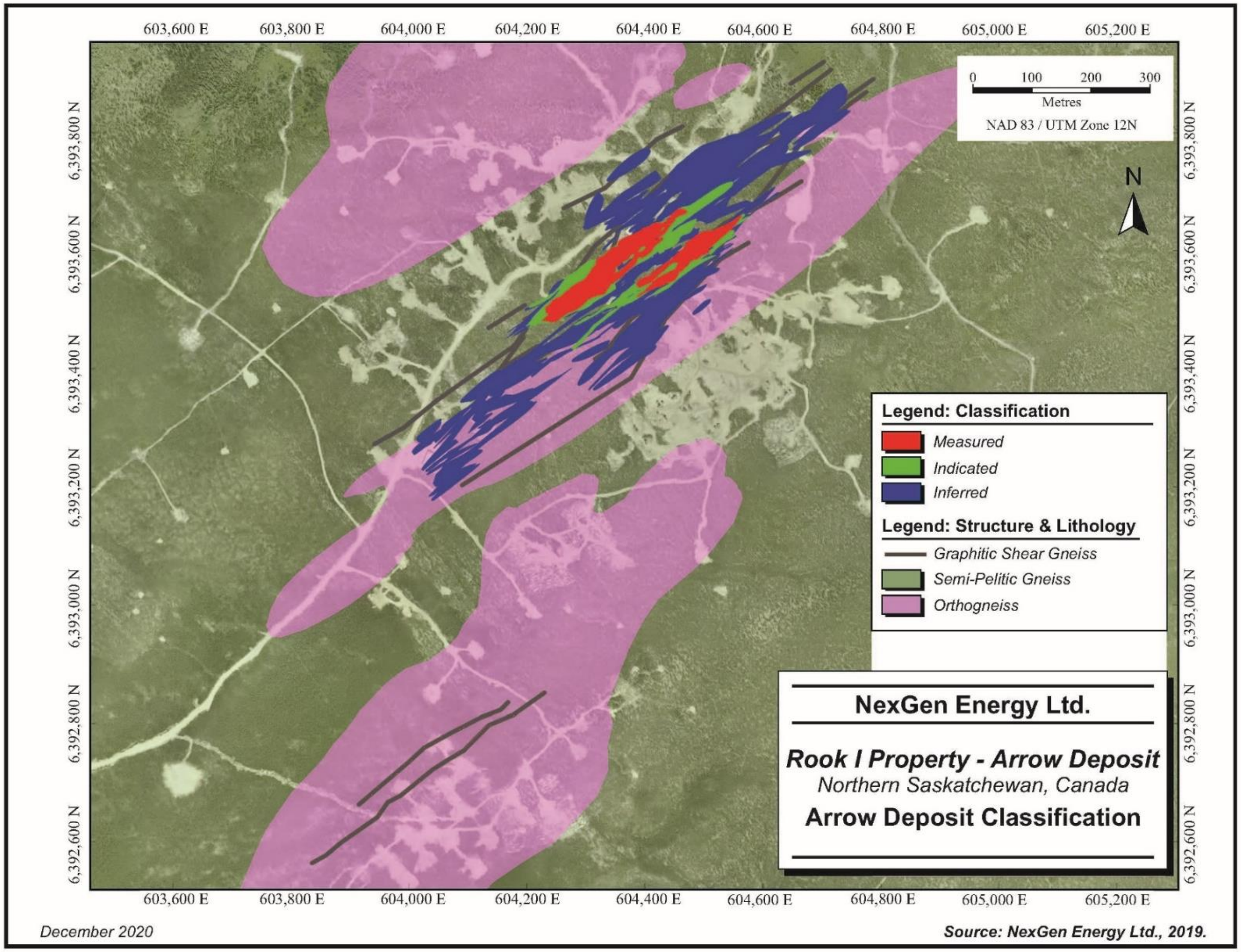
Based on the data spacing study, drill spacing ranging from 9.00 m to 16.75 m will have a 90% probability of being within 15% of the estimated mean at a monthly and quarterly production volume.

Measured, Indicated, and Inferred categories are based on the following parameters.

- **Measured Mineral Resources**
  - Defined by 9.00 m to 16.75 m in well defined areas as established by the 2018 drill hole study by Resource Modelling Solutions.
- **Indicated Mineral Resources**
  - Defined by 16.75 m to 32.0 m drill hole spacing, as established by the 2017 drill hole study by Deutsch Consultants.
- **Inferred Mineral Resources**
  - Defined by drill hole spacing that is greater than 25 m by 25 m and a NN distance of 32 m to 70 m with reasonable continuity assumed between holes
  - It is reasonably expected that the majority of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Figure 14-21 shows the Arrow Deposit classification.

Figure 14-21: Arrow Deposit Classification



14.10 Mineral Resource Reporting

Measured Mineral Resources at the Arrow Deposit total 2.18 Mt, at an average grade of 4.35% U_3O_8, for a total of 209.6 Mlb of U_3O_8. Indicated Mineral Resources total 1.57 Mt, at an average grade of 1.36% U_3O_8, for a total of 47.1 Mlb U_3O_8. Inferred Mineral Resources total 4.40 Mt, at an average grade of 0.83% U_3O_8, for a total of 80.7 Mlb U_3O_8 (see Table 14-1). The effective date of the Mineral Resource estimate is 19 July 2019. Estimated block model grades are based on chemical assays only. Mineral Resources were estimated by NexGen and audited by RPA. Mineral Resources are inclusive of Mineral Reserves. CIM (2014) definitions were used for Mineral Resource classification.

The Mineral Resource estimate is broken down by domain in Table 14-17.

Table 14-17: Mineral Resource Estimate– 19 July 2019

Classification	Zone	Domain	Tonnage (t)	Grade (%U_3O_8)	Contained Metal (lb U_3O_8)
Measured	A2-LG	206	876,000	0.80	15,400,000
		213	44,000	0.56	500,000
	A2-LG Total	–	920,000	0.79	16,000,000
	A2-HG	1	130,000	19.63	56,200,000
		2	81,000	17.67	31,600,000
		3	45,000	16.26	16,200,000
		5	133,000	16.35	48,100,000
		6	2,000	14.04	700,000
		7	22,000	7.77	3,800,000
		8	27,000	8.87	5,300,000
	A2-HG Total	–	441,000	16.65	161,900,000
	A3-LG	301	639,000	1.86	26,200,000
		312	183,000	1.36	5,500,000
	A3-LG Total	–	821,000	1.75	31,700,000
Measured Total	–	–	2,183,000	4.35	209,600,000
Indicated	A2-LG	205	52,000	0.55	600,000
		206	316,000	0.82	5,700,000
		208	15,000	0.62	200,000
		210	57,000	0.89	1,100,000
		211	30,000	1.22	800,000
		213	82,000	0.72	1,300,000
		214	6,000	0.50	100,000
		225	50,000	0.97	1,100,000

Classification	Zone	Domain	Tonnage (t)	Grade (%U_3O_8)	Contained Metal (lb U_3O_8)
		229	56,000	0.61	800,000
		242	30,000	0.66	400,000
		246	6,000	0.34	0
	A2-LG Total	–	700,000	0.79	12,200,000
	A2-HG	1	1,000	15.28	200,000
		3	3,000	13.42	1,000,000
		5	21,000	10.91	5,100,000
		6	10,000	7.31	1,600,000
		7	20,000	9.30	4,200,000
		8	1,000	12.47	300,000
	A2-HG Total	–	56,000	9.92	12,300,000
	A3-LG	301	414,000	1.42	12,900,000
		302	47,000	0.61	600,000
		307	9,000	0.62	100,000
		312	271,000	1.35	8,100,000
		321	23,000	0.57	300,000
		323	37,000	0.58	500,000
		337	14,000	0.42	100,000
	A3-LG Total	–	815,000	1.26	22,700,000
Indicated Total	–	–	1,572,000	1.36	47,100,000
Inferred	A1	100	196,000	0.47	2,000,000
		101	111,000	0.35	900,000
		102	58,000	0.57	700,000
		103	11,000	0.38	100,000
		104	105,000	0.63	1,500,000
		105	9,000	0.63	100,000
		106	16,000	0.34	100,000
		107	62,000	0.61	800,000
		108	119,000	0.98	2,600,000
		109	79,000	1.79	3,100,000
		110	29,000	0.35	200,000
		111	9,000	0.40	100,000
		113	7,000	0.51	100,000
		114	32,000	0.44	300,000
		115	11,000	0.41	100,000

Classification	Zone	Domain	Tonnage (t)	Grade (%U_3O_8)	Contained Metal (lb U_3O_8)
		116	33,000	0.41	300,000
		117	285,000	0.78	4,900,000
		118	208,000	0.69	3,200,000
		119	20,000	0.47	200,000
		120	1,000	0.71	0
		121	70,000	0.98	1,500,000
		122	13,000	0.34	100,000
		123	21,000	0.55	300,000
		124	8,000	0.32	100,000
		125	18,000	0.40	200,000
		900	26,000	0.62	300,000
	A1 Total	–	1,557,000	0.69	23,700,000
	A2-LG	202	12,000	0.72	200,000
		203	4,000	0.59	100,000
		204	3,000	0.40	0
		205	10,000	0.45	100,000
		206	62,000	0.63	900,000
		207	41,000	0.55	500,000
		209	6,000	0.72	100,000
		211	9,000	0.35	100,000
		212	78,000	0.60	1,000,000
		214	5,000	0.57	100,000
		215	1,000	0.89	0
		216	31,000	0.77	500,000
		217	30	0.26	160
		218	490	0.29	3,190
		220	13,000	0.49	100,000
		221	7,000	0.43	100,000
		222	1,000	0.37	0
		223	10,000	1.03	200,000
		224	15,000	0.57	200,000
		225	6,000	0.67	100,000
		227	2,000	0.99	0
		228	12,000	0.56	100,000
		229	17,000	0.36	100,000

Classification	Zone	Domain	Tonnage (t)	Grade (%U_3O_8)	Contained Metal (lb U_3O_8)
		230	340	0.33	2,470
		231	5	0.32	0
		232	5,000	0.35	0
		233	2,000	0.36	0
		234	1,000	0.31	0
		235	4,000	0.29	0
		236	94,000	0.48	1,000,000
		237	49,000	0.41	400,000
		238	2,000	0.28	0
		240	9,000	0.67	100,000
		241	33,000	0.80	600,000
		245	8,000	1.04	200,000
		248	0	0.00	0
		249	10,000	0.92	200,000
		250	9,000	0.58	100,000
		252	4,000	0.52	0
		253	2,000	0.43	0
		254	72,000	0.49	800,000
		255	12,000	0.50	100,000
		256	88,000	0.59	1,200,000
		257	90,000	0.80	1,600,000
		258	17,000	1.07	400,000
		259	5,000	0.51	100,000
		260	1,000	0.29	0
	A2-LG Total	–	863,000	0.61	11,500,000
	A2-HG	5	2,000	14.61	500,000
		6	1,000	4.95	100,000
	A2-HG Total	–	3,000	10.95	600,000
	A3-LG	301	47,000	0.71	700,000
		302	11,000	0.53	100,000
		303	10,000	1.26	300,000
		305	0	0.00	0
		306	8,000	0.40	100,000
		308	3,000	0.41	0
		309	110,000	2.32	5,600,000

Classification	Zone	Domain	Tonnage (t)	Grade (%U_3O_8)	Contained Metal (lb U_3O_8)
		310	30,000	0.83	500,000
		311	20,000	0.47	200,000
		312	22,000	1.34	600,000
		313	119,000	1.64	4,300,000
		314	2,000	0.30	0
		315	26,000	0.49	300,000
		316	10,000	0.30	100,000
		317	4,000	0.42	0
		318	245,000	1.44	7,800,000
		319	37,000	0.62	500,000
		320	5,000	0.48	100,000
		321	9,000	0.58	100,000
		322	70,000	0.83	1,300,000
		323	11,000	0.87	200,000
		324	18,000	0.66	300,000
		325	79,000	0.65	1,100,000
		326	137,000	0.84	2,500,000
		327	39,000	0.39	300,000
		328	41,000	1.32	1,200,000
		329	2,000	0.53	0
		330	6,000	0.90	100,000
		331	25,000	0.97	500,000
		332	0	0.00	0
		333	1,000	0.27	0
		334	9,000	0.42	100,000
		335	20,000	0.44	200,000
		336	4,000	1.04	100,000
		337	5,000	0.42	0
		338	0	0.00	0
		339	8,000	1.03	200,000
		340	0	0.00	0
		341	1,000	0.32	0
		342	8,000	0.41	100,000
		343	7,000	0.52	100,000
	A3-LG Total	–	1,207,000	1.12	29,800,000

Classification	Zone	Domain	Tonnage (t)	Grade (%U_3O_8)	Contained Metal (lb U_3O_8)
	A4	402	44,000	0.75	700,000
		403	180,000	1.29	5,100,000
		404	0	0.00	0
		405	49,000	0.45	500,000
		406	10,000	0.36	100,000
		407	64,000	0.56	800,000
		408	13,000	1.52	400,000
		409	12,000	1.04	300,000
		410	69,000	1.11	1,700,000
		411	27,000	0.74	400,000
		413	13,000	0.78	200,000
		414	17,000	1.52	600,000
		415	41,000	0.52	500,000
		416	12,000	0.64	200,000
		417	2,000	0.33	0
		418	64,000	0.99	1,400,000
		419	31,000	0.98	700,000
		420	2,000	0.52	0
		422	2,000	0.32	0
		424	34,000	0.48	400,000
		425	12,000	0.43	100,000
		426	16,000	0.50	200,000
		427	18,000	0.58	200,000
		428	3,000	0.44	0
		429	18,000	0.74	300,000
		430	7,000	0.60	100,000
		431	5,000	0.59	100,000
		432	4,000	0.44	0
		434	2,000	1.08	0
	A4 Total	–	769,000	0.89	15,000,000
Inferred Total	–	–	4,399,000	0.83	80,700,000

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported at a cutoff grade of 0.25% U_3O_8.
3. Mineral Resources are estimated using a long-term uranium price of US$50/lb U_3O_8 and estimated mining, operating, and processing costs.
4. A minimum thickness of one metre was used.
5. Tonnes are based on bulk density weighting.
6. Mineral Resources are inclusive of Mineral Reserves.

7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. Numbers may not sum due to rounding.

In RPA's opinion, the estimation methodology is consistent with standard industry practice and the Arrow Deposit Measured, Indicated, and Inferred Mineral Resource estimate is considered to be reasonable and acceptable.

15.0 MINERAL RESERVE ESTIMATE

The Mineral Reserve estimate was prepared by Stantec Consulting Ltd. (Stantec). The NexGen Arrow Deposit Mineral Reserve estimate is based on the M&I Resource material identified in the block model provided by NexGen (arw_4x4x4_id2_ok_2019Q3_rev3, issued on 02 October 2019). Resource material in the resource block model that was classified as Inferred Resources was assigned a grade of 0%.

The Mineral Reserves included in the Mineral Reserve estimate consist of selected portions of the Indicated and Measured Mineral Resources that are above a 0.30% U_3O_8 cutoff grade. This cutoff grade was applied at the level of stoping solids, after inclusion of waste and backfill dilution.

The Mineral Reserve is limited to the A2 and A3 veins in the Arrow Deposit. It is assumed that both transverse stope and longitudinal retreat stope mining methods will be used.

15.1 Mineral Reserves Statement

The FS defines probable Mineral Reserves of 239.6 Mlb of U_3O_8 contained in 4,575 kt grading 2.37% U_3O_8 from the M&I Mineral Resources This reserve estimate includes special waste (material between 0.03% and 0.26% U_3O_8 that must be extracted to access mining areas, which is uneconomic to process) and waste required for the purposes of ramping up the mill and grade control.

Estimates of mineralization and other technical information included herein have been prepared in accordance with NI 43-101 – Standards of Disclosure for Mineral Projects.

Table 15-1 presents the estimated Mineral Reserves, including the waste tonnes processed for mill ramp up and grade control. When calculating Mineral Reserves, Measured Resources were converted to probable reserves and not converted to proven reserves due the confidence in modifying factors. This confidence in modifying factors is a result of the Arrow Deposit being a new deposit with no mining operations history at the site but in no way impacts the geological confidence associated with Mineral Resources.

Table 15-1: Mineral Reserve Estimate

Classification	Recovered Ore Tonnes (thousands)	U_3O_8 Grade (%)	U_3O_8 lb (millions)
Proven	0	0	0
Probable	4,575	2.37%	239.6
Total	**4,575**	**2.37%**	**239.6**

Notes:
1. CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards) were followed for Mineral Reserves.
2. Mineral Reserves are reported with an effective date of 21 January 2021.

3. Mineral Reserves include transverse and longitudinal stopes, ore development, marginal ore, special waste, and a nominal amount of waste required for mill ramp-up and grade control.
4. Stopes were estimated at a cutoff grade of 0.30% U_3O_8.
5. Marginal ore is material between 0.26% U_3O_8 and 0.30% U_3O_8 that must be extracted to access mining areas.
6. Special waste in material between 0.03% and 0.26% U_3O_8 that must be extracted to access mining areas. 0.03% U_3O_8 is the limit for what is considered benign waste and material that must be treated and stockpiled in an engineered facility.
7. Mineral Reserves are estimated using a long-term metal price of US$50 per pound U_3O_8, and a 0.75 US$/C$ exchange rate (C$1.00 = US$0.75). The cost to ship the YC product to a refinery is considered to be included in the metal price.
8. A minimum mining width of 3.0 m was applied for all longhole stopes.
9. Mineral Reserves are estimated using a combined UG mining recovery of 95.5% and total dilution (planned and unplanned) of 33.8%.
10. The density varies according to the U_3O_8 grade in the block model. Waste density is 2.464 t/m^3.
11. Numbers may not add due to rounding.

15.2 Factors that May Affect the Mineral Reserves

Factors that may affect the Mineral Reserve estimate include the following.

- Commodity price assumptions.
- Changes in local interpretations of mineralization geometry and continuity of mineralization zones.
- Changes to geotechnical, hydrogeological, and metallurgical recovery assumptions.
- Input factors used to assess stope dilution or recoveries.
- Assumptions the operation can obtain all required permits to operate.
- Assumptions as to social, permitting, and environmental conditions.
- Additional infill or step out drilling.

15.3 Underground Assumptions / Design Criteria

15.3.1 Throughput Rate and Supporting Assumptions

The assumed process plant capacity is 1,300 t/d. Additional information is provided in Section 16.0.

15.3.2 Stope Shape Design

The mine design is based on using the sublevel longhole stoping mining method to extract the reserves. The mine design utilizes a mixture of longitudinal and transverse stope orientation.

Mine stope shapes were created using the Deswik Stope Optimizer (DSO). The DSO parameters used to create the stope shapes are presented in Table 15-2.

Table 15-2: Deswik Stope Optimizer Parameters

Parameter	Value
Orientation of DSO	-33°
Stope Width (Transverse) along Strike	12 m
Stope Length (Longitudinal) along Strike	12 m

Parameter	Value
Stope Height	30 m
Minimum Stope Width Horizontal	3 m
Minimum Stope Dip Angle	50°

15.4 Modifying Factors

Modifying factors applied in preparing the Mineral Reserve estimate includes planned dilution, unplanned dilution (such as external overbreak dilution, paste fill dilution), and mining recovery. The following sections describe the modifying factors and the application of the factors to the mine design.

15.4.1 Planned Dilution

Mining methods such as longhole stoping typically capture material below the cut off grade within each stope. Planned dilution is classified as material below the 0.3% U_3O_8 cutoff grade that is contained within the stope shapes and mined along with material above the cutoff grade. This planned dilution was calculated for all production stopes at 23.5% and is factored into the Mineral Reserve estimate.

15.4.2 Unplanned Dilution

External Overbreak Dilution

External overbreak dilution is material that is outside the stope shape but will be expected to overbreak into the stope and be recovered with the ore. Geotechnical domain and average horizontal stope width were used to determine which external dilution factor to apply. DSO utilized the external dilution factor to ensure the stope was above the input cutoff grade with the external dilution included.

Table 15-3 provides a summary of the external overbreak dilution factors used for the two rock domains. These domains are defined as the moderately altered Basement geotechnical domains (ABMT-2) and the slightly altered Basement geotechnical domains (ABMT-1).

Table 15-3: External Overbreak Dilution Factors

Domain	Average Horizontal Stope Width (m)	Footwall (FW) Dilution (m)	Hanging Wall (HW) Dilution (m)
ABMT-1	<10	0.50	0.50
ABMT-1	>10	0.25	0.25
ABMT-2	<10	0.75	0.75
ABMT-2	>10	0.50	0.50

North Rock Mining Solutions Inc. (NRMS) provided a 3D representation of each domain. The ABMT-2 domain is located within the ABMT-1 domain. The 3D representation of the ABMT-2 domain was used to classify the domain of each stope. If either the boundary midpoint or center of gravity of the stope fell within the ABMT-2 domain, the stope was classified as ABMT-2. Stopes not classified as ABMT-2 were classified as ABMT-1.

Any metal contained in the geological block model associated with the overbreak dilution will be included within the Mineral Reserve estimate. Resource material in the resource block model that was classified as Inferred Resources was assigned a grade of 0%.

Paste Fill Wall and Floor Dilution

Transverse secondary stopes will include some overbreak which will result in sidewall dilution, as these stopes will be mined adjacent to paste fill walls from the primary stopes. Additional fill dilution will be derived from the LHD mucking cycle, which will unintentionally recover some paste fill from the floor. An estimated 4% dilution from paste fill was included in secondary transverse stopes.

When secondary longitudinal stopes will be mined adjacent to a paste fill end wall from the previous stope, overbreak of the end wall will results in some paste fill being recovered. Additional fill dilution will be derived from the LHD mucking cycle, which will unintentionally recover some paste fill from the floor. An estimated 1% dilution from the paste fill was included in secondary longitudinal stopes.

Zero grade was assigned to the paste fill dilution.

The total estimated unplanned dilution is 13.4%, the total dilution (planned and unplanned) is estimated at 33.8%.

15.4.3 Mining Recovery

Mining losses account for Mineral Resources that will be mined but will not be recovered due to losses that occur through the mining process.

Mining losses in the ore development drifts are assumed to be zero as any unrecovered development ore will be extracted and included as part of the longhole stope.

Several factors influence mining losses from longhole stoping such as mucking line of sight, depth of sight, possible hang ups on the FW, drill hole layout, and blast complications.

It is expected that some ore that is blasted will not be recovered. Line of sight and maneuverability will prevent the LHDs from accessing muck in the front corners of the stope. It is assumed the maximum angle at which the LHD will be able to operate from the draw point will be approximately 45°. Cleanup at the back of the stope will be difficult to gauge and may result in additional lost ore recovery. Some of the unblasted ore in the side walls may be recoverable with the adjacent stope.

A mining recovery factor was applied to each variation of the longhole mining method. The overall average for mine stope recovery is 95.5%.

Table 15-4 provides a summary of the recovery factors used in the design.

Table 15-4: Recovery Factors

Mining Method	Recovery Factor (%)
Development Ore	100.0
Longitudinal	95.0
Transverse	95.5
Sill Recovery (530 and 650 Levels)	93.0
Overall Average	**95.5**

15.5 Cutoff Criteria

To determine a Mineral Reserve for the Arrow Deposit (Rook I Project FS), an estimated preliminary cutoff grade of 0.30% U_3O_8 was selected. The parameters used to estimate the cutoff grade are the same as the parameters used in the PFS prepared by Wood and RPA, taking into account any updated information estimated during the first phase of the FS.

Based on the Mineral Resource and cutoff grade calculation, stope shapes greater than 0.30% U_3O_8 were considered in the mine plan. The stope shapes created in the mining software Deswik were then visually evaluated against access costs and proximity to other stopes to determine if the stopes would be included in the mine plan.

The input parameters for the cutoff grade calculation are listed in Table 15-5.

Table 15-5: Cutoff Grade Calculation

Item	Value	Unit
Mine Operating Cost	$91.20	$/t
Mill Operating Cost (Processing)	$181.50	$/t
Tailing Storage Operating Cost (UGTMF)	$97.20	$/t
General and Administration Operating Costs	$67.11	$/t
Sustaining Capital Cost	$62.65	$/t
Total Operating and Sustaining Capital Cost	**$499.66**	**$/t**
Mine Operating Cost—Variable OPEX	$54.72	$/t
Mill Operating Cost (Processing)	$181.50	$/t

Item	Value	Unit
Tailing Storage Operating Cost (UGTMF)	$97.20	$/t
General and Administration Operating Costs	$67.11	$/t
Total Operating Cost—Incremental (Variable OPEX, No Sustaining Capital Cost)	**$400.53**	**$/t**
Uranium Price	$66.67	$/lb
Transportation Cost	$0.34	$/lb
Royalties	7.25%	%
Mill Recovery	97.6%	%
Uranium Revenue ($/lb U_3O_8 mined)	$60.05	$/lb U_3O_8
Uranium Revenue ($/t U_3O_8 mined)	**$132,378**	$/t U_3O_8
Cutoff Grade with Sustaining Capital	**0.38%**	%
Cutoff Grade without Sustaining Capital	**0.33%**	%
Incremental Cutoff (Var. Mining, G&A, Proc. and UGTMF)	**0.30%**	%
Marginal Cutoff (No Mine Operating Cost)	**0.26%**	%

Note : Cost assumptions based on the PFS completed in 2018 and updates prepared in Phase 1 of the FS in 2020. Costs are consistent with the January 2021 FS.

The incremental cutoff grade of 0.30% U_3O_8 was used for the initial stope optimizer inputs, which drove the mine design.

A nominal amount of material between cutoff grades of 0.03% U_3O_8 (the regulatory limit between benign waste and mineralized material) and 0.26% U_3O_8 (which is uneconomic to process) have been included in the mine plan.

15.6 Comments on Section 15

Mineral Reserves are reported herein according to the 2014 CIM Definition Standards.

The QP has reviewed the risks, opportunities, conclusions, and recommendation and is not aware of any conditions that would put the Mineral Reserve at a higher risk level than any other North American developing project.

16.0 MINING METHODS

16.1 Overview

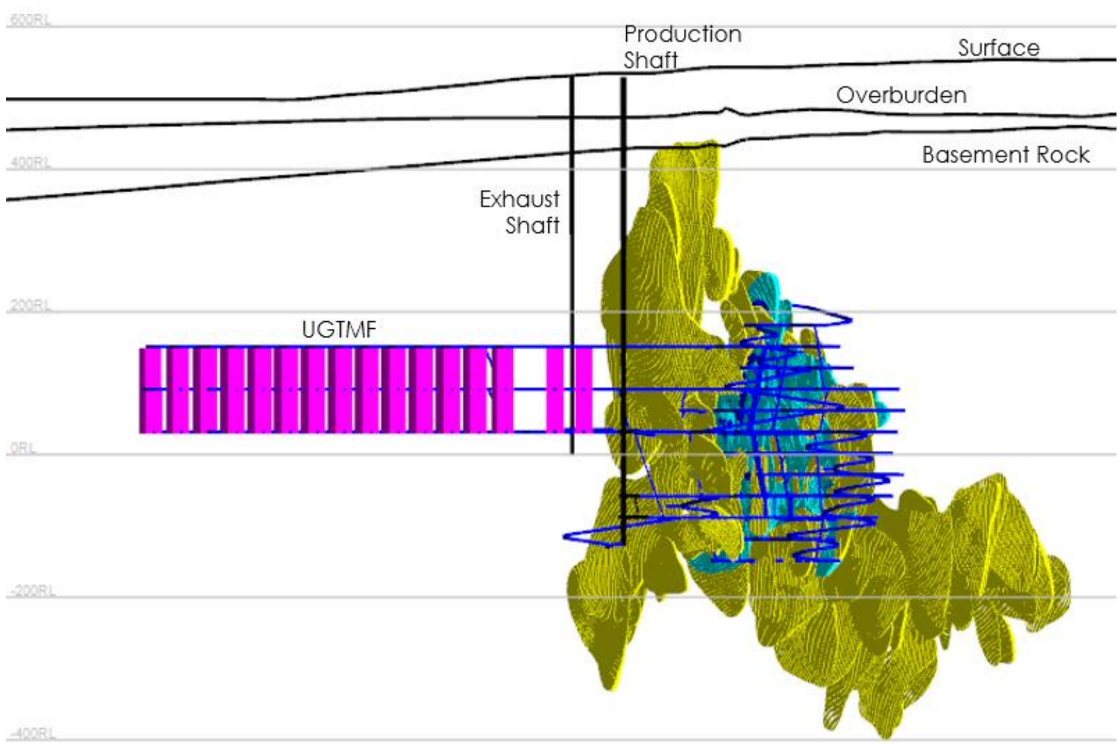
Access to the UG Arrow Deposit will be via two shafts: an 8.0 m diameter Production Shaft for intake air, and a 5.5 m diameter Exhaust Shaft for return ventilation and second egress. Access to the mine will be via the Production Shaft, with mine access shaft stations on the 500 Level and 590 Level, and a loading pocket shaft station on the 620 Level. Levels will be spaced 30 m apart UG and will be connected via an internal ramp.

Production will use a conventional longhole mining method. The longhole mining method and mine design presented in this section were selected to achieve the following objectives.

- Optimize safety performance.
- Reduce worker exposure to physical hazards and radiation.
- Maximize Mineral Resource extraction.
- Increase operational flexibility and productivity by achieving concurrent production from multiple mining fronts.

Figure 16-1 presents a longitudinal projection of the UG mine.

Figure 16-1: Section of Mine Looking North



The targeted mill capacity is 1,300 t/d of ore. To realize this target, the mine plan will include longhole production from four separate mining blocks, with multiple stopes available per block. The estimated production rate of the stopes ranges from 250 t/d to 300 t/d, and will require approximately five stopes to be active to achieve 1,300 t/d. Additionally, the mill will be capped at a head grade of 5.0% U_3O_8. The grade varies within each stope, allowing for better ability to control the head grade from the mine.

The Mineral Resource for M&I material extends from approximately 290 m below surface to 720 m below surface. The resource dips steeply, averaging between 80° to 85° from horizontal.

The M&I resources are contained within two shear zones, the A2 and A3. The A2 shear zone is located FW to the A3 with a natural pillar between the two shear zones. A summary of the M&I Resources by shear zone is included in Table 16-1.

Table 16-1: M&I Resource Summary

Zone	Tonnes (millions)	U_3O_8 Grade (%)
A2	2.12	4.33
A3	1.64	1.51
Total	**3.75**	**3.10**

The tailings produced by the mill will be returned UG as either paste backfill for the production stopes, or as cemented paste tailings into waste stopes that will be excavated for this purpose. The UGTMF will be located on the north side of the deposit and will consist of approximately 100 UGTMF stopes and related development.

The mining method will use mechanized equipment and conventional processes widely employed in the global mining industry.

Design criteria and parameters specific to each aspect of the mining method and mine design are presented in subsequent sections. The following were considered when determining the criteria and parameters during the mine design process.

- Ensure minimized and effective management of radiation exposure to operating personnel.
- Develop and design the mine to achieve regulatory approvals.
- Achieve buy-in from local stakeholders.
- Minimize the mine environmental footprint.
- Health and safety for the workers, local communities, and the environment.
- Company standards and specifications (or industry best practices where company standards and specifications were not available).
- Prevention through design concepts.
- UG tailings storage.
- Minimizing risk to production.
- Use of proven industry technology, equipment, and processes.

- Use of automation to reduce worker exposure.
- Operational flexibility.
- Minimizing operating costs.
- Mineral Resource recovery (extraction rate).

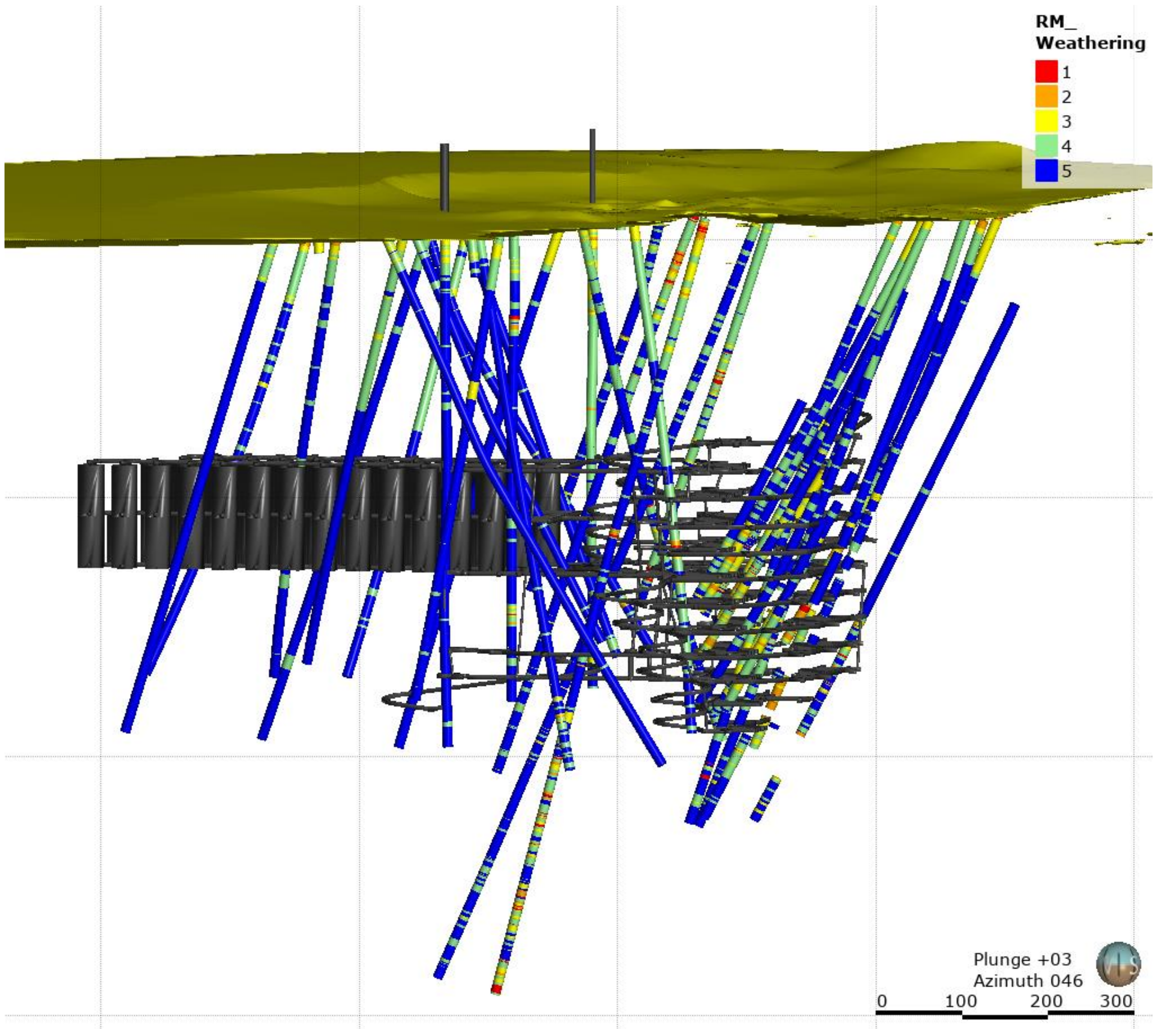
16.2 Geotechnical Assessment

An UG mining geotechnical assessment of basement rock was completed by North Rock Mining Solutions Inc. (NRMS).

16.2.1 Mining Geotechnical Conditions

An understanding of basement rock mass conditions is required to reliably predict rock mass responses due to mining at the Arrow Deposit. The property has been the subject of substantial geotechnical investigation starting in 2014. Since 2016, NexGen has been collecting four rock mass classification parameters (i.e., intact rock strength, rock quality designation, joint spacing, and joint condition data) per drill interval for every drill hole, in addition to 43 specifically-targeted bedrock geotechnical drill holes, as presented in Figure 16-2.

Figure 16-2: Detailed Geotechnical Site Investigation Holes (2014–2019) – by Rock Mass Weathering / Alteration Drill Hole Data



Rock mass weathering / alterations have been rated per Table 16-2.

Table 16-2: Rock Mass Weathering / Alterations

Rock Mass Weathering		Alteration Style		
Grade	Value	Grade	Value	Style Description
None	5	None	0	No visible sign of rock material or weathering.
Slight	4	Trace	1	Less than 1% of the rock mass has been altered.
Moderate	3	Weak	2	1%–5% of the rock mass has been altered.
High	2	Moderate	3	5%–15% of the rock mass has been altered.
Decomposed	1	Strong	4	15%–30% of the rock mass has been altered.
Core Loss	0	Intense	5	More than 30% of the rock mass has been altered.

This geotechnical data set is valuable for its broad spatial distribution and for identification of rock classification boundaries or geotechnical domain boundaries, with domains further verified and characterized using data from the targeted geotechnical holes.

The Arrow Deposit is exclusively hosted in crystalline basement lithologies below an unconformity that is overlain by sedimentary units, glacial till, and overburden. In descending order, the overlying units are overburden, lower glacial till unit, the Cretaceous Manneville Group, the Devonian La Loche Formation, and the Athabasca Sandstone.

Directly below the unconformity is variably weathered basement rock, where the weathering depth and profile varies and penetrates deeper into the basement along conduits for water (i.e., shears, faults, and other persistent geologic structures).

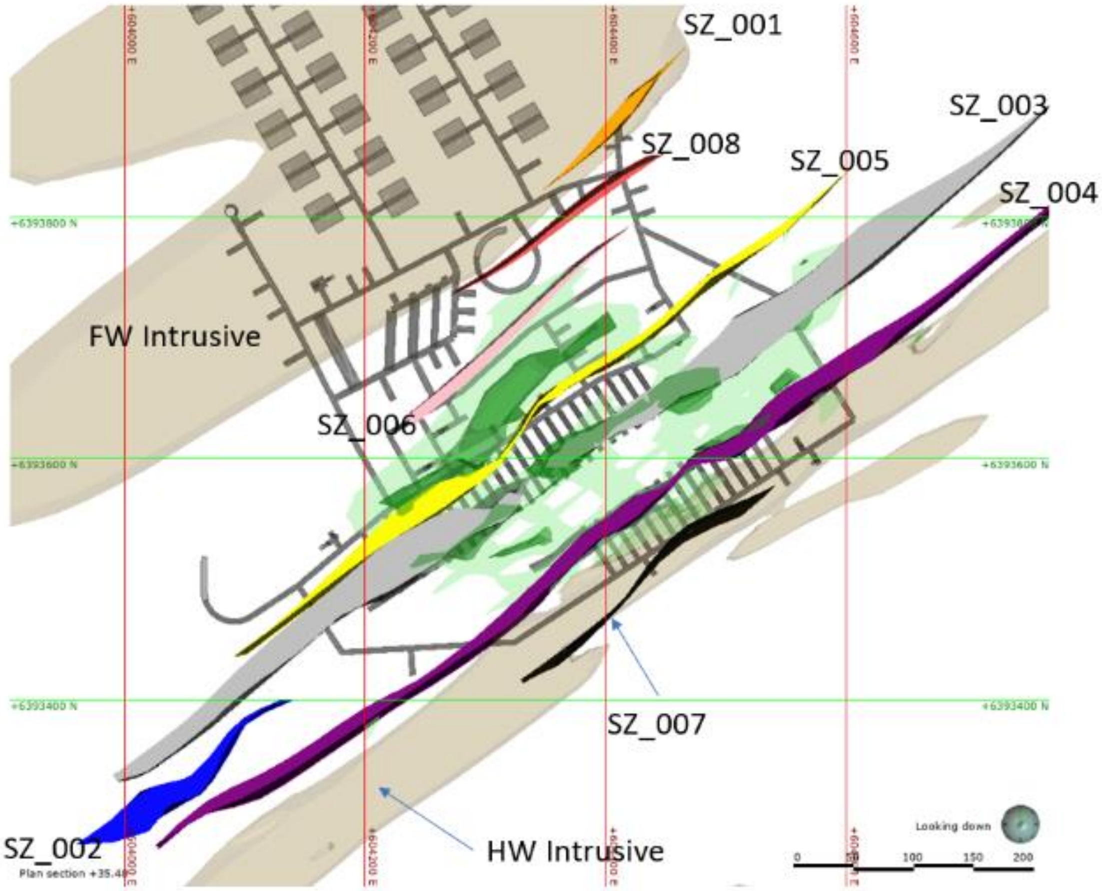
The primary basement lithological units include the following.

- Semi-pelitic gneiss (SPGN) / quartz-feldspar-garnet-biotite (principal host / country rock).
- Granitic intrusive bodies (intrusive) located in the FW and the HW and southwest of the Arrow Deposit.
- Quartz veins (and breccias) concordant and approximately 45° to mineralization.

Several interpreted basement shears and faults are concordant and acute to mineralization. Shear zones are closely related to controls on rock mass quality. There are eight primary shear zones between the HW and FW intrusives that are approximately concordant with mineralization. There are five interpreted tertiary shear zones that are approximately 45° to the primary shears.

Shears and geotechnical domain models are presented in Figure 16-3.

Figure 16-3: Shears and Geotechnical Domain Models



1. Light Green = slightly altered basement domain (ABMT-1).
2. Dark Green = moderately altered basement domain (ABMT-2).
3. Shear Zones 4 to 6 will have the most impact on ground stability and the mine plan.
4. 500 Level shown is from previous mine design.

Lab-measured UCS at the Arrow Deposit range from 10 MPa to nearly 250 MPa. Approximately 100 tests have been completed in basement rock. Resultant strength distribution is presented in Table 16-3.

Table 16-3: Basement UCS Statistics by Rock Mass Weathering

Rock Mass Weathering	Count	Average	Std Dev	Min	Max
5	65	113.7	50.7	14.4	235.6
4	31	101.0	60.7	10.3	238.3
3	4	42.0	18.7	24.9	64.9
Total	**100**	**106.9**	**54.8**	**10.3**	**238.3**

There is a relatively even distribution representing the full spectrum of strengths from nearly negligible (i.e., intense weathering / alteration) to over 200 MPa. Rock strength is similar between the primary basement lithologies, the SPGN and the Intrusive Package (INTR), when unaltered.

Rock mass classification systems have been used to classify rock mass quality in basement rock. This data has been used to assess the range in anticipated rock quality in the primary areas of interest associated with the FS (i.e., mining, shaft, and UGTMF zones). In addition to rock quality, other geoparameters were spatially interpreted to develop geotechnical domain models to relate the geotechnical conditions to the mine plan developed by Stantec.

The basement geotechnical domains are presented in Table 16-4.

Table 16-4: Basement Geotechnical Domains

Domain	Sub-domain	Description	Intact Rock Strength (MPa)	RMR Range	Quality (Q') Design Range (Avg.)
Weathered Basement Domain (WBMT)	Primary Lithology	Paleoweathered basement below unconformity, extending deeper in the SPGN.	<50	40 to 60	1–4
Basement Domain (BMT)	Primary Lithology	Unaltered basement rock, including SPGN and intrusives. Good to very good rock quality.	100 to 250+	60+	10+
Altered Basement Domain (ABMT)	ABMT-1	Slightly altered basement. Fair rock quality. The outer halo of alteration. Primarily in SPGN. Encompasses all stoping areas.	50 to 100	50 to 60	4–10 (6)
	ABMT-2	Moderately-altered to strongly-altered BMT. Predominantly fair to poor rock quality. The inner halo of alteration. Primarily SPGN.	25 to 50	40 to 50	1–4 (3)

Using widely accepted techniques combining laboratory test data with Geological Strength Index (GSI) estimates of rock quality (correlated to RMR-89 ratings), results in Hoek-Brown rock mass strength criteria and related failure envelopes (useful for mine design) for the various geotechnical domains are presented in Table 16-5.

Table 16-5: Hoek-Brown Strength Criteria in Basement Rock

Unit	UCS	GSI	mi	Erm
WBMT	<50	40 to 60	12-15	5-20
BMT	>100	>60	15-25	10-30
ABMT-1	50 to 100	>50	10-15	5-20

Unit	UCS	GSI	mi	Erm
ABMT-2	<50	40 to 60	8-12	5-20

16.2.2 Mining Geotechnical Design

Uranium mining began in Saskatchewan in the 1950s. The Arrow Deposit is located within the Athabasca Basin, which is recognized as a well-explored and well-developed uranium mining region. Many brownfield sites with analogous geotechnical conditions and histories of UG extraction use similar methods and have accessible public and academic information.

The Arrow Deposit offers the advantage of steeply dipping stopes in basement rock that is not as significantly degraded / altered as other HG, unconformity-hosted uranium deposits (i.e., Cigar Lake, McArthur River, Roughrider Upper West, Midwest, and Phoenix).

The ground conditions are similar to Eagle Point Mine, part of Cameco's Rabbit Lake Operation, a basement-hosted deposit mined using the same methods proposed at the Arrow Deposit. Mining at the Rabbit Lake Operation commenced in the 1970s.

The rock classification at the Eagle Point Mine is analogous to the Arrow Deposit, with Q-values ranging from approximately 0.5–10 and stopes moderately dipping at 45°–60°. Stopes were successfully mined both with and without cable bolt support, depending largely on local conditions and rock mass quality variability within the stope HW (Capes, 2009). The standard stope size at the Eagle Point Mine is approximately 15 m wide by 15 m long over a sublevel height of 30 m (Capes, 2009), similar to the stope dimensions planned at the Arrow Deposit.

Similar to the PFS, the planned mining method for the Arrow Deposit is longitudinal and transverse longhole stoping with CPB. The stope sequence is a bottom-up and inward-out pyramid sequence, with an upward advance of two mining areas in the A2 and A3 veins.

Four primary areas of mine production are termed "mining blocks": lower A2 block, upper A2 block, lower A3 block, and upper A3 block, divided on the 500 Level.

The mining blocks and mining fronts start on the 620 Level in the lower A2 and A3 blocks, respectively, and on the 500 Level for the upper A2 and A3 blocks, with undercut levels on the 620 and 500 Levels. There will be a small mining front below the 620 Level down to the 680 Level, but will only be used to augment tonnes from the lower blocks as required. This will require engineered sill pillars in the A2 and A3 veins to maximize recovery, just below the 500 Level and 620 Level.

Transverse stopes are 12 m wide by 12 m long. Longitudinal stopes are typically 7 m wide, and up to 24 m long. The typical sublevel height is 30 m. The stopes primarily dip discordant to foliation to the south-southeast. The A2 stope HW dips between 75°–90°. The A3 shear is steeper, dipping greater than 80°.

As part of the mine and stope design, and dimensioning geotechnical evaluation, the following tasks have been completed by NRMS.

- Mining geotechnical site inspections.
- Analysis of the structural data set to define dominant discontinuity orientations for kinematic analysis (wedge analysis) and stope stability assessments.
- Empirical stability assessments of all excavations using the widely accepted stability graph method (Hutchinson and Diederichs, 1996, and Nickson et al., 1992, after Potvin, 1988).
- Development of ground support recommendations in development and production excavations.
- Determination of stable transverse and longitudinal longhole stope dimensions.

Stope dilution estimates were prepared using the empirical Equivalent Linear Overbreak Slough (ELOS) method by Clark (1998). Stope dimensions (HRs) were plotted on the empirical ELOS chart with minimum N-values to define depths of overbreak ranges.

Stope dilution by domain using ELOS method is presented in Table 16-6.

Table 16-6: Stope Dilution by Domain using ELOS Method (Clark and Pakalnis, 1997)

Domain	Mine Method	ELOS Estimate (m)	Comments
ABMT-1	Longitudinal	0.5	24 m stope lengths stable without support.
	Transverse	0.25	12 m stope lengths stable without support.
ABMT-2	Longitudinal	0.75+	24 m stope lengths unstable except at the upper Q' limit in ABMT-2. Stable walls up to 12–15 m in strike.
	Transverse	0.5+	12 m stope lengths stable without support.

To inform the empirical stability analyses and design process, preliminary 2D and 3D stress models for the conceptual PFS and FS mine designs were developed.

The stress models were used to further investigate and verify empirical designs and stoping concepts by determining probable stress conditions and magnitudes in the backs, walls, pillars, and abutments of excavations at various stages and elevations within the proposed mining zone and by determining locations of stress shadowing, relaxation, concentrations, and zones of potential over-stressing.

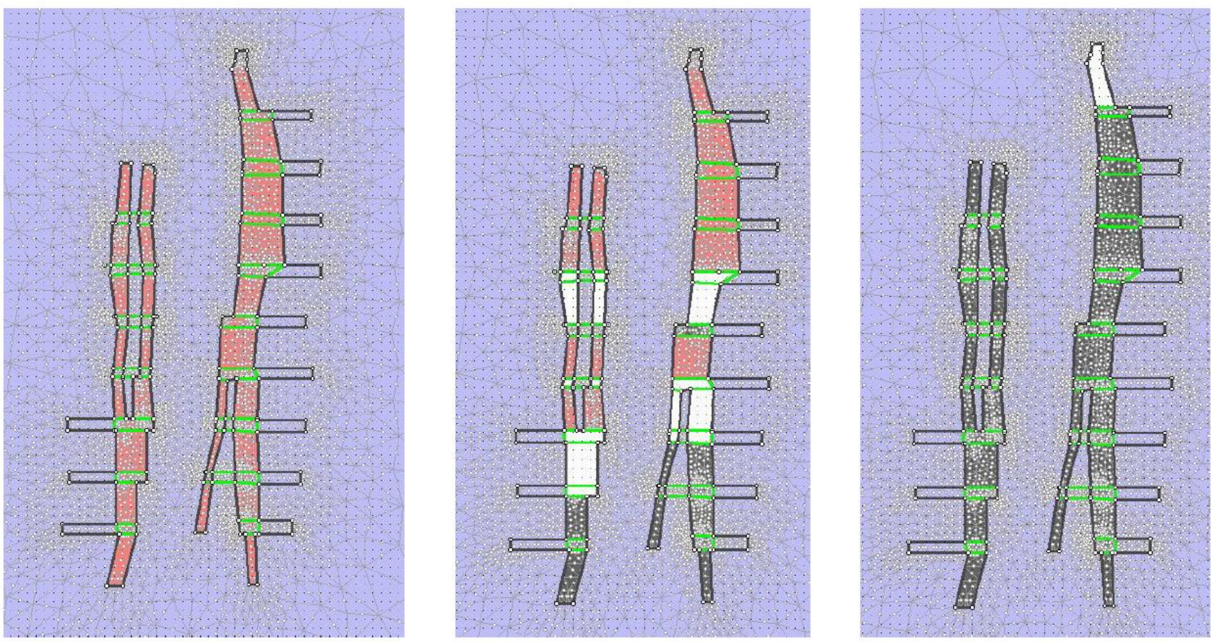
Modelling results were reviewed by NRMS to identify and assess the following.

- Areas where potential rock damage may be concentrated.
- Areas where stress shadows (i.e., loss of confinement or relaxation) are possible.

- Areas where standoff distances and/or pillar thicknesses are important to isolate excavations from significant mining-induced stress changes.

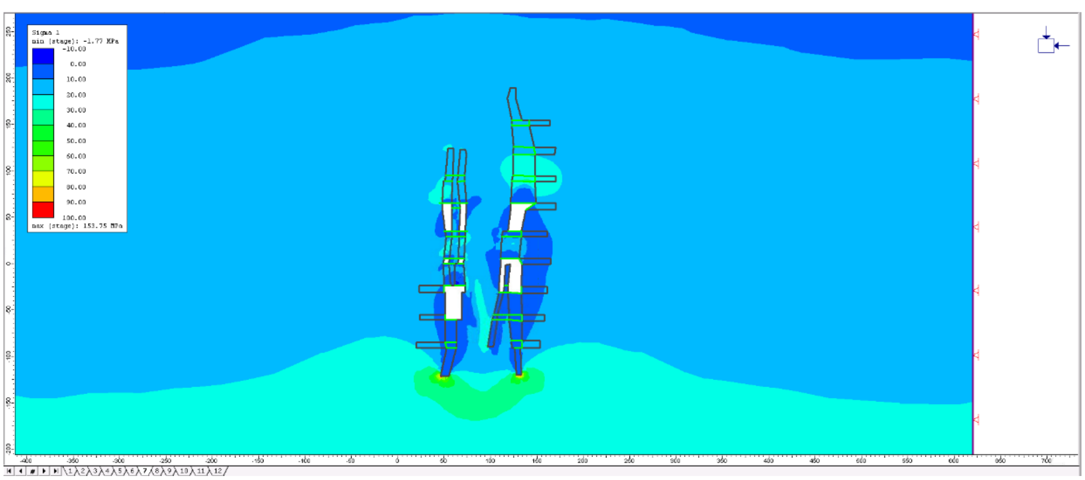
The typical 2D stress assessment indicating model geometry and sequence is presented in Figure 16-4.

Figure 16-4: Typical 2D Stress Assessment - Model Geometry and Sequence



The typical 2D stress assessment indicating major principal stress (σ_1) is presented in Figure 16-5.

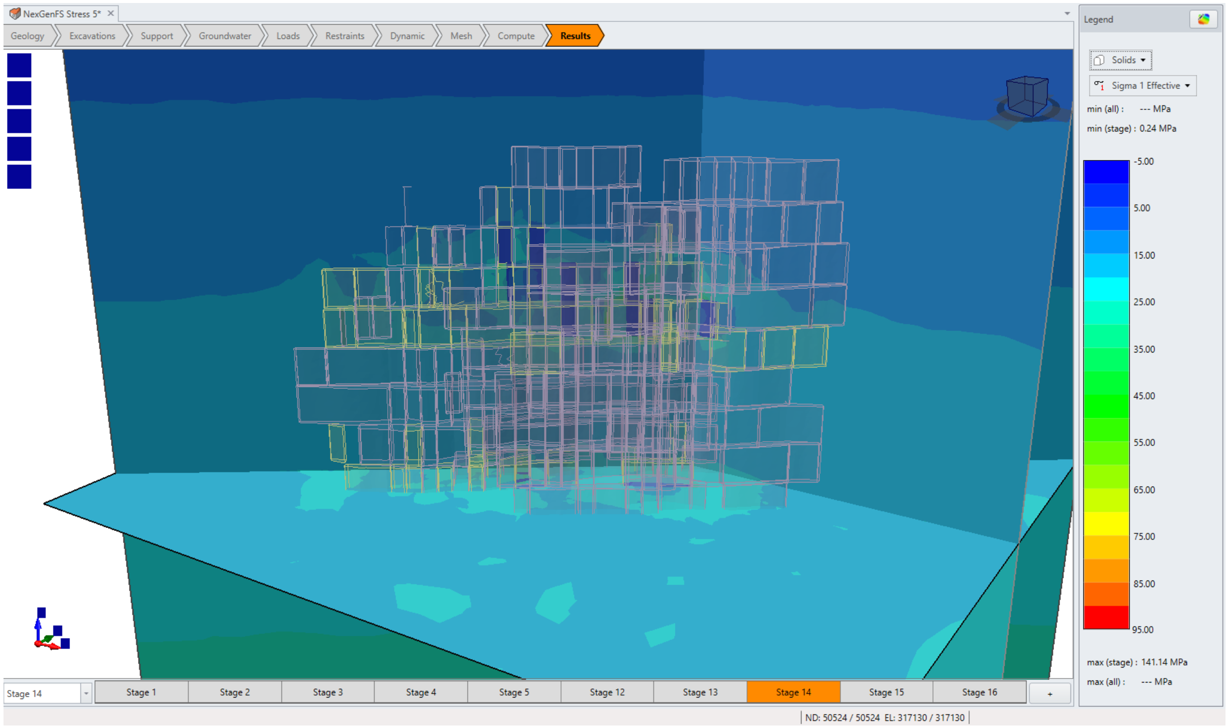
Figure 16-5: Typical 2D Stress Assessment –σ_1 Stress



Note: Models include ABMT-1 / 2 domains, stress redistributions, and general de-stressed conditions for 'in-production' stope HWs and FWs.

The typical 3D stress assessment is presented in Figure 16-6.

Figure 16-6: Typical 3D Stress Assessment – σ_1 Stress



Note: Modelled $\sigma_1$stress (MPa) in active stoping zones during typical mid-stage mine development.

Pillars in the mine include sill pillars, rib pillars, shaft pillars, and crown pillars beneath the unconformity. The pillars will be under a range of loading conditions in differing ground conditions and were assessed using primarily 3D stress models to evaluate stress / strength relationships and sensitivity to intact and rock mass inputs.

Standard empirical pillar design methods (Lunder, Hoek) were reviewed for this study as a pseudo-validation of the numerical stress assessment.

A high strength paste sill pillar will be constructed to increase the number of available production faces. Based on experience in similar geotechnical conditions, and results from the preliminary stress assessments, a high-strength paste will be adequate. Detailed design of the paste sill pillar will be required at the detailed engineering stage, prior to UG placement.

16.2.3 Ground Support Design

Ground support designs for planned excavations were assessed and analyzed using the widely accepted empirical design methodology after Grimstad and Barton (1993, 2014).

All ore development requires shotcrete to provide a gamma radiation barrier to protect UG personnel. The minimum shotcrete thickness is 50 mm reinforced with fibre. Alternatively, shotcrete without fibre may be used in conjunction with welded wire mesh. The minimum compressive strength of the shotcrete is 35 MPa at 28 days.

It is probable the shotcrete thickness required for gamma blocking (i.e., 50 mm) will be sufficient for and enhance support for most rock mass conditions. This will assist to support the stope backs and drift walls that are not directly addressed in the empirical stope stability assessments. The recommendations for shotcrete thickness are the minimum required for support from a geotechnical perspective. Additional shotcrete may be required to protect personnel from gamma radiation.

The summary for BMT ground support is presented in Table 16-7.

Table 16-7: BMT Ground Support Summary

Opening Type	Cross Section (w × h [m])	Ground Support Element	Bolt Length (m)	Bolt Spacing (m)	Notes
Lateral Development	5 × 5.5	#7 Resin rebar	1.8	1.5 × 1.5	100% coverage of back and shoulders. Bolt and screen walls down to 2 m above sill.
Wider Spans	Spans >6 m wide	#7 Resin rebar	2.4	1.5 × 1.5	100% coverage of back and shoulders. Bolt and screen walls down to 2 m above sill.
		Coupled fully-grouted #7 Resin rebar OR single-strand 17.8 mm (0.7 inch) diameter cable bolts	Half the span	2.5 × 2.5	Long secondary support approximately half the span. Alternatives include expandable rock bolts or connectable expandable rock bolts.
Personnel-Entry Raises (Alimak)	3 × 3	#7 Resin rebar	1.8	1.2 × 1.2	Screen 100% coverage. Brows supported with wider span support (i.e., longer support).
Raisebored Raises	3.5	–	–	–	Support not required in BMT Domain.

The summary for ABMT-1 ground support is presented in Table 16-8.

Table 16-8: ABMT-1 Ground Support Summary

Opening Type	Cross Section (w x h [m])	Ground Support Element	Bolt Length (m)	Bolt Spacing (m)	Notes
Lateral Development	5 × 5.5	#7 Resin rebar	1.8	1.2 × 1.2	100% coverage of back and shoulders. Bolt and screen walls down to 1 m above sill.
Wider Spans	Spans >6 m wide	#7 Resin rebar	2.4	1.2 × 1.2	100% coverage of back and shoulders. Bolt and screen walls down to 1 m above sill.
		Coupled fully-grouted #7 Resin rebar OR single-strand 17.8mm (0.7 inch) diameter cable bolts	Half the span	2.5 × 2.5	Long secondary support approximately half the span. Alternatives include expandable rock bolts or connectable expandable rock bolts.
Personnel-Entry Raises	4 × 4	#7 Resin rebar	1.8	1.0 × 1.0	Screen 100% coverage. Brows supported with wider span support (i.e., longer support). Depending on purpose may require shotcrete.
Raisebored Raises	Up to 4 m diameter	–	–	–	Support not required when raisebored in ABMT-1 domain and non-entry. Final raises should be evaluated on an individual basis, in relation to geotechnical domain models, as some may require shotcrete and/or rock bolts.

The summary for ABMT-2 ground support is presented in Table 16-9.

Table 16-9: ABMT-2 Ground Support Summary

Opening Type	Cross Section (w × h [m])	Ground Support Element	Bolt Length (m)	Bolt Spacing (m)	Notes
Lateral Development	5 × 5.5	#7 Resin rebar	1.8	1.2 × 1.2	100% coverage of back and shoulders. Bolt and screen walls down to 1 m above sill.
		Fibre Shotcrete	–	–	2 inch to 4 inch thickness
Wider Spans	Spans >6 m wide	#7 Resin rebar	2.4	1.2 × 1.2	100% coverage of back and shoulders. Bolt and screen walls down to 1 m above sill.
		Fibre Shotcrete	–	–	4 inch thickness

Opening Type	Cross Section (w × h [m])	Ground Support Element	Bolt Length (m)	Bolt Spacing (m)	Notes
		Coupled fully-grouted #7 Resin rebar OR single-strand 17.8 mm (0.7 inch) diameter cable bolts	Half the span	2.5 × 2.5	Long Secondary Support approximately half the span. Alternatives include expandable rock bolts or connectable expandable rock bolts.
Personnel-Entry Raises	4 × 4	#7 Resin rebar	1.8	1.0 × 1.0	Screen 100% coverage. Brows supported with wider span support (i.e., longer support). Shotcrete thickness depends on purpose of raise, minimum 2 inch.

Cable bolt designs for stopes and large span excavations used rock mass classification and stope surface dimensions to determine bolt length and spacing / density. Kinematic analysis of potential structural wedges that could form due to the joint network was completed. This confirmed the bolt lengths recommended using empirical methods, with the empirical methods producing the most conservative recommendations.

Generally, cable bolting of transverse stope walls will not be required at the 12 m length. Stope backs for transverse stopes will require patterned cable bolts. Longitudinal stopes exceeding 7.5 m span will require longer support in the back.

Cable bolt design recommendations for the Arrow Deposit are presented in Table 16-10.

Table 16-10: Cable Bolt Design in Stope Backs by Geotechnical Domain

Domain	Span (m)	Cable Bolt Density (bolts/m²)	Approx. Cable Bolt Pattern (m × m)	Cable Bolt Length (m)
BMT	<7.5	–	–	–
	7.5 to 10	0.1	1.8 × 2.8	5
	10 to 12	0.2	1.8 × 2.4	6
	12 to 15	0.3	1.8 × 1.8	7
ABMT-1	<7.5	–	–	–
	7.5 to 10	0.25	1.8 × 2.8	5
	10 to 12	0.35	1.8 × 2.4	6
	12 to 15	0.45	1.8 × 1.8	7
ABMT-2	<7.5	–	–	–
	7.5 to 10	0.4	1.8 × 2.8	5.5
	10 to 12	0.45	1.8 × 2.4	6.5

	12 to 15	0.55	1.8 × 1.8	7.5

The Athabasca Basin is seismically inactive as per the National Building Code of Canada. The estimate peak ground acceleration (PGA) with a return period of 4,975 years is less than 0.036 g at a probability of 2% over 50 years (Golder, 2019). The risk of naturally occurring seismic events is low. Because the mining is low-depth to moderate-depth, the risk of seismic events due to mining is low.

16.2.4 Backfill

The Arrow Deposit is a unique backfill project where 100% of all mill processed waste will be deposited UG, either as CPB for stope backfill or as CPT in an UGTMF for permanent deposition. The following materials will be contained in the tailings.

- NLR
- Effluent precipitate
- Gypsum precipitate

Paterson & Cooke (P&C) completed an FS of the backfill system for the Project (Patterson & Cooke Canada Inc., 2020). As indicated by P&C, two modules of the paste plant will be operating in parallel.

The first module will be mixing only NLR, binder, and water to create a CPB suitable for use in the mining stopes. The CPB can be diverted to the UGTMF when stope filling is not required and for high strength plug and cap of each UGTMF chamber.

The second module will be mixing NLR, rejected gypsum (in which the uranium content exceeds 250 ppm), effluent precipitates, binder, and water to create CPT for disposal into the UGTMF.

Backfill capacities and strength targets were established by the project team and agreed upon by NexGen. The paste plant has been designed by the FS team for the following.

- CPB to the mining stopes, at a rate of 80t/h.
- CPT to the UGTMF, at a tonnage rate of 75 t/h.

Paste strength recommendations are summarized in Table 16-11.

Table 16-11: Paste Strength Recommendations

Fill Strength Category	28-Day Strength Requirements (kPa)	Location or End Use	Comments
High	1,500	UGTMF / Undercut Paste	Unchanged from PFS
Medium	1,000	Primary Stopes	Unchanged from PFS
Low	500	Secondary Stopes	Unchanged from PFS

Fill Strength Category	28-Day Strength Requirements (kPa)	Location or End Use	Comments
UGTMF	200	UGTMF Stopes	Prevent liquefaction, provide stability

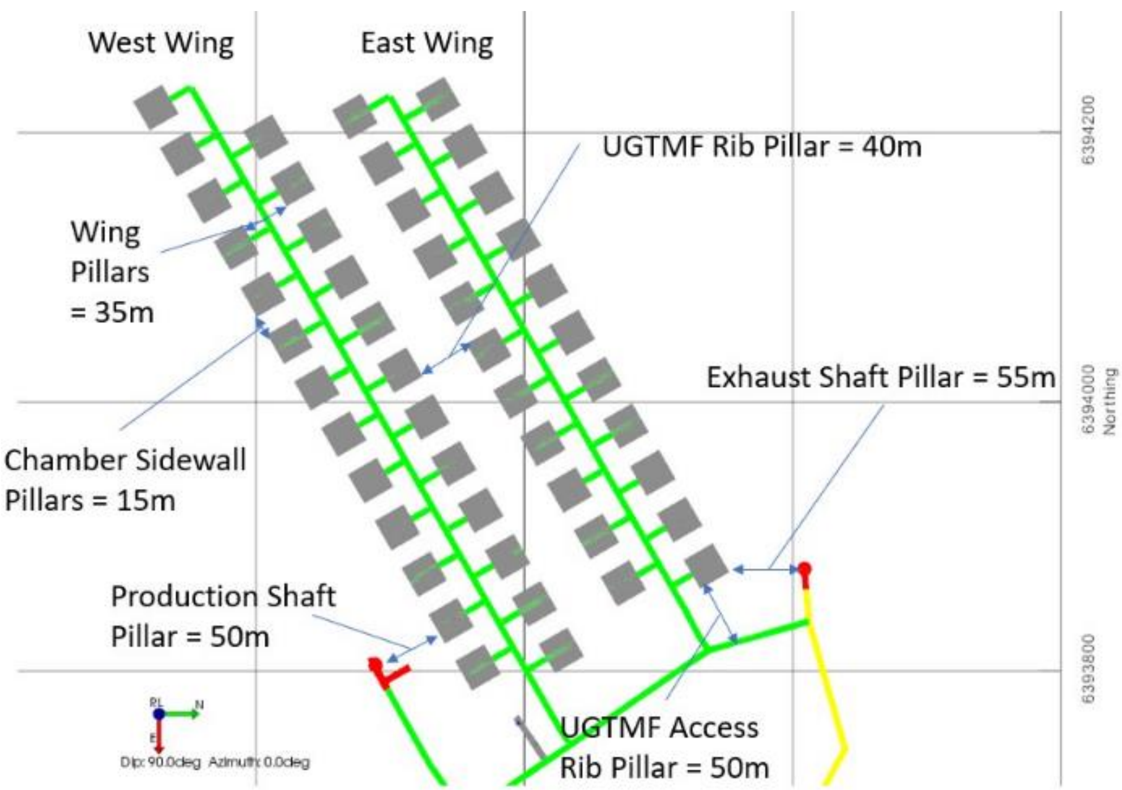
The required paste fill strengths for various applications are based on similar operations and experience, and include review against empirical, analytical, and project-specific numerical models to provide minimum strength requirements. Additional refinement to these models as the project advances will allow the optimization of the paste strength requirements.

16.2.5 Underground Tailings Management Facility

The UGTMF is situated between the Shafts to the north, where hydrothermal alteration is expected to be minimal and ground conditions are expected to be amenable to the proposed excavation and fill sequence for long-term storage. Ground support specifications consistent with the BMT domain are incorporated into UGTMF designs and cost estimates.

The general layout of the UGTMF is presented in Figure 16-7.

Figure 16-7: UGTMF General Layout



Source: Stantec December 2019

The UGTMF infrastructure consists of internal mine access development, wing and chamber access development, and chamber excavations nominally measuring 25 m wide × 25 m long × 60 m high. As per BMT ground support specifications, deep cable support is planned for chamber backs. Access to the chambers in the FS mine plan is required on the 380 Level, 440 Level, and 500 Level.

General rock mass conditions within the UGTMF zone typically range from good to very good using standard rating systems. There is minimal hydrothermal alteration (primarily locally associated with shear interpretations).

Similar to analyses completed for the mine stopes, UGTMF chambers were assessed empirically using the stability graph method. Stability graph results indicate generally stable conditions for the proposed chambers, based largely on the superior rock mass quality of the BMT domain, proposed primary / secondary sequencing, and ability to periodically relocated chambers away from relatively poorer quality rock masses.

Based on a 5.5 m height and 5.0 m width, lateral development within the UGTMF zone ground support will be patterned bolting with screen for most of the development.

Within the chambers proper, backs and exposed shoulders will be supported with minimum 10 m long cable bolts spaced approximately 2 m on centre, as per stability graph results and confirmatory stress assessment modelling.

Sequencing of cell chamber construction will be important to ensure a new cell chamber is not constructed until the adjacent cell is backfilled and cured to maintain stability.

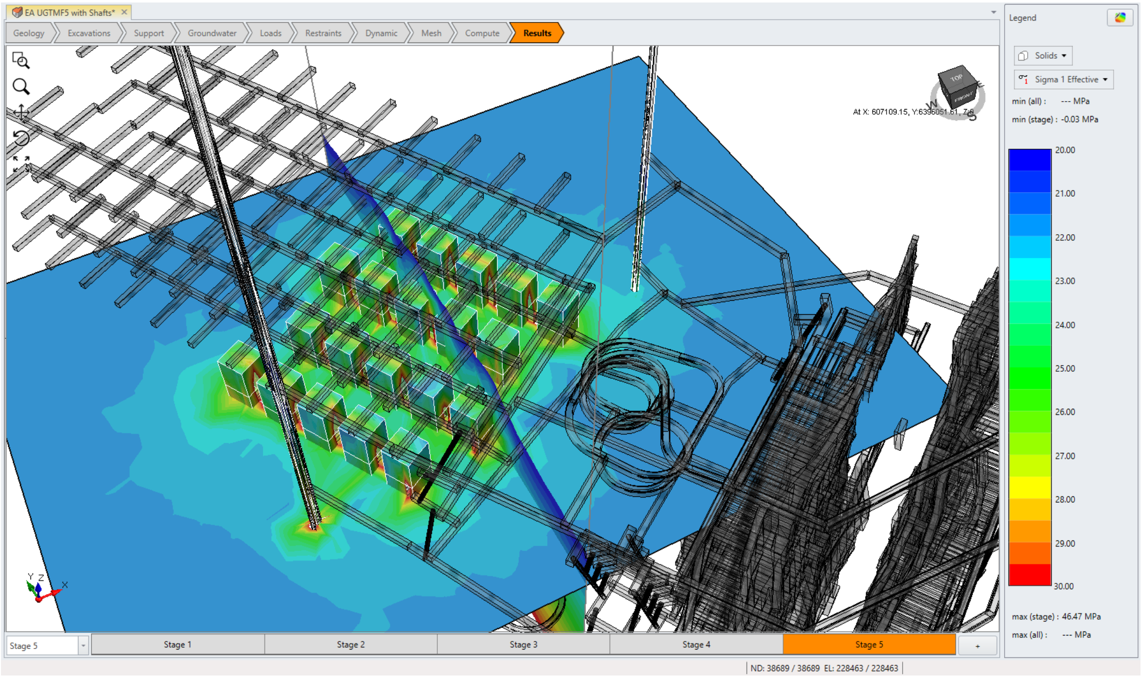
Rock pillars established with the UGTMF excavations included the following.

- Wing pillars
- Chamber sidewall pillars
- UGTMF chamber rib pillars
- Shaft pillars
- UGTMF access rib pillars

The pillars were assessed using primarily 3D stress models to evaluate stress / strength relationships and sensitivity to intact and rock mass inputs. The FS designs include an assessment of potential structural failure mechanisms and empirical methods as a pseudo-validation of the assessment. The analyses indicate that pillars formed during UGTMF development are predicted to exhibit general overall stability conditions, with modeled stress moderately below standard damage imitation threshold criteria.

UGTMF stress assessment of pillars is presented in Figure 16-8.

Figure 16-8: UGTMF Stress Assessment of Pillars



Note: Interstitial Pillar σ_1 <= 0.3 UCS for most areas.

16.2.6 Shaft Geotechnics

The Arrow Deposit will be accessed using two vertical circular shafts for production and mine air exhaust. Both shafts will be used for the LOM. The shafts will be excavated in artificially frozen soil / bedrock. A hydrostatic liner will be installed into geotechnically competent and low-hydraulic conductivity rock mass below the paleoweathered zone in the upper basement / WBMT.

Below the hydrostatic liner, the shafts will advance under probe cover and grouting in advance of the shaft face.

In the Production Shaft, competent basement rock is encountered at approximately 159 m depth, with the paleoweathered zone or WBMT geotechnical domain approximately 50 m thick. Below the WBMT domain is the BMT domain (to the end of shaft). Rock mass quality is classified as follows.

- Good to very good quality.
- Strong to very strong (average 100 MPa), with relatively low fracture frequency (0–5 per metre, with a mean of 1.9 per metre).
- Patterned joint network with random jointing.
- Fair joint conditions (i.e., planar, slightly rough), typical of unaltered basement rock.

In comparison to the Production Shaft, the Exhaust Shaft has a relatively increased vertical thickness of WBMT domain and relatively lowered rock quality, with more intervals of fair quality intermittent with good quality. The WBMT domain transitions to alteration associated with the FW Intrusive, which is interpreted to be predominantly related to thermal processes since the on-rig densities below the unconformity are relatively consistent.

Below the WBMT, the geotechnical domain alternates between BMT and ABMT-1 domain, with an intact rock strength between 50–100 MPa. The zones of ABMT-2 are negligible along both shaft alignments.

The rock conditions are good at all shaft stations. The infrequent and short intervals of fair quality rock are between levels.

The ground support recommendations for the Production Shaft are as follows.

- 2.7 m long rebar on a 1.7 m × 1.7 m pattern, spaced circumferentially and parallel to shaft alignment.
- Welded wire mesh.
- Minimal shotcrete required (approximately less than 5%).

The ground support recommendations for the Exhaust Shaft are as follows.

- 2.0 m long rebar on a 1.7 m × 1.7 m pattern spaced circumferentially and parallel to shaft alignment.
- Welded wire mesh.
- Increased shotcrete requirement relative to the Production Shaft (approximately 10%–15% of the alignment will require 50 mm of shotcrete).

Shaft stations will require brow support, including shotcrete, cable bolts, and a transitioning probing pattern as the vertical shaft transitions to lateral development.

In addition to standard shaft support, shotcrete and cable bolts are recommended to support the shaft station area.

16.3 Mining Method

16.3.1 Mine Access (Shafts)

There are two shafts included in the design that serve as access points to the Arrow Deposit. The Production Shaft and the Ventilation Exhaust Shaft (Exhaust Shaft). The shafts' locations and sizes were designed for the geotechnical and ventilation requirements of the mine, respectively.

Personnel and material will access the mine via the 8.0 m finished diameter Production Shaft, which will have two skips, large cage, auxiliary cage, and the mine services. The Production Shaft infrastructure will include a hoist house, headframe and collar house, two main access shaft stations, loading pocket shaft station, and shaft bottom. The

Production Shaft will have a hoisting capacity of 5,290 t/d, which includes 10% design factor allowance plus 5% moisture content.

The Production Shaft will also provide the fresh air intake for the UG operations. Access to the Production Shaft will be from the 500 Level and 590 Level, along with the 620 Level loading pocket.

Exhaust air will be returned to the surface via the 5.5 m finished diameter Exhaust Shaft. For steady-state operations, this shaft will be bald. The Exhaust Shaft will be used for second egress if the Production Shaft becomes incapacitated. A trailer-mounted, diesel-powered winch will be located at the Exhaust Shaft. This winch can be used to lower an escape pod if second egress is required.

Shaft Freeze and Liner Design

Both shafts will be constructed using ground freezing in conjunction with a hydrostatic liner for the upper portion of the shafts. A freezing of the ground is recommended based on the shaft excavation will proceed through water bearing formations and weak to poor ground. The freeze design for the construction of the shafts entails freezing down to the competent ground located 175.0 m below surface for the Production Shaft, and 220.0 m below surface for the Exhaust Shaft.

Geotechnical information was used to determine the required depth of the freeze at each shaft location. Based on the geotechnical report, the depth of the altered basement rock is deeper in the vicinity of the Exhaust Shaft. For this reason, the freeze design for the Exhaust Shaft is deeper than for the Production Shaft.

Due to the distance between the shafts, there will be a dedicated freeze plant for each shaft. Each freeze plant will have a capacity of approximately 500 t of refrigerant at a temperature of -30 °C. A modular freeze plant was selected, so it can be set up for the construction phase and removed after the freeze is complete and no longer required. There will be four monitoring holes per shaft, which will include in-ground monitoring and resistance temperature detectors (RTDs) for brine temperature monitoring.

The Production Shaft freeze ring will be constructed approximately 12.4 m below grade (collar level) to allow the freeze wall to form below the water table. The fresh air ventilation plenum will also be located at 12.4 m below grade (refer to Section 16.8.1 for a description of the ventilation system). For this reason, the freeze ring will sit within the ventilation plenum for the duration of the freeze. The ventilation plenum will be the primary means of access to the freeze cellar, and a second means of access / egress will be provided via a ladderway to the surface.

The Exhaust Shaft freeze ring will be constructed approximately 10.0 m below grade to allow the freeze wall to form below the water table. Since there is no ventilation plenum for the Exhaust Shaft, two independent ladder ways will be installed to provide access from the surface.

As the shaft is developed, a hydrostatic liner will be installed to prevent water migration into the shaft once the freeze is removed. The Production Shaft liner design will include a 600 mm-thick liner from the surface to 170 m below grade, with a 750 mm-thick section from 170 m to 190 m below grade. The Exhaust Shaft liner design will include a 600 mm-thick liner from surface to 207 m below grade, with a 750 mm-thick section from 200 m to 220 m below grade.

Grout pour locations will be included within the construction of the liner to allow for backwall grouting when the ground behind the hydrostatic liner has thawed and moves away from the liner.

Shaft Sinking

The permanent headframe of the Production Shaft will be used for sinking operations. Temporary stage winches will be installed near the headframe to facilitate shaft sinking. A temporary sheave deck will be installed in the headframe to mount the sinking stage head sheaves used during shaft sinking.

The permanent production hoist will be used for the Production Shaft sinking. These considerations will facilitate efficient changeover from shaft sinking to permanent operations.

A contractor-supplied temporary hoist house and double drum sinking hoist will be installed to sink the Exhaust Shaft. The contractor will provide a temporary headframe for sinking operations at the Exhaust Shaft. The temporary headframe and sinking plant will be removed once complete.

Production Shaft

In addition to hoisting waste rock and ore to the surface, the Production Shaft will provide access for personnel and material entering the mine. The headframe design includes a collar house, sub-collar, ventilation plenum, head sheaves, skip dump, ore bin, and waste bin.

The collar house will have sufficient room to stage personnel and materials to be sent UG. It will include an overhead crane for moving materials to and from the cage. There will be suitable clearances for removing the skips for service and maintenance on the north and south sides of the collar house.

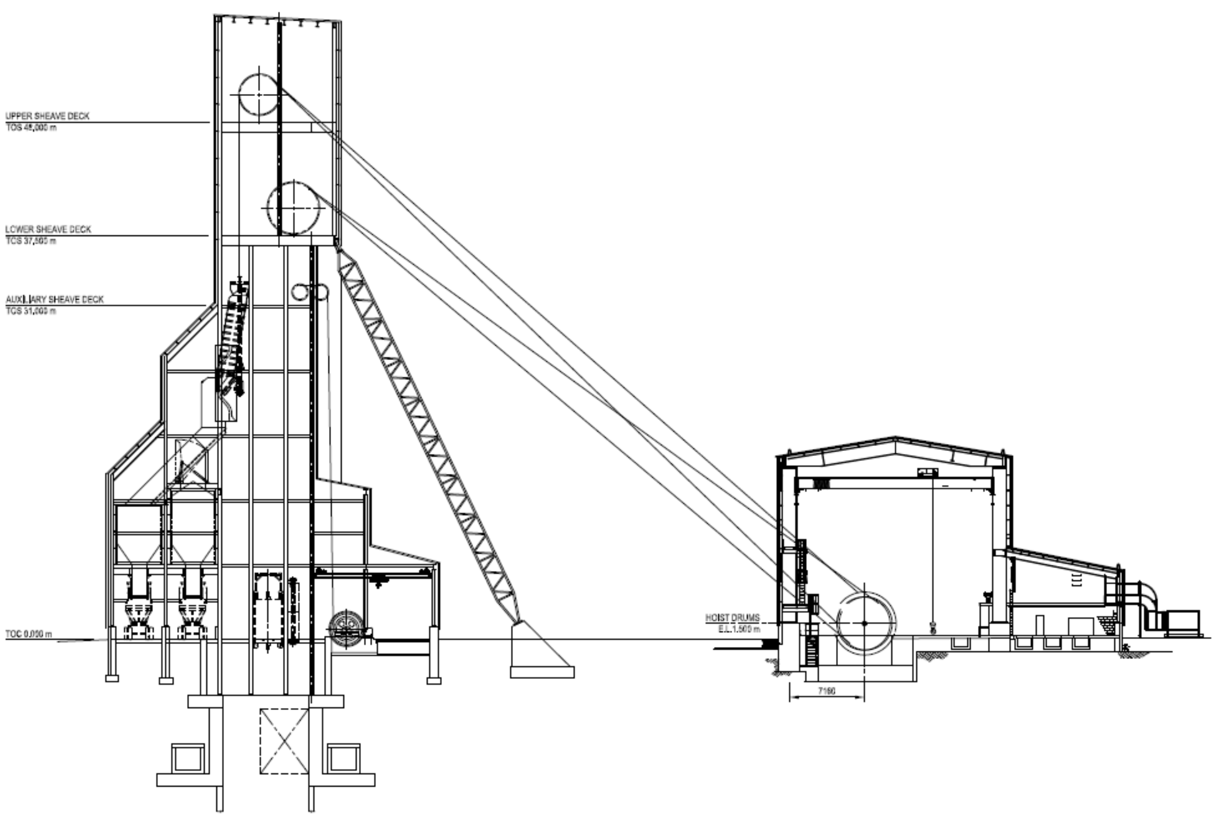
The sub-collar will house the utilities entering the shaft and will include the utilidor connection that leads to the hoist house. The ventilation plenum will join the shaft below the sub-collar to provide fresh air ventilation to the UG workings. The tower of the headframe will be 57.75 m tall and will house the head sheaves and skip-dumping equipment.

Skip dumping will be accomplished using a fixed scroll plate with a diverter gate to direct material into the ore or waste bins as required. The ore and waste bins will each discharge into truck loadouts that will be used to deliver ore to the crushing plant or ore stockpile and waste rock to the designated location on surface.

The hoist house will include two permanent double drum hoists, an auxiliary single drum hoist, compressor room, control booth, electrical room, and an overhead crane. The crane will allow for materials to be moved around the hoist house for operations and maintenance. The hoist plant will be available for hoisting an average of 16 hours per day.

Figure 16-9 presents the Production Shaft headframe and hoist house arrangement.

Figure 16-9: Production Shaft Headframe and Hoist House Arrangement

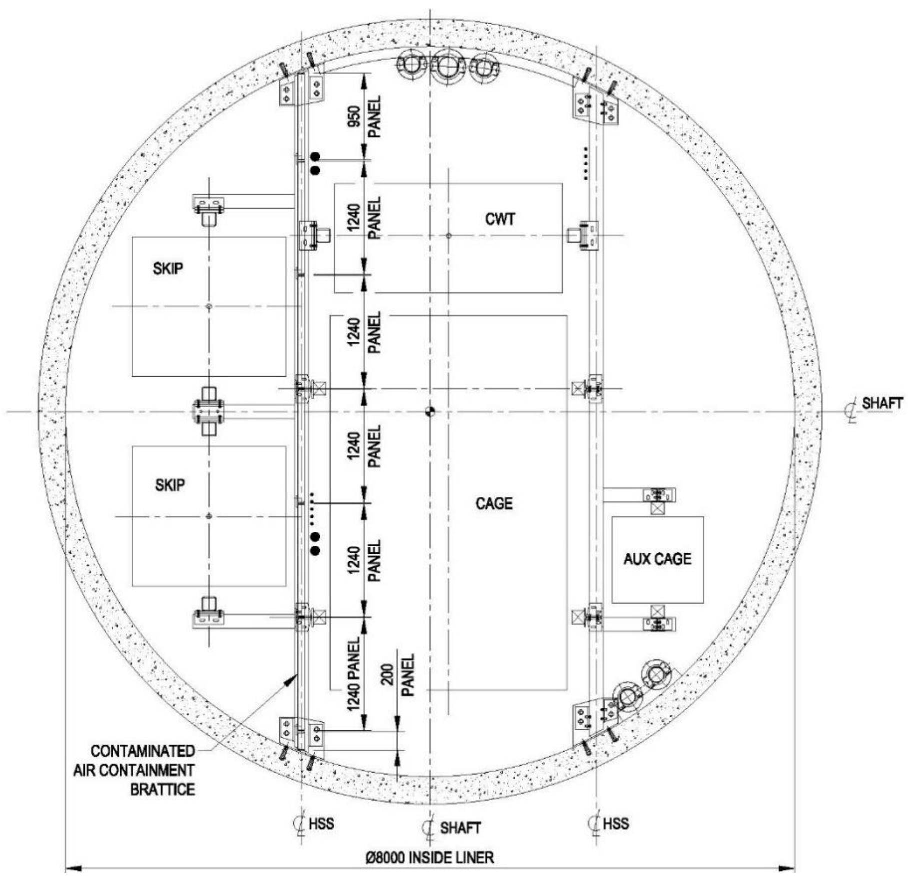


The shaft steel set design uses cantilever buntons with fixed steel guides for the skips. The counterweight also has steel guides. The service cage and the auxiliary cage will both have fixed timber guides. The following piping will be installed on the north side of the shaft: an 8-inch diameter DN200 dewatering line, a 6-inch diameter DN150 fresh water line, and a 6-inch diameter DN150 compressed air line.

There will be two, 6-inch diameter DN150 slick lines located on the south side of the shaft for access to the shaft stations. Solid brattice panels will run the entire length of the shaft to separate the contaminated air within the skip compartments from the fresh air within the cage compartments of the shaft.

Figure 16-10 presents a plan view of the Production Shaft.

Figure 16-10: Plan View of the Production Shaft



The shaft will have stations located on the 500, 590, and 620 Levels. The loading pocket will receive material from the 620 Level. The shaft stations will include utilities on each level, electrical substations, and ventilation doors.

Exhaust Shaft

At the Exhaust Shaft, an airlock is required for the cage pod entering the shaft. In the event of an emergency requiring second egress, the ventilation fans would be turned down or off to reduce the amount of buffeting of the suspended (i.e., unguided) escape pod.

A ventilation duct elbow will be connected from the collar of the Exhaust Shaft to the return air fans. A double airlock arrangement will be installed to allow the cage pod to travel through the exhaust duct elbow into the Exhaust Shaft without disrupting the exhaust airflow to the exhaust fans.

There will be one shaft station at the 500 Level.

16.3.2 Mine Development and Early Ore

The Exhaust Shaft will be used for early development, early ore, and access to the UGTMF area. A temporary loading pocket will be installed on the 500 Level. During shaft sinking, once the 500 Level station is developed the shaft will be excavated approximately 12 m deeper and will be slashed to fit a bin / gate / chute arrangement.

The bins and supports will be constructed, positioned, and sized to match the sinking bucket capacity (not including freeboard). The sinking ventilation duct will be removed and replaced with 1,372 mm (54 inch) diameter rigid duct. The muck gates and chute extensions will then be installed. Chairs for the buckets will be constructed on the galloway.

A grizzly with 600 mm × 600 mm (2 ft × 2 ft) openings will be constructed on the top of the bins with the appropriate guard rails to secure the temporary loading area. This arrangement will allow for mucking the required 4,000 t/d and provide a safe, well-lit area to receive the equipment, gear, and consumables required for the off-shaft development.

Nominal 3.6 tonne (2 yd^3) LHDs will be sized to suit the bin capacity which matches the sinking bucket capacity) and used to load the bins. This arrangement will achieve the required daily tonnage.

The Exhaust Shaft 500 Level will be used for primary development of the mine and some early ore production. The 500 Level station will be developed from the shaft large enough to sling and assemble development equipment. The shaft bottom will be determined by the temporary loading system requirements.

Lateral Development

All decline and lateral excavations will be developed using drill and blast methods and diesel-powered mobile equipment. The mobile equipment required for development activities is as follows.

- Drill – two-boom electric-hydraulic jumbo.
- Blast – mobile explosives loader.

- Muck – 17-tonne class LHD.
- Ground support installation – mechanical bolter.
- Secondary haulage – UG haulage truck (for early off-shaft development prior to passes being established).

Mine development will include all capital and operating lateral and vertical UG development. Lateral development will consist of all level, ore, ramp, and infrastructure development. Vertical development will consist of all raise and pass development.

There will be four main development heading profiles for the UG workings, which are presented in Table 16-12.

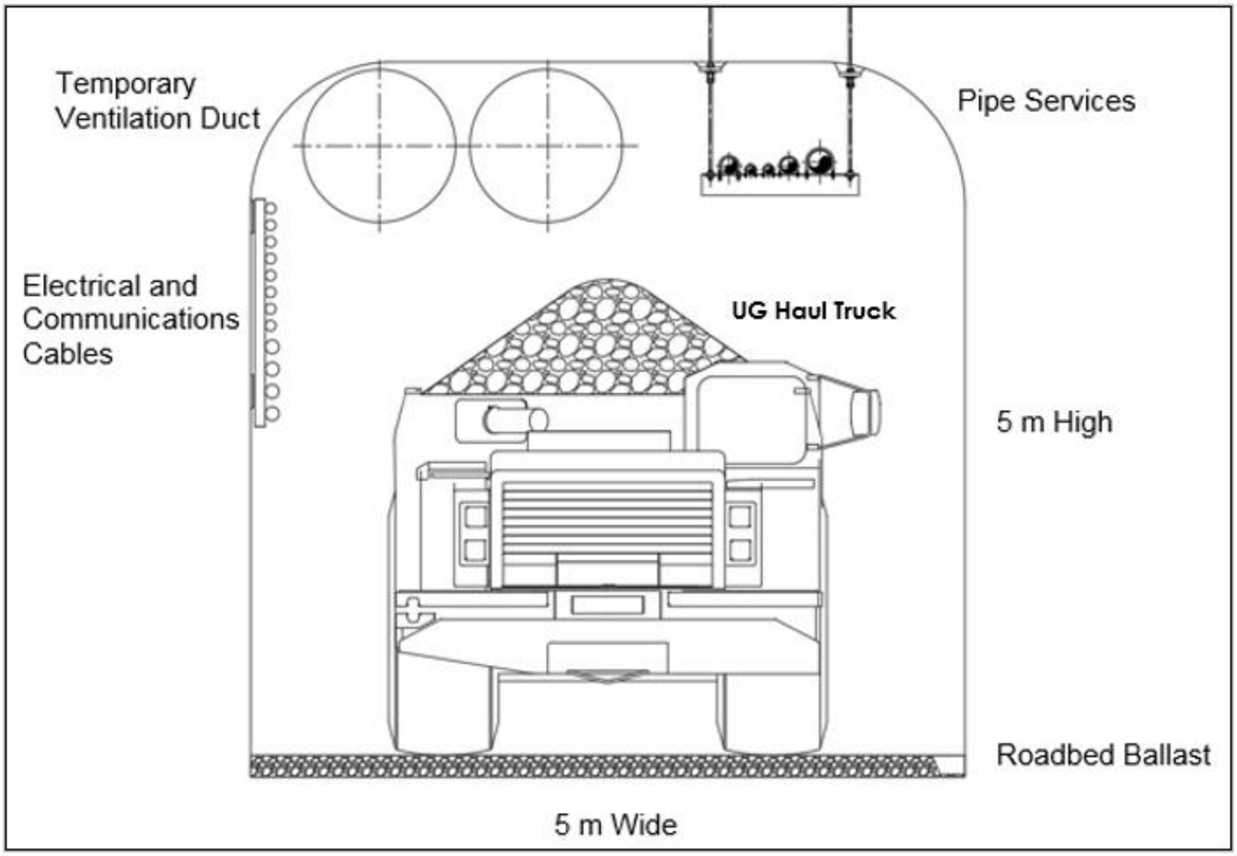
Table 16-12: Main Development Heading Profiles

Heading Profile	Development Uses
5.0 m width × 5.0 m height Flatback	Access drift, conveyor drift, cross cuts into stopes, remucks, and storage facilities
5.0 m width × 5.0 m height Arched back	Ramps
5.0 m width × 5.5 m height Arched back	HW and FW drifts, UGTMF drifts
5.0 m width × 5.0 m height Flatback	Ore sills (driven up to 8.0 m wide in UGTMF stopes)

General arrangement drawings were prepared for larger infrastructure excavations (i.e., shaft stations, rock breaker stations, shops, and sumps), and the excavation dimensions were incorporated in the 3D mine model. Initial pilot drifts will be developed for these excavations. A combination of wall-slashing, floor-benching, and back-slashing techniques will be used as required to achieve the final dimensions.

The access drift profile is presented in Figure 16-11.

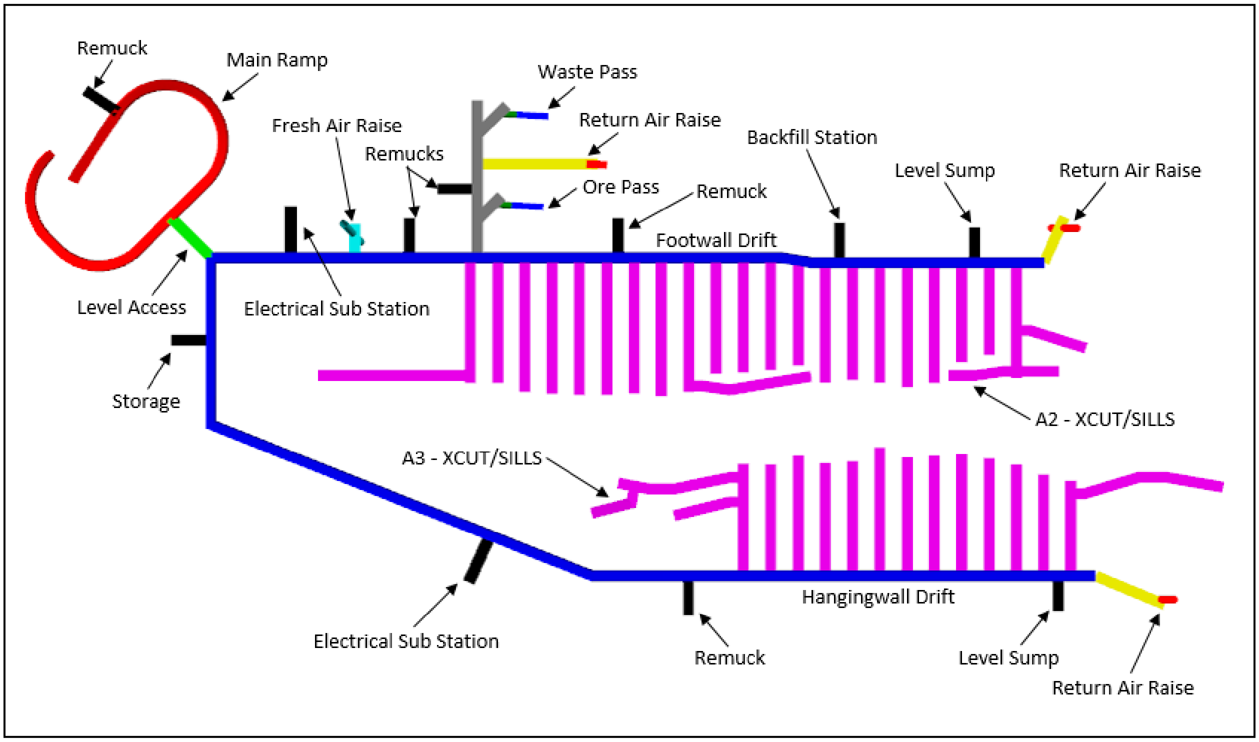
Figure 16-11: Access Drift Profile



Level Development

Levels will be developed at 30 m intervals. Levels will have varying strike lengths, depending on the Mineral Resource at each elevation. A typical level is represented by the 530 Level example in Figure 16-12.

Figure 16-12: Typical Level Development



Each level will have a level access (i.e., the connection between the ramp and the FW and/or HW drift on each level). Ore and waste passes will be accessed via the FW drift. All infrastructure will be located along the FW and HW drifts.

Typical level infrastructure includes the following: storage, sumps, electrical sub-stations, remucks, and paste backfill stations. Each HW and FW drift will tie into the return air system via the return air raise (RAR). This connection will be required prior to ore development.

Development Quantities

The 3D mine model includes all ramp, level, and infrastructure development required to access and extract the reserves. Table 16-13 presents the lateral development quantities by excavation type.

Table 16-13: Lateral Development Quantities*

Item	Metres	Equivalent Metres (5.0 m width × 5.0 m height)	Total Tonnes
Accesses	3,460	3,460	235,000
HW and FW	8,357	9,036	613,812
Ramps	3,495	3,428	232,866
Crosscuts and Sills (Ore and Waste)	16,740	16,740	1,041,259
Undercuts (Ore and Waste)	1,592	1,580	98,563
UGTMF	8,875	8,875	548,008
Ventilation Transfer and Ventilation Raise Access	2,103	2,103	129,898
Ore and Waste Pass Access (including Rockbreaker Stations)	882	968	59,787
Infrastructure (including Conveyor)	2,887	3,439	212,397
Total	**48,391**	**49,630**	**3,171,589**

*An allowance was added to the lateral waste development quantities to account for slashes, corners, and take-down backs.

Development Drilling

Development rounds will be drilled using a fully automated, two-boom electric hydraulic drill jumbo.

Blasting

Development rounds will be loaded using a mobile mechanical explosives' loader.

Early development from the 500 Level shaft station will have opportunity to blast at will. Once multiple headings are established and development commences from the Production Shaft, blasting will be restricted to end-of-shift.

Development Mucking

Development rounds will be mucked using a 17-tonne class LHD. The LHD will muck blasted rock from the face to a remuck bay and subsequently remuck the rock and haul it to the nearest ore or waste pass facility. Prior to passes being established in the early development phase, the LHD will haul to the nearest shaft station or load a haul truck.

Remuck bays will be spaced 150 m apart for long development drives, resulting in an average tramming distance of 75 m.

Ground Support Installation

Ground support installation will be completed using a mechanical bolter. Ground support requirements were identified for the various rock domains that will be encountered during the life of the mine.

The primary ground support will include a 1.8 m long resin rebar installed on a 1.5 m × 1.5 m staggered pattern with a welded-wire mesh screen installed on the back, shoulders, and walls to a depth of within 1.25 m of the floor.

As part of primary ground support to accommodate local poor-quality ground, shotcrete will be applied to 15% of all waste development. Shotcrete will be applied to 100% of the ore headings to reduce potential radiation exposure.

Secondary ground support consisting of cable bolts will be applied to larger areas at intersections and infrastructure excavations. Where possible, four-way intersections will be avoided in the mine design.

At intersections, there will be coupled, fully grouted #7 Resin rebar or single-strand 0.7-inch diameter cable bolts. Approximately half of the span will be installed on a 2.5 m × 2.5 m pattern. There will be an intersection for approximately every 150 m of development.

16.3.3 Vertical Development

Vertical raise development will consist of ventilation raises, ore and waste passes, and bins. The ventilation raises will be excavated primarily using raise boring methods completed by a qualified mining contractor.

The passes will be excavated using an Alimak method to allow for installation of ground support. The larger diameter bins will be excavated using raise boring methods, with slashing and liner / support installation via an Alimak method.

Ventilation Raises

Internal ventilation raises will be predominantly raise-bored and will connect to each production level. All internal ventilation raises will be 3.5 m diameter. The UG internal ventilation raise accesses will include a station for raise bore set-up and gear and rod storage. Internal ventilation raises that are equipped with an escapeway for second egress will have ground support installed.

The shorter exhaust raises may be developed via drop raise method or raise bore method and may not be supported. Geotechnical approval is pending.

Ore and Waste Passes and Bins

The ore and waste handling system will consist of multiple passes feeding rock breakers that size the material prior to loading on the 620 Level conveyor belt. The passes which feed the rock breakers will be driven via Alimak at 3.0 m × 3.0 m and fully supported. Finger raises will be driven off the passes to levels that do not have direct access to the pass.

There will be three rock breaker arrangements in the mine: one on the 500 Level and two on the 590 Level. The 500 Level rock breaker will be predominantly used for the UGTMF stope tonnes. During early mine development, the 500 Level rock breaker will be used to size the production tonnes from the 500 Level stopes. The two facilities on the 590 Level will handle the production ore and waste tonnes.

Below the 500 Level rock breaker, a 3.0 m diameter pass will connect to the 620 Level conveyor. The pass will be raise-bored and supported from the 500 Level.

Below the 590 Level rock breakers, a 6.0 m diameter bin will be excavated to the 620 Level conveyor. The bins will have a live capacity of approximately 500 m^3 or 1,000 t. These bins will be excavated in a multi-pass approach, supported, and lined with shotcrete.

The passes will be supported with 1.8 m long resin rebar on a 1.2 m × 0.75 m staggered pattern. Blasting will be completed on 2.4 m long rounds using ammonium nitrate / fuel oil (ANFO).

16.4 Mining Method Selection

For production mining, a combination of transverse and longitudinal longhole stoping approaches will be used to extract the Mineral Resource.

Longhole stoping is generally associated with steeply dipping ore bodies. Longhole stoping requires dividing the targeted Mineral Resource into individual stopes, and establishing mining levels to access the stopes and position development to facilitate drilling, blasting, and extraction of the material.

Once extraction of material within a stope is completed, the stope will be filled with paste backfill. Longhole is a non-entry mining method (i.e., during mining, personnel will be prohibited from entering the open portion of a stope). Material from within the stope is mucked remotely using LHDs to provide reduced worker exposure to potential ground hazards and radiation.

The transverse approach consists of primary and secondary stopes and is generally applied to areas where the average true thickness (perpendicular to dip) of the Mineral Resource is greater than 10 m. For areas where the true thickness is less than 10 m, the longitudinal approach will be applied. Generally, the approach selected will be modified to best suit surrounding mine areas.

Table 16-14 presents the recovered ore tonnes by mining approach. Most Mineral Resources will be extracted via the transverse approach.

Table 16-14: Ore Tonnes by Mining Method

Mining Method	Recovered Ore Tonnes (millions)	U_3O_8 Grade (%)	U_3O_8 lb (millions)
Longitudinal	1.09	1.10%	26.2
Transverse	2.82	3.06%	190.5
Development	0.44	2.32%	22.4
Total	**4.35**	**2.49%**	**239.1**

Note: Represents mining method only, does not include special waste, marginal ore and waste included in the Mineral Reserve estimate.

The transverse approach will also be applied to the UGTMF; however, instead of utilizing primary and secondary stopes as with production, 15 m wide pillars will be left between each stope.

16.4.1 Level Interval

The level interval for production stoping was first analyzed at the Project onset based on the block model used in the PFS. Level spacings of 20 m, 25 m, and 30 m were analyzed by generating stope shapes utilizing the DSO tool. The stopes were then interrogated to estimate the resource-to-reserve extraction versus the lateral development requirement.

Due to the steep dip in the deposit, there was a minimal variation in the resource-to-reserve extraction for each scenario. 30 m level spacing was selected because it requires significantly less overall waste development.

Level spacing greater than 30 m was not analyzed for production due to the high value of the deposit and the potential lower recovery issues associated with increased stope height.

For the UGTMF, an increased level interval spacing of 60 m was selected because it correlates with the 30 m level spacing for production stoping. 60 m level spacing also allows for decreased waste development.

16.4.2 Mining Blocks

To achieve the planned production rate and allow flexibility in the schedule, production will be required concurrently from multiple mining fronts. Each mining front will be developed using a bottom-up approach.

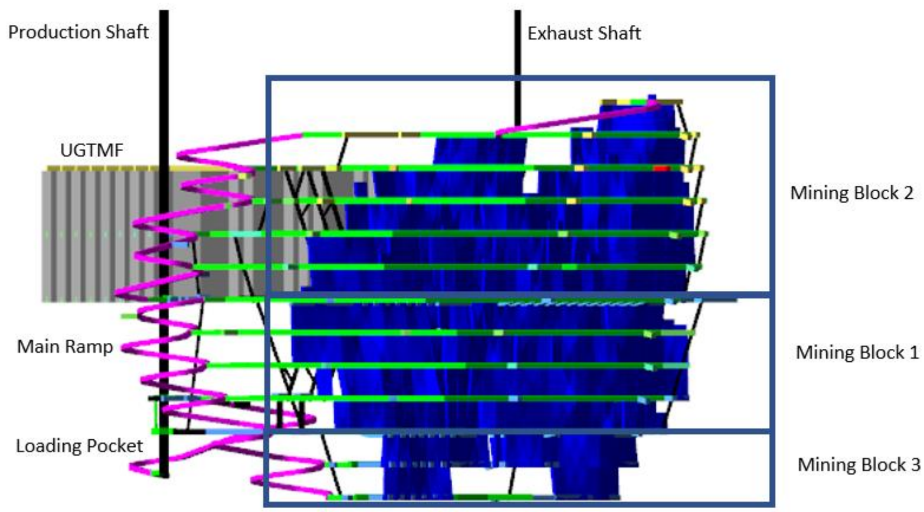
The three main vertical mining blocks are as follows.

- 500 Level to 320 Level
- 620 Level to 500 Level
- 680 Level to 620 Level

Within Blocks 1 and 2, the A2 and A3 veins will be mined as independent blocks from the 620 Level to the 350 Level. Block 3 has limited tonnes and will only augment production from Block 2 once production activities with Block 2 decrease. This will allow for five separate mining fronts, as presented in Figure 16-13.

UGTMF production will commence on the 500 Level with the overcut on the 440 Level. All stopes on the 500 Level will be excavated and filled prior to UGTMF production commencing on the 440 Level with the overcut on the 380 Level.

Figure 16-13: Mining Blocks Looking North



16.4.3 Transverse Longhole Stoping

For transverse longhole stoping, a drift will be established in the FW drift for the A2 and/or the HW drift for the A3. Primary and secondary stopes will be defined at 12 m wide intervals along strike. Each stope will be accessed from the FW or HW, with a crosscut developed through the centre of the stope.

16.4.4 Longitudinal Longhole Stoping

A longitudinal approach will be used in areas where the true thickness of the ore body is less than the transverse longhole stoping area and is greater than 3 m. The longitudinal areas will be accessed from the FW or HW.

From the access, a sill drift will be developed along the strike of the ore body. Stoping will start at the end of each sill and retreat towards the access. Each stope will be 24 m along strike. Once extracted, each stope will be paste-filled prior to mining the adjacent stope.

16.4.5 Sill Pillar

Mining blocks will begin on the 500, 620, and 680 Levels. This will result in sill pillars on the 530 and 650 Levels. To recover 530 Level and 650 Level stoping, undercuts will be developed under the 500 Level and 620 Level paste backfill. Once the undercuts are complete, stoping will commence using the transverse and longitudinal longhole stoping approaches.

16.4.6 Underground Tailings Management Facility

The UGTMF has been designed so that all solid waste generated from the process plant will be returned UG for long-term storage. The UGTMF will use the transverse longhole stoping method. Stopes will be placed at 40 m intervals along strike to allow for 25 m wide stopes and a 15 m wide rock pillar. Each stope will be accessed with a crosscut developed through the centre of the stope.

The NLR, process precipitates (precips), and gypsum generated by the mill will undergo final filtration in the mill before being fed to the paste plant. To prepare the CPB, 66% of all NLR generated will be mixed with water and binder. To prepare the CPT, the remaining 34% of the NLR will be blended with precips, gypsum, water, and binder.

The combined paste tailings will contain a binder content that will depend on the application and required UCS. The binder will use a combination of portland cement and ground slag.

The UGTMF is designed as a series of open excavations, with pillars between each excavation.

The UGTMF will occur in two lifts. Each lift will have two wings of chambers. Each wing will have chambers off both sides of the drift. This layout will ensure storage can be achieved with the minimum amount of capital expenditure.

The UGTMF will be accessed by three levels, as follows.

- 380 Level – overcut of top lift
- 440 Level – overcut of bottom lift, undercut of top lift
- 500 Level – undercut of bottom lift

The dimensions of each excavation will be as follows.

- 60 m height (two mining levels)

- 25 m length
- 25 m width
- 15 m minimum pillar between excavations

Each excavation will have a volume of 33,750 m^3. The shape of each chamber has been optimized for ease of longhole drilling, blasting, and mucking. A total of 97 chambers will be required over the LOM. A high strength plug will be poured to a height above the undercut drift and will require 28 days to cure to a strength of 1.5 MPa. The main body of the UGTMF stopes will be filled with a lower-strength, 0.2 MPa paste fill and will be capped with a high-strength, 1.5 MPa paste fill cap. During the chamber filling and consolidation process of the fill, chamber drains, pipes, and valves will be installed to direct water to the nearest sump.

The following will be intended for any given time.

- Two chambers will be available for deposition.
- Two chambers are in the process of mucking.
- One chamber is blasted, but not mucked out.
- One chamber is drilled off, but not blasted.

Fill times and open exposure times should be minimized. Approximately 11 chambers will be completely cycled per year (i.e., developed, drilled, blasted, mucked out, and filled).

16.4.7 Stoping

All production stopes have individual shapes in the Deswik mine model. A 30 m height, 12 m width for transverse, and 24 m strike length for longitudinal is generally consistent throughout the Mineral Resource; however, sub-shapes using smaller heights and strike lengths were also included to recover the Mineral Resource. The true thickness of the longitudinal stopes and true length of the transverse stopes (i.e., from HW to FW) varies.

The UGTMF stopes are designed to be 25 m wide × 25 m long × 60 m high. Stopes are placed every 40 m along strike, to allow for a 15 m rock pillar to avoid mining next to paste fill.

Stope activities will include longhole drilling, blasting, mucking, and paste filling.

Longhole Drilling

Longhole drilling includes two separate activities: slot raise drilling and production drilling. Both activities will use an in-the-hole (ITH) drill. The majority of production stope drilling will be downhole drilling. Uphole drilling will be used to recover stopes that are not full height. All UGTMF stopes will use downhole drilling.

The production drills will be equipped with control systems and automated functions that will reduce worker exposure and improve safety, hole placement accuracy and precision, and drill productivity. Information (i.e., hole dip, dump, and length) from drilling designs provided by mine engineering will be programmed into the drill. Proper drill ring survey and initial drill set-up on a ring will be critical to achieve proper drilling results. During drilling operations, quality checks on ring mark-up, drill set-up, hole accuracy (i.e., collar location, dip, and azimuth), and breakthroughs will be conducted by mine engineering technicians.

Longhole Blasting

Emulsion products will be used for all longhole stope blasting.

Production Mucking

Blasted ore will be mucked from stopes using 17-t (8 yd^3) class LHDs. The LHDs will be operated remotely from the draw point. One LHD will tram and dump into a remuck and a second LHD will re-handle the ore and dump it into the ore pass system.

Stope Results Evaluation

Following the completion of mucking and prior to backfilling, the empty stope cavity will be surveyed (i.e., a 3D-scanned image of the void will be taken). Mine engineering / geology will then evaluate the stope results against the planned design (i.e., tonnes mined, external dilution, and recovery) and will reconcile the tonnes and grade of the actual stope versus the planned tonnes and sampled grades.

This reconciliation exercise will allow the operation to adjust the stoping process as part of an overall site continuous improvement program. The stope cavity survey will also be used for mine planning for adjacent stopes.

Paste Fill Cycle

A paste fill barricade will be constructed at the stope draw point to contain the initial paste backfill plug that will be poured. The barricade will have drainage to allow the stope to decant water and relieve pressure buildup on the barricade.

Table 16-15 presents the backfill component of the stope cycle time for each stoping approach.

Table 16-15: Backfill Cycle Parameters

Parameter	Longitudinal	Transverse	UGTMF
Backfill Density (t/m^3)	1.63	1.63	1.63
Barricade Construction (days)	2	2	2
Plug Cure Time (days)	4	4	28
Body Cure Time (days)	7	7	7
Fill Rate (t/h)	80	80	75

16.5 Backfill – Underground

The CPB and CPT will be delivered from the paste plant located on surface adjacent to the mill, where it will be pumped down one of three surface boreholes, ranging from 65°–70° inclination from horizontal. The boreholes will be drilled from surface to the 440 Level that will be used in the initial years of mine production. When upper levels of the mine are developed, the rock around the boreholes will be blasted on the 380 Level, and the casing pipe will be cut and routed for CPB and CPT to be delivered on the 380 Level and lower levels.

The surface boreholes will be cased with ceramic-in-epoxy lined steel pipe for enhanced wear protection. The boreholes will be fed from the three paste modules on surface.

For geotechnical stability reasons, the surface boreholes will breakthrough into individual cutouts. Diverter valves will be used as dump valves (i.e., one on each borehole) to divert backfill to a sump area near the cutouts in case of process upsets or emergency. The initial rerouting of piping for CPB to be diverted to the UGTMF will happen manually via a removable piping elbow.

By Year 2 of UG operations, a total of three boreholes and seven automated diverter valves will be required at the breakthrough location to fully automate the backfill flow from the boreholes to the stopes and UGTMF. Three diverter valves will be required under each of the three boreholes, respectively, as emergency dump valves. Four diverter valves will be required to route the backfill to the two distinct areas of the mine. One borehole will be fully dedicated to the UGTMF; the other two boreholes can be used for the UGTMF or the mine stopes.

Two side-by-side DN 100 pipeline systems will be installed to all major areas of the UGTMF. A twinned system is specified to prevent a potential operational issue that could occur if two boreholes were used at the same time and the flows merged UG at the change-over station. One DN 100 pipeline system will be installed to all areas of the mine stopes. Figure 16-14 and Figure 16-15 present the proposed borehole cutouts and general piping arrangements for the change-over station.

Figure 16-14: Surface Borehole Breakthrough Cutouts – 440 Level

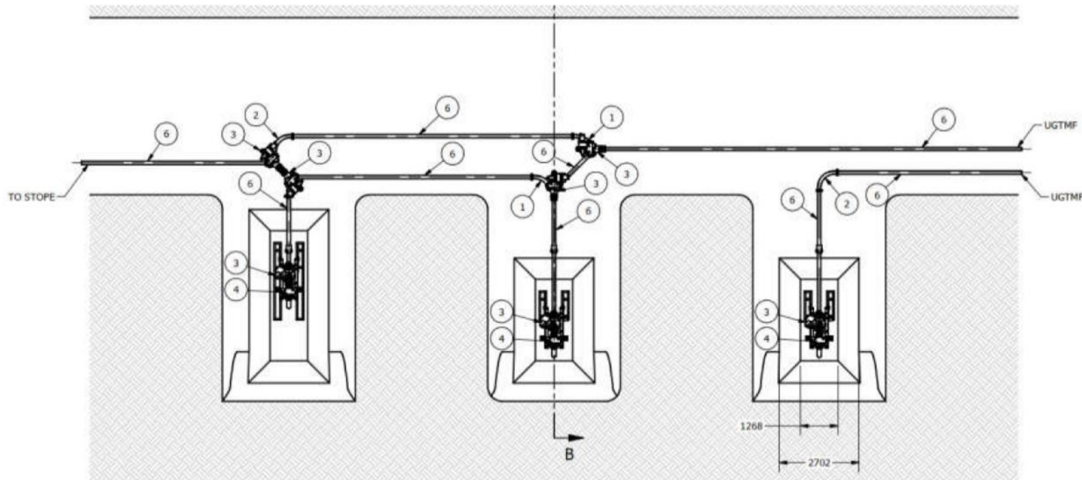


Figure provided by Paterson and Cooke, 2020

The routing of the UG distribution system (UDS) between levels will be completed using 70° inter-level boreholes. Separate, inter-level boreholes will be used for routes to the mine stopes or to the UGTMF.

Once UG, distribution of the paste to the various working areas will be via manual switch-overs, from the main trunk lines to the level piping and eventually the stope piping and UGTMF chamber piping.

Instrumentation will be installed in key locations to report pressure data to the plant operator. Emergency blast-off spools and manual knife gate valves will be installed at critical locations. Manual valves near the discharge to the stopes and UGTMF chambers will be used to divert flush water to containment areas provisioned in the mine.

Figure 16-15: Surface Borehole Breakthrough Piping General Arrangement (Future)

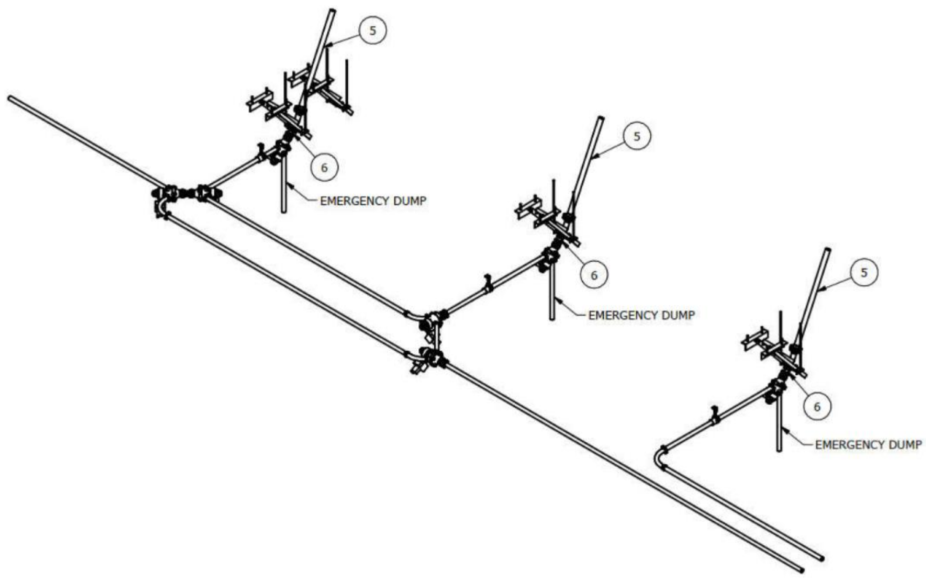


Figure provided by Paterson and Cooke, 2020

The preparation of CPT is directly proportional to the amount of material processed through the plant. For each tonne of processed material, 0.82 m^3 of NLR will result, along with 0.32 m^3 of combined waste precipitates. For CPB, a combination of NLR and binder will be used. The CPT will be combination of NLR, and precipitates mixed with a binder to achieve the required fill strength.

The amount of binder will depend on the paste application. Primary transverse stopes and longitudinal stopes with adjacent stopes on strike will require the highest level of binder. Based on a steady-state production rate, an average demand is 210,000 m^3/year for CPB and 310,000 m^3/year for CPT.

The paste fill system is designed to operate 24 hours per day. In a scenario where production stope voids are not available, the CPB will be diverted to an available UGTMF stope. A UGMTF stope must always be available and the backfill piping must be installed in advance for changeover from one UGTMF stope to the next within one hour. An UG backfill crew will install backfill lines, build barricades, divert paste to the ore stopes and UGTMF, and monitor filling progress.

16.6 Productivities

16.6.1 Effective Hours

The UG operations will consist of two 12-hour shifts per day, seven days per week. Time worked during these 12-hour shifts is expected to fall into one of two categories: effective or non-effective. Non-effective time may result from necessary parts of the job (e.g., refueling and shift safety meetings) during which workers may not be actively conducting mining work (i.e., effective time).

The worker effective time per shift was estimated to consider the amount of non-effective time during a shift. Table 16-16 presents the estimated worker effective time per shift.

Table 16-16: Estimated Worker Effective Time per Shift

Activity (Non-Effective)	Time	Unit
Lunch	30.0	min
Breaks	30.0	min
Morning Lineup	15.0	min
Weekly Safety meeting	9.0	min
Blasting Delays	30.0	min
Shifter's Visits	10.0	min
Engineer / Senior Supervision Outside Interference	10.0	min
Travel in Shaft	22.0	min
Total Non-Effective Shift Time	**156**	**min**
Total Non-Effective Shift Time	**2.6**	**hr**
Total Shift Length	**12**	**hr**
Total Effective Shift Length	**9.4**	**hr**

16.6.2 Labour

The UG labour will consist of contractors for shaft sinking, lateral development, raising, and major UG construction projects, along with NexGen personnel for operations, sustaining capital, and miscellaneous UG construction projects. The UG labour will peak at 237 people on-site in Year -1.

At this time, the mine will have just commenced production, and there will ongoing capital lateral development, exaction of ventilation raises and bins, and ongoing major construction activities. The total labour (peak) by year is shown in Table 16-17. The contractors are split between the shaft sinking contractor and the contractor for lateral development, raising, and construction. The NexGen UG labour is separated into the following four main groups.

- Management
- Technical Services
- Mine Operations
- Mine Maintenance

Table 16-17: Total On-site Labour – Underground Only

Labour Type	Year of Mine Life														
	Y-4	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Shaft Sinking – Contactor															
Staff	13	23	14	0	–	–	–	–	–	–	–	–	–	–	–
Hourly	75	106	50	0	–	–	–	–	–	–	–	–	–	–	–
Lateral Development – Contractor															
Staff	0	0	43	26	–	–	–	–	–	–	–	–	–	–	–
Hourly	0	0	35	164	–	–	–	–	–	–	–	–	–	–	–
NexGen UG Labour															
Mine Management	0	1	1	2	7	7	7	7	7	7	7	7	6	6	5
Technical Services	0	0	0	1	24	24	24	24	24	24	24	24	23	22	14
Mine Operations	0	0	0	32	94	96	94	92	86	72	70	66	61	59	55
Mine Maintenance	0	0	0	12	38	40	40	40	38	34	35	33	32	31	26
Total – On-site	**87**	**129**	**143**	**237**	**163**	**167**	**165**	**163**	**155**	**137**	**136**	**130**	**122**	**118**	**100**
Total – Payroll	**175**	**258**	**286**	**473**	**324**	**332**	**328**	**324**	**308**	**272**	**270**	**258**	**242**	**234**	**198**

Note: Numbers may not sum due to rounding.

16.6.3 Development

The lateral development advance rates were divided into the components of the drill-blast-muck-bolt cycle and estimated from first principles. Advance rates were developed for single and multiple heading scenarios and for the three separate geomechanical domains (refer to Section 16.2.1 for a description of each domain). In the scheduling program (i.e., Deswik), single heading rates were used to limit the daily advance of each heading and multiple heading scenarios were used to define the total daily advance for the mining area.

The rates reflect the advance each drill jumbo crew and its associated gear will achieve over extended periods of operation. These rates were benchmarked against other operations and against the experience of the project team members. The rates reflect long-term averages and include an efficiency allowance. The efficiency allowance accounts for interferences from other activities and conflicting priorities that occur during the operating period.

To determine an average advance rate for scheduling and estimating purposes, the single heading lateral development advance rates for the four major heading types were averaged across the three geomechanical domains.

Table 16-18 presents the single heading lateral development advance rates for each heading type.

Table 16-18: Lateral Development Advance Rates

Heading Type	Single Heading (m/day)
Off-Shaft Development 6.0 m × 6.0 m	2.1
5.0 m × 5.0 m Flatback Accesses	3.9
5.0 m × 5.0 m Arched Ramp	3.7
5.0 m × 5.5 m Arched HW / FW	3.7
5.0 m × 5.0 m Flatback Ore Sills	2.9

The multiple heading rates for a mining complex varies, based on the productivities of the drill jumbos at 11 m/day and the number of available jumbos and faces. The quantity of remaining equipment required to support lateral development was calculated based on the total advance rate and the productivity of each piece of equipment. The productivities of the direct mobile equipment are presented in Table 16-19.

Table 16-19: Lateral Development Mobile Equipment Productivity

Mobile Equipment	Productivity (Average)
Jumbo	11 m/day
Bolter	7 m/day
LHD	675 t/d
Scissor Lift	One per Jumbo

For the initial development (approximately 150 m) out of the shaft stations on the 500, 590, and 620 Levels, the advance rate for the crew reflects the following: drilling and blasting the top cut of the 6.0 m × 6.0 m heading, supporting the back, and then excavating the bottom cut.

At this stage, there will be insufficient space for a full set of equipment; therefore, bolting will be completed from a leveled muck pile until there is room to sling down a scissor lift. This process will be followed by a bolter, as more space becomes available. The resulting advance rate will be an average of 2.1 m/day.

The first development UG will occur on the 500 Level from the Exhaust Shaft. The focus will be on development, with minimal interference from other activities. There will be the opportunity for in-shift blasting during this initial development.

Once this development reaches the Production Shaft, flow-through ventilation can be established, and a second development crew can commence on the 500 Level. This will eliminate the ability to conduct in-shift blasting. Additional equipment will be added to the 500 Level to achieve a total advance rate of approximately 16 m/day. All material produced will be hoisted up the Exhaust Shaft through a temporary loading pocket on the 500 Level.

Once the Production Shaft sinking is complete and the shaft is equipped and commissioned, development will commence on the 620 Level followed by development on the 590 Level. The initial development rates on these levels will be 1.8 m/day until additional equipment can be added to the crew.

The development material on the 620 Level will be hauled to a remuck near the loading pocket and subsequently hauled to the temporary loading pocket arrangement installed on the 620 Level. The development material on the 590 Level will be hauled to a waste pass near the shaft that will transfer the material to the 620 Level. From here, it will be hauled to the temporary loading pocket arrangement.

The breakdown of time allotted for each element of the development cycle regarding a 5 m × 5 m waste rock heading in good quality ground is presented in Table 16-20.

Table 16-20: Development Cycle for 5 m × 5 m Round (Good Quality Ground)

Item	Cycle Time
Drill	3.4 hr
Blast	2.3 hr
Muck	4.1 hr
Ground Support	6.5 hr
Shotcreting Allowance	0.7 hr
Services Install (Critical Path)	0.6 hr
Total Cycle Time	**17.6 hr**

Once the headings advance to allow for a full complement of equipment (assuming the cycle presented in Table 16-20), the advance will be an average of 3.7 m/day for a single heading in waste and an average of 2.9 m/day in ore. Vertical development (i.e., raises) will be developed using both raise boring methods and Alimak mining.

The vertical development advance rate using the raise boring method will be approximately 3.5 m/day for a raise with a 3.5 m diameter. This advance rate includes setup and teardown (12 to 14 days) as well as pilot hole drilling (21.4 m/day) and reaming of the raise (7.9 m/day).

The vertical development advance rate using the Alimak method would be approximately 2.7 m/day for a raise with a 3.5 m diameter. This includes drilling, blasting, mucking, bolting, and installation of rails, and excludes setup and teardown.

16.6.4 Stope Productivity

Stope production rates were divided into the components of the drill-blast-muck (DBM) and backfill cycle and estimated from first principles. The time allotted for each element of the DBM cycle for an average transverse stope (i.e., 12 m wide × 11.7 m long × 30 m high) is presented in Table 16-21.

The 40.4-day cycle time averages an estimated 244 t/d for a typical 9,800 t stope. To achieve the 1,300 t/d production target, 5–6 stopes would have to be active at any time. This is less than the tonnes produced from development.

Table 16-21: DBM Cycle of a Typical Transverse Stope

Element of the DBM Cycle	Time allotted (days)
Bottom Sill Slash	2.9
Stope Preparation	1.0
Drilling	9.1
Blasting	3.3

Element of the DBM Cycle	Time allotted (days)
Mucking	7.2
Barricade	2.0
Backfilling (includes cure and pour)	14.9
Total Cycle Time	**40.4**

Stoping DBM productivities were divided into the three main mining methods: transverse, longitudinal, and UGTMF with representative stope sizes and average productivities presented in Table 16-22.

Table 16-22: DBM Cycle for Representative Stope Sizes

Method	Stope Dimensions			Tonnes (excludes sill)	Cycle Time (Days)	Tonnes per Day
	Length (m)	Width (m)	Height (m)			
Transverse	11.7	12	30	9,834	40.4	244
Longitudinal	24	4.6	30	6,820	34.7	200
UGTMF	25	25	60	91,081	214.9	426

16.7 Mine Development and Production Schedules

All mine development and production scheduling has been completed using Deswik scheduling software. The schedule is interactively linked to the 3D mine model. All development and production scheduling is based on dependencies linked within the mine model. All data is contained within the mine model and schedule. The total tonnes of ore and waste produced annually is shown in Table 16-23.

Table 16-23: Summary of Ore and Waste Tonnes Annual (tonnes × 1,000)

Material	Year or Mine Life													
	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Total
Total Waste Tonnes	138	522	1,146	1,368	1,097	900	1,090	901	756	741	673	753	502	**10,585**
Total Ore Tonnes	0	14	247	424	410	471	456	454	455	455	369	335	258	**4,349**

16.7.1 Development Scheduling

Mine development will be divided into the following three main phases of activity.

Phase 1 – 500 Level Exhaust Shaft Development

The first phase of development will include the initial development from the 500 Level Exhaust Shaft station. During this period, the primary development focus will be to connect to the 500 Level Production Shaft station. This will allow for flow-through ventilation. The connection is necessary prior to adding additional development equipment and capacity on the 500 Level and above.

Phase 2 – 590 Level and 620 Level Production Shaft Development

The second phase of development will consist of development from the 590 and 620 Level Production Shaft stations. Primary focus during this period will be to develop to the resource on the 590 and 620 Levels to establish the ore and waste pass systems, as well as a connection between the two levels.

Along with development on the 590 and 620 Levels, development will continue on the 500 Level. The primary focus will be to establish the return air drift system, initial infrastructure, and ore development. Concurrently, development will begin above the 500 Level, with a ramp being developed to the 440 Level. Development on the 440 Level will be required as part of the initial UGTMF system.

Phase 3 – Ongoing Development

The final phase of development will consist of ongoing development from the upper mine (above the 500 Level) and lower mine (below the 500 Level) concurrently. During this period, full production will be achieved and a connection between the lower mine and upper mine will be established.

The total development is separated into Capital and Operating and summarized annually in Table 16-24. The capital and operating development was divided by the work breakdown structure (WBS) and is shown in Table 16-25 and Table 16-26.

The total vertical development, excluding the excavation of the shafts, is shown in Table 16-27.

Table 16-24: Summary of Lateral Development Annually – Capital and Operating (metres)

Development Type	Year of Mine Life													Total
	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	
Capital	2,102	5,144	7,916	6,075	4,434	4,290	3,749	895	0	0	0	0	0	34,606
Operating	–	616	1,173	3,023	2,798	2,257	1,708	981	503	251	586	594	534	15,025
Total	**2,102**	**5,759**	**9,089**	**9,098**	**7,232**	**6,548**	**5,457**	**1,876**	**503**	**251**	**586**	**594**	**534**	**49,630**

Table 16-25: Summary of Capital Lateral Development Annually (metres)

WBS	Development Type	Year of Mine Life								Total
		Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	
1320	HW/FW Drift	165	1,763	2,194	2,831	2,029	2,242	596	513	**12,332**
1330	Ramp	268	478	1,675	347	285	376	–	–	**3,428**
1340	Access Drift	718	389	552	51	133	154	–	–	**1,997**
1350	Infrastructure	303	1,034	1,683	443	437	385	50	71	**4,408**
1360	Ventilation Access	295	395	533	285	292	236	47	20	**2,103**
4100	UGTMF Access	353	1,086	1,278	2,118	1,258	899	3,055	290	**10,337**
Total		**2,102**	**5,144**	**7,916**	**6,075**	**4,434**	**4,290**	**3,749**	**895**	**34,606**

Table 16-26: Summary of Operating Lateral Development Annually – Ore and Waste (metres)

WBS	Development Type	Year of Mine Life												Total
		Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	
1000	Undercut Sills - Waste	–	419	970	1,008	634	238	278	263	31	4	174	65	**4,085**
	Undercut Sills - Ore	–	755	2,053	1,789	1,623	890	703	240	160	57	397	76	**8,744**
1380	Overcut Sill Pillar - Waste	–	–	–	–	–	316	–	–	53	294	–	303	**966**
	O/C Sill Pillar - Ore	–	–	–	–	–	264			7	232	23	89	**614**

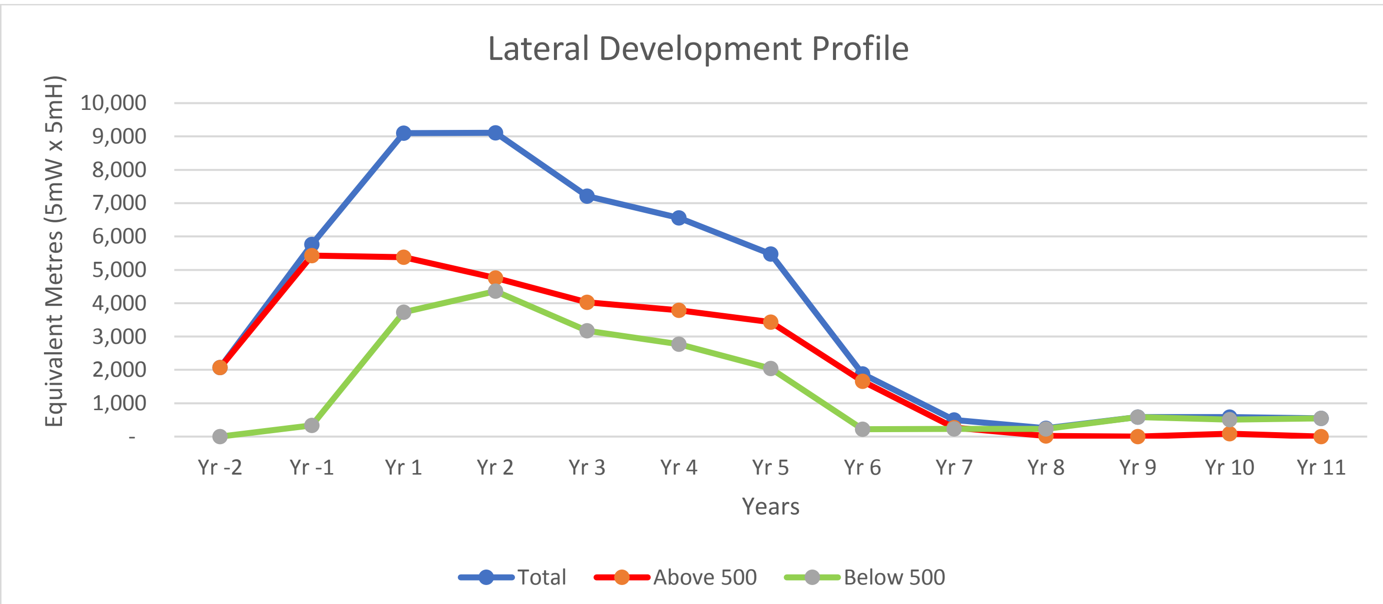
WBS	Development Type	Year of Mine Life												Total
		Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	
8100	Pre-Production Ore	197	–	–	–	–	–	–	–	–	–	–	–	**197**
8150	Pre-Production Waste	419	–	–	–	–	–	–	–	–	–	–	–	**419**
Total		**616**	**1,173**	**3,023**	**2,798**	**2,257**	**1,708**	**981**	**503**	**251**	**586**	**594**	**534**	**15,025**

Table 16-27: Summary of Vertical Development Annually (metres)

WBS	Development Type	Year of Mine Life							Total
		Y-1	Y1	Y2	Y3	Y4	Y5	Y6	
1371	Ore and Waste Passes	169	480	26	26	26	–	–	**727**
1372	Ore and Waste Bins	–	52	–	–	–	–	–	**52**
1373	Ventilation Raises	80	477	321	214	44	59	80	**1,274**
Total		**249**	**1,009**	**347**	**240**	**70**	**59**	**80**	**2,053**

Figure 16-16 presents a graph of the LOM development schedule.

Figure 16-16: Lateral Development Profile



16.7.2 Production Scheduling

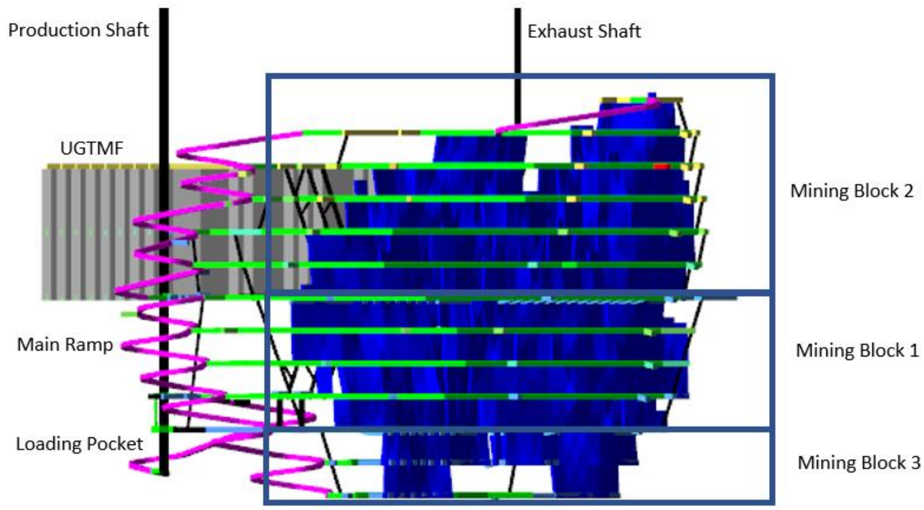
The production target for the Arrow Deposit is to achieve 30 Mlb of packaged U_3O_8 per annum after applying metallurgical recovery. To achieve this production target, the following constraints were applied.

- The process plant can handle a maximum of nominally 455,000 tonnes/annum (t/a).
- The process plant can handle a maximum grade of 5% U_3O_8.
- The process recovery is estimated to be 97.6%.

In general, the A2 vein is higher grade than the A3 vein, and the highest-grade stopes are clustered around the 500 Level. For scheduling purposes, the mine was divided into three vertical mining blocks, refer to Figure 16-17.

Initially mining activities will commence from both Block 1 and 2, and in the A2 and A3 vein, for a total of four separate production areas. Having four separate production areas will provide operational flexibility for mine scheduling and sequencing. The daily ore production rate will range from 1,000–1,300 tonnes/day, and will average 1,207 tonnes/day over the LOM. As production mining is completed on the 620 Level, production will commence in Block 3. Block 3 has limited tonnes and will only augment production from Block 2.

Figure 16-17: Mining Blocks Looking North



In the area of transverse stopes, a primary and secondary stope system will be used to maximize recovery. Primary stopes will be recovered first on the initial level, followed by primary stopes on two vertical levels above, and then secondary stopes on the initial level. This sequence is primarily a result of geotechnical considerations.

The production plan will focus on optimizing ramp-up and maximizing productivity. Targets for the ramp-up will include 1.5 Mlb of U_3O_8 in Year -1, 26.5 Mlb of U_3O_8 in Year 1, and then full production of 30 Mlb of U_3O_8 in Year 2 and onward. To help achieve these targets early in the life of the mine, higher grade areas on the 500 Level will be prioritized.

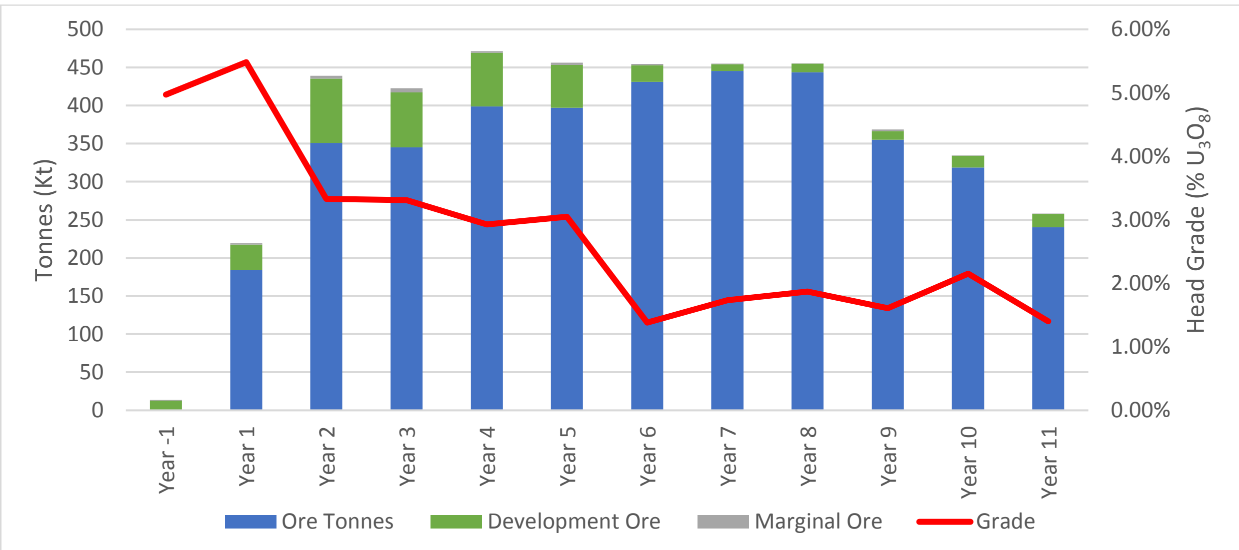
Table 16-28 and Figure 16-18 present the production summary and production profile.

Table 16-28: Production Summary

Mining Method	Recovered Tonnes (thousands)	U_3O_8 Grade (%)	U_3O_8 lb (millions)
Stoping	3,910.7	2.51%	216.7
Development Ore	415.3	2.43%	22.3
Marginal Ore (Development)	23.2	0.28%	0.1
Total	**4,349.1**	**2.49%**	**239.1**

Note: Represents tonnes recovered during mining, does not include special waste, and waste included in the Mineral Reserve estimate.

Figure 16-18: Production Profile



The 1.5 million lbs target in Year -1 will be achieved via ore development only, as the permanent ore handling system will not be in place to handle stoping-sized material until the end of the Year 1.

Full production of 30 Mlb/year will be achieved for approximately four years (Year 2 through Year 5), after which the U_3O_8 pound profile will reduce, even though the overall mine tonnage will remain constant (i.e., at or near design capacity). The anticipated reduction in U_3O_8 pounds is correlated to the reduced average head grade.

Stope sequencing in each mining block is directly related to which stoping method is being used. Guidelines for transverse and longitudinal stope sequencing are presented in Figure 16-19 and Figure 16-20.

Figure 16-19: Transverse Stope Sequence (Cross Section)

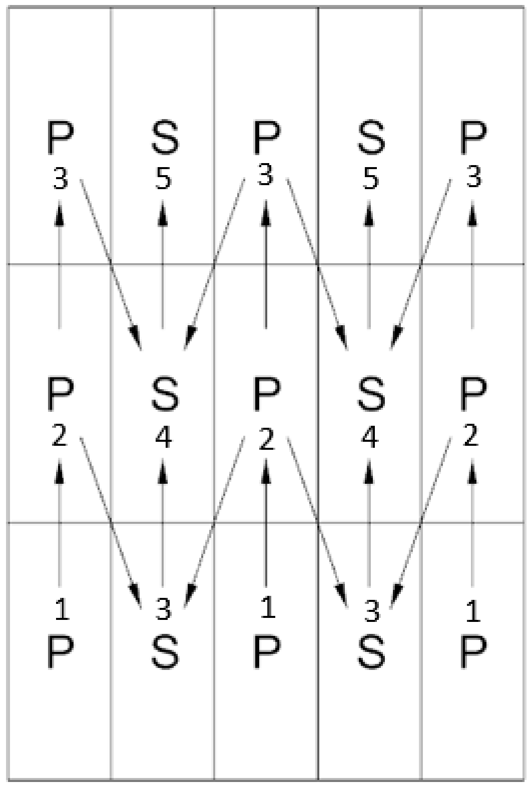
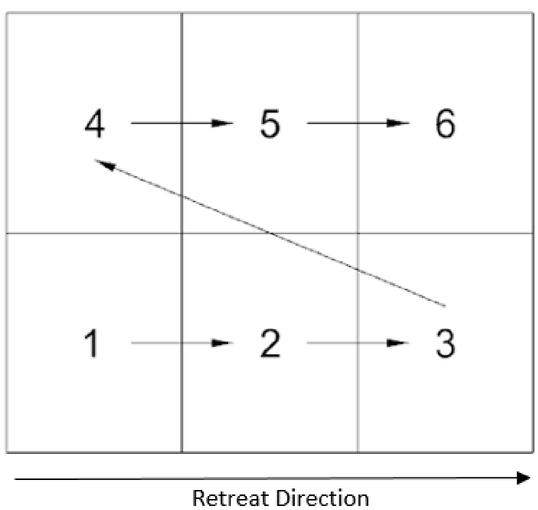


Figure 16-20: Longitudinal Stope Sequence (Long Section)



16.7.3 UGTMF Scheduling

UGTMF stopes will be required as the mill begins to process tonnes and produce tailings. The UGTMF requirements and schedule were derived directly from the mill processing schedule to ensure there is sufficient storage for tailings. During the Project period, prior to the process plant operating, three UGTMF stopes will be excavated.

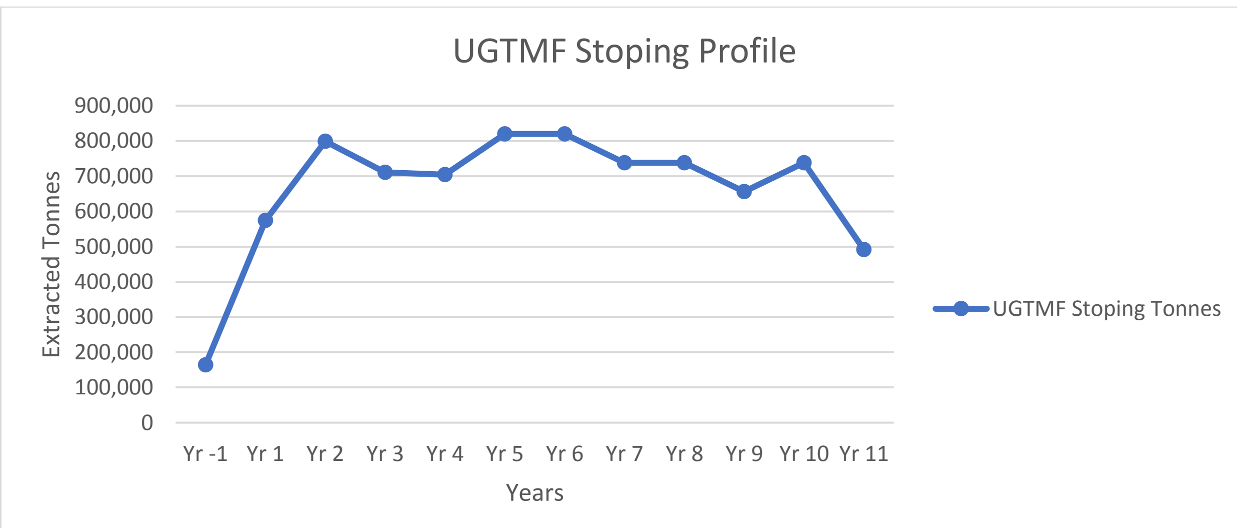
The UGMTF backfill plug needs to be cured sufficiently to support the remainder of the backfill material; therefore, a 28-day cure time has been included in the schedule. To reduce the number of UGTMF stopes required during the project period, a high cement content plug will be used for the initial three UGTMF stopes, reducing the cure time to 7 days. Once there is a sufficient number of stopes excavated, the cure time will no longer be a critical path, and the high cement content plug will no longer be required.

For each tonne of ore processed, 0.82 m^3 of tailings will be produced, along with 0.32 m^3 of combined precipitates. A portion of the tailings produced will be deposited as paste fill into available production stopes; the remainder of the material will be deposited as backfill into the UGTMF stopes. To meet this schedule, 10 to 11 UGTMF stopes will need to be excavated per year, for a total of 97 UGTMF stopes over the LOM.

A solid waste pillar, at a minimum of 15 m, will be left between all UGMTF stopes; therefore, the sequence of the UGTMF stope excavation is not critical.

A profile of the UGTMF tonnes is presented in Figure 16-21.

Figure 16-21: UGTMF Stoping Profile



16.8 Mine Services

16.8.1 Ventilation

The UG mine ventilation system is designed to be a “push-pull” system, with the exhaust fans being the main fans. The main fans will be installed on surface at the Exhaust Shaft, and the intake fans at the Production Shaft will require a ventilation plenum.

The intake fans will pull air through the natural gas heater and then push the air through the ventilation plenum. Once it has reached the plenum, some of the air will be upcasted into the headframe to prevent freezing (when required), and the rest of the air will be downcast through the Production Shaft.

The exhaust fans will maintain the pressure to ventilate the mine and keep the mine under negative pressure. The intake ventilation fans will have a maximum design capacity of 460 m^3/s (based on the maximum shaft velocity), and the exhaust fans will have a maximum design capacity of 440 m^3/s (based on an optimal exhaust raise velocity). This ventilation capacity is based on the requirement to provide the airflow requirement for radiation control, diesel particulate matter (PM) dilution, dust control, and to maintain adequate air changes through all active areas.

Single pass ventilation will be employed throughout the mine in ore headings, stopes, and where contact with ore or tailings will be encountered, including dewatering sumps. Air will be reused in non-production areas, such as waste development headings. Ventilation on Demand (VOD) will be used to monitor and control the ventilation system to ensure that adequate air quality is maintained on all working levels.

Ventilation Assumptions and Design Criteria

The ventilation system must maintain safe operating conditions UG and abide by applicable legislative and licensing requirements. The following assumptions were considered during the design process. Modelling of the ventilation system was completed using VentSim software.

- Fresh air in the Production Shaft will be split and isolated using a solid brattice; the air flowing through the skip compartments will be used to ventilate the UG material handling system, and the air flowing through the cage compartments will be used to ventilate the mine.
- Tier 4 diesel mobile equipment will be used for UG operations with applicable airflow rates specified by CANMET.
- Airflow requirements will be dependent on maintaining air quality below threshold limit values for contaminants and heat (according to American Conference of Governmental Industrial Hygienists [ACGIH] standards) in areas without diesel mobile equipment or running electric mobile equipment.
- Airflow requirements include a consideration for radiation contamination management, based on input from the Arcadis’ Canada.

- A 15% leakage factor was included in the ventilation total flow requirement. This is to ensure the ventilation system has proper control, and to ensure it is consistent with the system designed herein with VOD controls.
- Single pass air will be provided for any airflow exposed to a radiation source. For waste development, reuse of air is assumed as long as the CANMET engine rating requirements are met, and radiological limits are not exceeded.
- Airflow demand will be based on the required number of activities and the number of active levels, factoring in VOD.
- Main fans will be installed on the surface and equipped with variable frequency drives (VFDs), to regulate the speed of the fans according to the airflow required.
- Second egresses must be located in fresh air where possible.
- Fuel bays will ventilate directly to an exhaust airway to minimize impact to operations in the event of a fire.
- Water drainage is included in the exhaust fan design, as well as a mist eliminator at the exhaust outlet to collect any condensation from the exhaust air stream. The condensation collected will be pumped to the settling pond.
- Stench injection will be incorporated in the design of the fresh air system where the intake fans are located.

16.8.2 Airflow Requirements

The airflow requirements for the mine ventilation system are based on the dilution of diesel contaminants and radon concentration dilution, and ensuring minimum airway velocities are maintained to provide adequate air exchanges in working areas, and in areas with high dust (to optimize dust entrapment).

A summary of the airflow requirements is provided in Table 16-29. Table 16-29 is presented according to the primary ventilation stages expected throughout the LOM. Stage 1 refers to off-shaft development (before the main exhaust is in operation), Stage 2 refers to the pre-production period, and Stage 3 refers to full-production. The ventilation stages are closely aligned with the three development phases listed in Section 16.7.1.

Table 16-29: Airflow Requirements by Stage

	Airflow Requirements (m^3/s)		
Working Area	**Stage 1**	**Stage 2**	**Stage 3**
Active Heading (Mucking / Longhole Drilling / Idle)	–	50	120
Active Heading (Shotcreting / Jumbo)	–	60	80
Active Ore Pass	–	10	20
Rock Breaker	–	20	30
UGTMF	–	10	30
Fuel Bay and Primary Sump	–	–	15
Workshop	–	–	30
Shaft Bottom	–	–	10

	Airflow Requirements (m^3/s)		
Working Area	**Stage 1**	**Stage 2**	**Stage 3**
Skip Compartment	–	30	30
Development Heading (re-use air for Stage 2-3)	40	40	20
Leakage (15%)	6	21	52
Contingency (30% for Stage 1 and 2)	12	54	–
Density Adjustment	4	22	20
Total Airflow Required	**62**	**277**	**436**

A contingency of 30% was included in the ventilation design for Stages 1 and 2 as there will be more development during these two stages, and construction crews will be more active within the mine. The miscellaneous equipment requirements may not be fully accounted for in the airflow calculation.

16.8.3 System Description

The ventilation design is primarily a "pull" system, with a smaller size "push" component to get the ventilation air across the heaters and into the ventilation plenum at the Production Shaft. The "pull" aspect of the ventilation system will be comprised of exhaust fans on surface at the Exhaust Shaft. This will provide the negative pressure required to ventilate the UG mine. The intake fans will push fresh air through the ventilation plenum and into the Production Shaft to avoid pulling all ventilation air through the headframe. In the winter months, intake air will be heated to 3 °C using natural gas burners before it is introduced UG. This will prevent the freezing of services (e.g., piping, material handling systems, conveyors) and the buildup of ice.

The fresh air flow in the Production Shaft will be split between the cage compartments and the skip compartments. A solid brattice will ensure the fresh air is not contaminated between the two compartments.

Fresh air from the cage compartment will be distributed through the ramp system and internal fresh air raises (FARs), which will also be equipped with escapeways for second egress. Internal exhaust raises will be excavated at the end of each mining level, to which the production headings will be directly exhausted. A regulator (louver or door style for raise bottoms which might need to be accessed for mucking) will be installed within a bulkhead at the exhaust raise access. This will provide additional ventilation controls on the level, as required.

Air quality monitoring, airflow, temperature, humidity, CO, and radon will be measured on the level to ensure adequate air quality.

The internal exhaust raises will tie-in to the dedicated exhaust drift on the 500 Level, which will connect to the Exhaust Shaft. This exhaust drift will have tie-ins from the exhaust raises dedicated to the ore and waste handling system, workshops, and primary sump on the 500 Level. Airlocks will be installed in working areas to avoid contamination between the fresh air and the exhaust air.

16.8.4 Main Surface Fans and Heater

The main fan motor power requirements were estimated using the ventilation model and system capabilities that are summarized in Table 16-30.

Table 16-30: Total Main Fan Ventilation Requirements

Location	Number of Fans	VFD Capable (Y/N)	Peak Airflow (m^3/s)	Peak Pressure (at Collar) (Pa)	Estimated Power (kW)
Surface, Intake	2	Y	460	230	2 × 260
Surface, Exhaust	2	Y	440	2,720	2 × 800

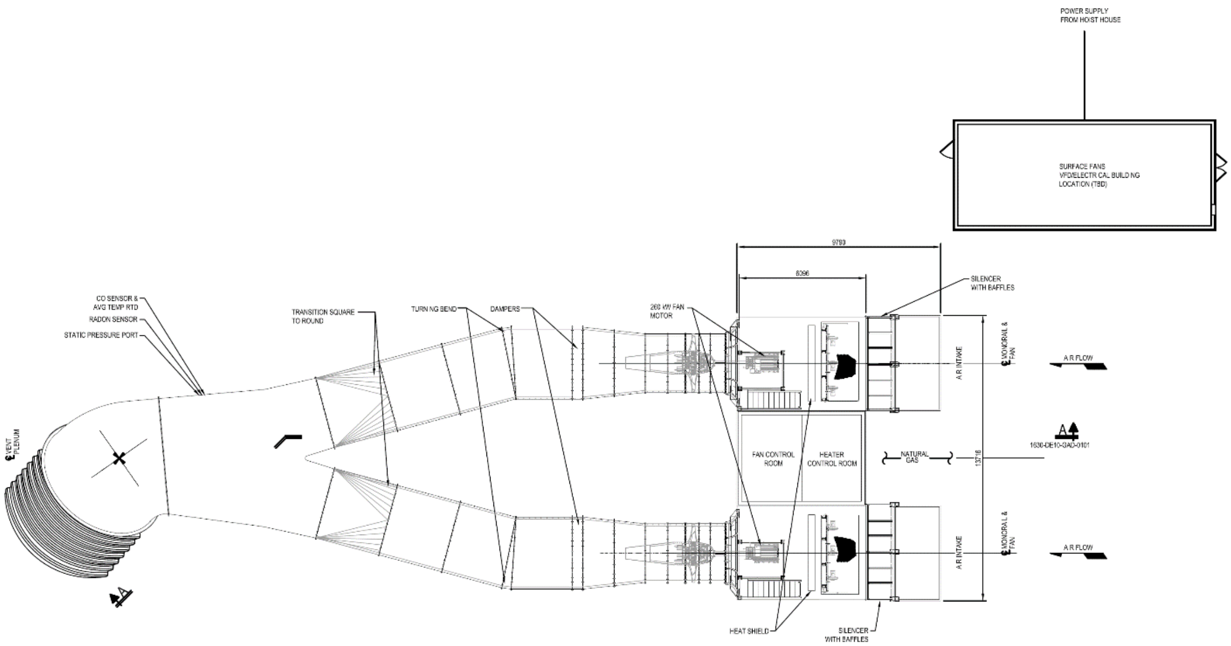
The mine air heaters will be sized to accommodate a maximum temperature rise of 45 °C (to a set point of 3 °C) based on the intake airflows specified in Table 16-31. Table 16-31 presents the estimated natural gas consumption for ventilation heating during the winter months. Figure 16-22 presents the schematic for the intake fans and heater. The surface fresh air fans will be installed in a horizontal arrangement with the natural gas direct fired heating system for use in the winter months. The warm air will be pushed through the ventilation plenum and into the Production Shaft.

Table 16-31: Peak Ventilation Heating Natural Gas Consumption Estimates

Month	Avg. Temp. (°C)	Heat Required (GJ/month)	Natural Gas Consumption ($m^3 \times 10^3$)
January	-30.4	55,153	1,500
February	-26.4	43,783	1,200
March	-18.7	35,774	980
April	-8.2	17,940	490
October	-5.1	13,409	370
November	-19.2	35,498	970
December	-27.3	50,007	1,370
Total	**–**	**251,564**	**6,870**

Based on common energy unit of 1 m^3 of natural gas = 36,625 kJ; values rounded.

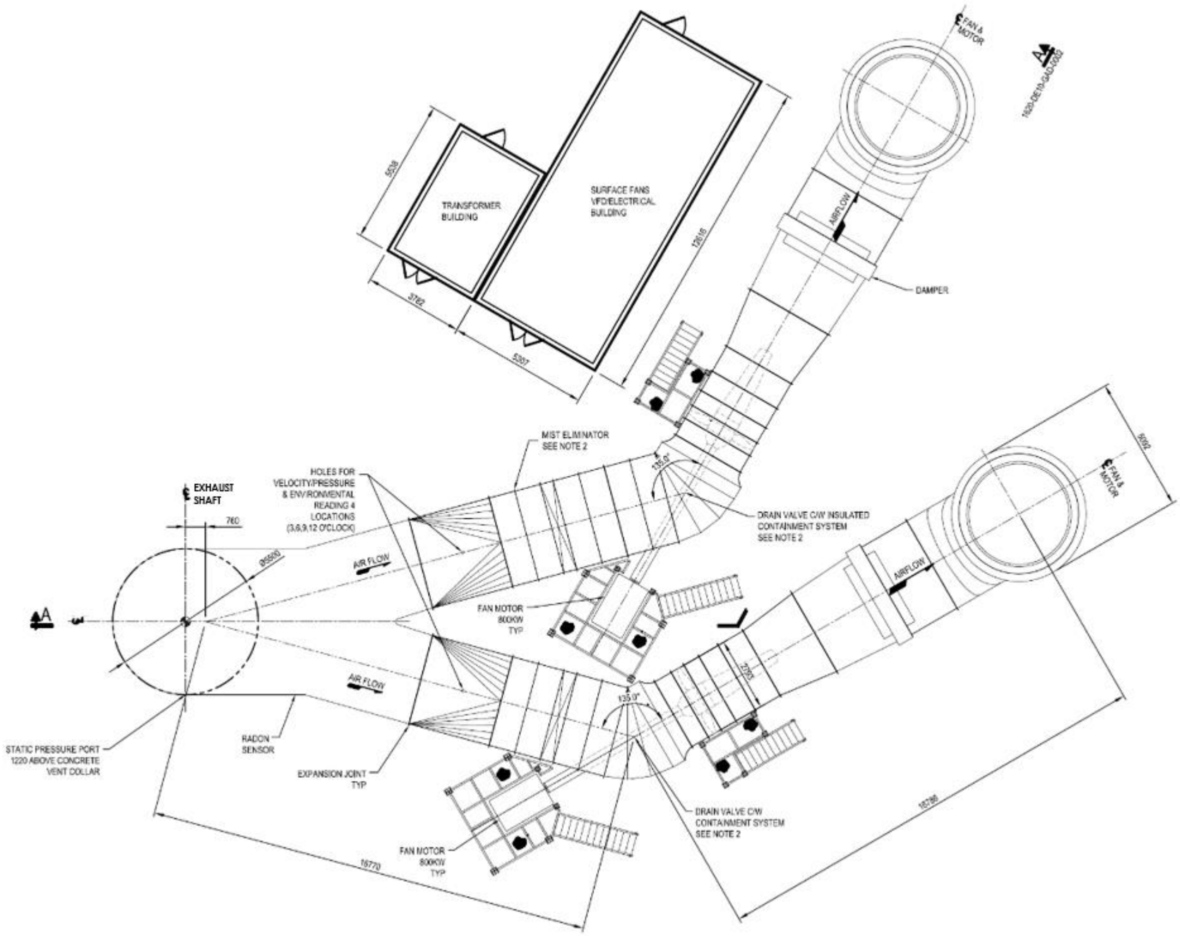
Figure 16-22: Intake Fans and Heater General Arrangement



Exhaust Fans

The exhaust air fans will be installed on the surface and will be arranged horizontally, with the exhaust outlet discharging upwards to provide better dispersion and to prevent the outlet from freezing. This will include a water drainage catchment and a mist eliminator at the exhaust outlet to capture moisture from the exhaust. This water will then be pumped to the settling pond and treated in the effluent treatment plant. Figure 16-23 presents the schematic for the exhaust fans.

Figure 16-23: Exhaust Shaft Fan General Arrangement



16.8.5 Auxiliary Fans

Development Headings

Temporary hoisting will be via the Exhaust Shaft while the mine is being developed and the Production Shaft is being equipped. Prior to the Exhaust Shaft and Production Shaft being connected, the airflow for development from the Exhaust Shaft will be via a 150 kW surface fan pushing 31 m^3/s through a 1.37 m diameter steel duct (or equivalent) from surface to the 500 Level, with duct connections to the development headings.

Two crews will be operating at this time, and 1.22 m diameter flexible ducting for each development heading will be tied into the shaft ducting through a wye connection, with a 56-kW fan installed at the connection to each duct.

The remaining development headings for the LOM will be similarly ventilated using 1.22 m diameter flexible duct with 56 kW fans in series at 500 m intervals. The 56-kW fan will be capable of pushing 20 m^3/s, which is double the requirement for an LHD, per

CANMET engine requirements, to also dissipate the heat and ensure adequate temperatures in the headings.

Production and UGTMF Headings

Ventilation for the development of the production and UGTMF headings will use an exhaust-overlap system to exhaust the radon from the headings directly to the exhaust raise. A schematic of an exhaust-overlap ventilation system is provided in Figure 16-24.

Figure 16-24: Typical Production Heading Ventilation

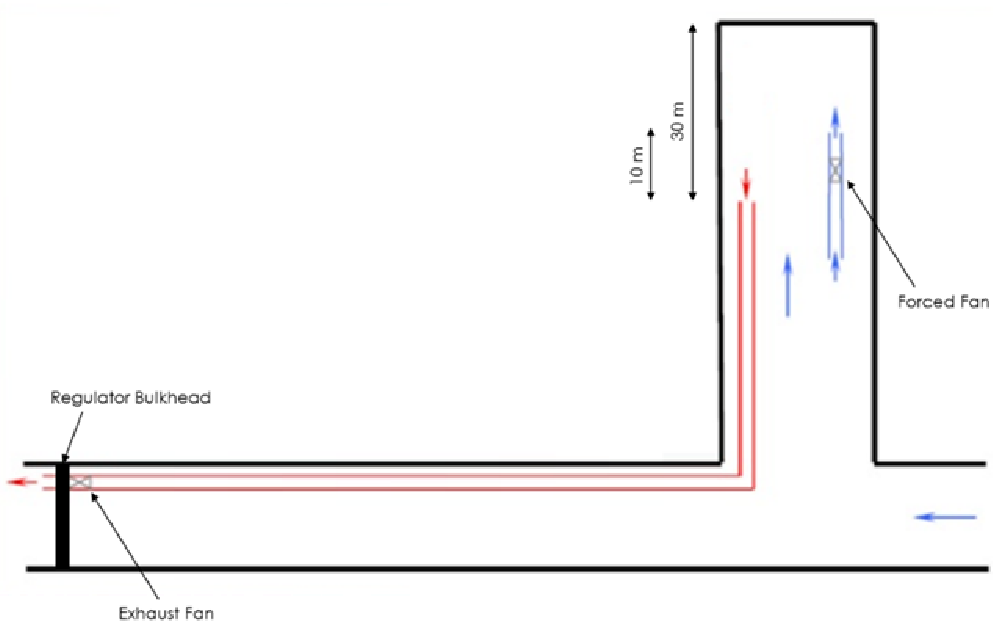


Figure 16-24 includes lengths for the minimum distance the exhaust duct should be from the face (30 m), and the minimum overlap length (10 m) for this ventilation system to be effective.

Prior to stope headings being shotcreted, the radon concentrations may be high; these headings will therefore be ventilated at 20 m^3/s. For the production headings with shotcrete (and the UGTMF headings), the ventilation volume will be 10 m^3/s. These values were confirmed to be adequate for dilution of the radon concentrations through the radiation modelling conducted by Arcadis Canada.

This ventilation volume can be achieved using two-speed fans, with the higher speed providing the higher ventilation rates (20 m^3/s) and the lower speed the lower ventilation rates (10 m^3/s). For the exhaust, a rigid duct will be required to maintain negative pressure within the mine. A PVC duct is recommended due to the low friction factor and leakage.

The same 56 kW fan used for development—but with two-speed capability—will be used to ventilate the production headings directly to the exhaust raise.

Auxiliary Ventilation Fans

The auxiliary systems have been sized to provide sufficient airflow, taking into consideration mobile equipment use, minimum velocities, and airflow to dilute radon concentrations.

Table 16-32 lists the auxiliary fan requirement for auxiliary ventilation, excluding fans listed in the UG facilities.

Table 16-32: Auxiliary Fan Ventilation Requirements

Location	Fan Qty	Flow per Fan (m^3/s)	Fan Pressure (Pa)	Duct Type	Duct Diameter (m)	Fan Diameter (m)	Estimated Power Rating per Fan (kW)
Off-shaft Development Shaft Fan	1	31	3,000	Rigid	1.37	1.37	150
500 Level Airlock	2	32	860	Rigid	1.22	1.22	56
Shaft Bottom Airlock	1	14	1,100	Rigid	0.76	0.86	22
Rock Breaker Fans	2	10	600	Rigid	0.61	0.81	11
Ore Pass and Waste Pass Fans	6	20	380	Rigid	0.81	1.14	15
Development Headings	6	20	2,100	Flexible	1.22	1.22	56
UGTMF Headings	3	10	1,700	Flexible	1.22	1.22	56
Production Headings – Exhaust	12	20	2,100	Rigid	1.22	1.22	56
Production Headings – Forced	12	5	260	Flexible	0.61	0.76	4

16.8.6 Ventilation Controls

General

Ventilation controls will be used to control airflow throughout the mine to optimize the ventilation system performance, and to reduce health and safety risks from recirculation and air short-circuiting. These will include airlocks, louvers, and overhead door regulators.

Dust Control

In airways with personnel and mobile equipment travel, air velocities will be maintained at approximately 6 m/s to mitigate dust entrainment. This lower velocity will be maintained using the FARs to transfer air, rather than limiting air transfer to only the primary ramps.

Dust will be controlled at ore and waste handling facilities (e.g., at the rock breakers, passes, and conveyors) by maintaining adequate air velocities in these areas. A dedicated exhaust raise will be developed for the ore and waste handling locations; fans will exhaust from any operating passes to prevent dust spreading throughout the mine. At the passes, plugs or covers will be required to ensure air does not short circuit through the passes.

Ventilation on Demand

VOD will be implemented for the Project to ensure adequate air quality UG and to avoid over-ventilation. The VOD system will employ a series of sensors distributed throughout the mine that send real-time information regarding the air quality, the location of equipment, and the location of personnel, to a central computer with specialized software. The software—with input from personnel in charge of the ventilation system—will determine the ventilation requirements to the level or to the development / production heading.

16.8.7 Mine Dewatering

Mine Dewatering System

The UG dewatering facilities are designed to receive water from three sources: ground water seepage, flush water used to clean paste backfill lines, and mine operations.

Ground water seepage will contribute approximately 100 m^3/hour of water by the end of mine life as more openings are developed. Backfill line flushes will contribute under 3 m^3/hour of water, on average. Mine operations—which include drilling, bolting, and shotcreting operations—will produce an average of 45 m^3/hour of water. Infiltration from these three sources will be directed to different areas of the UG mine workings, based on the source of the seepage, for a total combined average rate of 148 m^3/hr once the mine is fully developed.

The dewatering system will have a cascading design including borehole sumps, level sumps, and a main sump / pump station located on the 500 Level. The main dewatering station will be a clean water system that uses two filter sumps, an intermediate sump, and a clean water sump. Figure 16-25 presents a schematic process flow diagram that illustrates the various water flow streams. Table 16-33 summarizes the movement of UG water.

Figure 16-25: Schematic Dewatering PFD

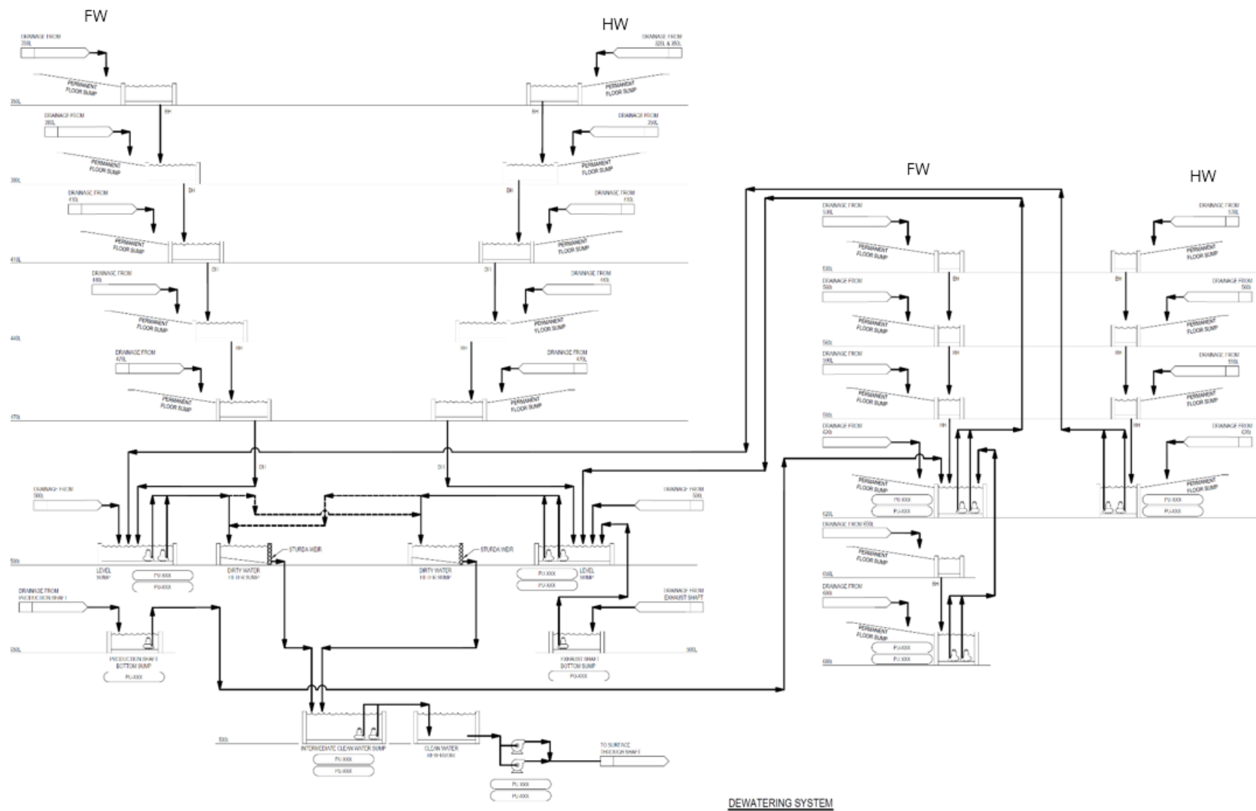


Table 16-33: Movement of Underground Water

Destination	Drain by Borehole - Sump Location	Pumped from -Sump Location
500 Level Sump – FW	350 Level, 380 Level, 410 Level, 440 Level, 470 Level, FW	620 Level, FW
500 Level Sump – HW	350 Level, 380 Level, 410 Level, 440 Level, 470 Level, HW	620 Level, HW, Exhaust Shaft bottom sump
620 Level Sump FW	530 Level, 560 Level, 590 Level, FW	650 and 680 Level FW, Production Shaft bottom
620 Level Sump – HW	530 Level, 560 Level, 590 Level, HW	–
680 Level Sump – FW	650 Level FW	–

A typical level sump will consist of either one or more submersible pumps located at the end of a walkway and staircase, with a draining borehole located at the front (shallow end) of the sump.

Radon gas accumulation will be a risk at all locations where groundwater collects. Air will continually be drawn out of all sump excavations on a continuous basis and direct any radon gas to the exhaust raise.

Sump Locations and Roles in the Mine Dewatering System

Borehole sumps will be located on the FW and HW sides of the upper mine on the 350, 380, 410, 440, and 470 Levels. Water collected on these levels will drain through a series of boreholes to the two sumps located on the 500 Level.

Borehole sumps will also be located on the FW and HW sides of the lower mine on the 530, 560, and 590 Levels. Water collected on these levels will decant through a series of boreholes to the two sumps located on the 620 Level. From these sumps, the water will be pumped to the two sumps located on the 500 Level.

A single borehole sump will be located on the FW side of the resource on the 650 Level. Water collected on this level will decant through a borehole to the level sump located on the 680 Level. From here, the water will be pumped to the FW level sump located on the 620 Level. Water collected in a sump at the Production Shaft bottom (650 Level) will also be pumped to the 620 Level FW sump.

Water collected in the two sumps on the 500 Level will be pumped alternately into one of two dirty water sumps located on the 500 Level. The dirty water sumps will sit on a base graded at +3% slope to allow water to drain away through a porous filter membrane (Sturda weir).

Based on an estimate of 2% solids by weight, the proposed weir will take in excess of 15 calendar days to fill completely with solids. Allowing three days for the remaining water to drain from the captured solids once the Sturda weir has been opened, each dirty water sump will operate for ±12 days before mining operations need to drain and muck out the sump.

Screened water from the dirty water sump will be collected in an intermediate sump, which is located between the two dirty water sumps. The intermediate sump can hold 320 m^3 of water. Based on the average mine water inflow rate of 148 m^3/hour, this sump will offer a retention time of approximately two hours. Submersible pumps located in the intermediate sump will pump water to the clean water sump located on the same level.

The clean water sump occupies a cut-out using a concrete dam to provide a temporary storage location for the clean water. Level monitoring will allow for a minimum static head of water to be maintained when feeding the dewatering pumps. There will be two horizontal centrifugal pumps in the pump station, one operating and one spare.

16.8.8 Electrical

Underground Electrical Sources and Facilities

Power to the UG facilities will be sourced from the LNG power plant on surface. From the 13.8 kV E-House, power will be distributed to two main switchgear lineups located at the Production Shaft hoist building electrical room and the mill electrical room.

From the Production Shaft hoist building switchgear, two feeders will be routed via the buried utilidor to the collar area in the Production Shaft headframe, and terminated in high voltage (HV) junction boxes to transverse from Teck cable to vertical self-supporting shaft cable (Airguard™).

From the Production Shaft headframe, the two shaft cables will be installed on separate bracket systems located in different areas of the shaft to minimize the potential of both being damaged at the same time. Both cables have the capacity for the total UG load in case either one is damaged. These cables will be installed to the 500 Level and terminate in the shaft station electrical room. From there, the cables will return to the shaft and run to the 590 Level, terminating in the 500 Level shaft station electrical room.

From the shaft station electrical rooms on 500 Level and 590 Level, power will be distributed through numerous electrical rooms on a level-to-level basis. The level-to-level routing will be completed via the FARs (escapeways) to reduce additional boreholes.

Major Underground Equipment Loads

The major UG power loads include development and production equipment (e.g., drills and bolters), ventilation fans (e.g., fresh and exhaust air circuits), dewatering pumps, material handling equipment (e.g., conveyors and rock breakers), and UG infrastructure facilities (e.g., maintenance facility and refuge stations).

Primary Electrical Rooms (Shaft Station)

The primary electrical rooms located at the 500 Level and 590 Level shaft stations are major facilities with 13.8 kV switchgear lineups and a main-tie-main configuration. These facilities also include 13.8 kV-600 V transformers (750 kVA for 500 Level and 1500 kVA for 590 Level), and all required 600 V distribution equipment (i.e., circuit breakers and starters) for local equipment.

One key feature of the 590 Level primary electrical room is a circuit that back feeds power to the 500 Level. This circuit provides a secondary feed to the mine's main dewatering station on the 500 Level.

Primary Electrical Rooms (Level)

Primary electrical rooms on each level are located near the FAR; these are considered the power backbone for the mine. These rooms have a small 15 kV switchgear lineup to distribute power to the next level and to secondary electrical room(s) on the level; a 13.8 kV/600 V transformer for local equipment loads; and a spare 15 kV fused disconnect switch will be used to power a mine power centre for production crews, when required.

Secondary Electrical Rooms

Secondary electrical rooms are power extensions for the 13.8 kV system on the majority of levels. These rooms contain a three-section 15 kV switchgear lineup, a small (300 kVA) 13.8 kV/600 V transformer, and a spare 15 kV fused disconnect switch used to power a mine power centre for production crews when required.

On the 500 Level, an additional secondary electrical room is required to support the maintenance facility. This room has a main-tie-main configuration to have incoming power from primary electrical rooms. The intention for this dual incoming feed is to ensure the main dewatering station has redundant power feeds.

Electrical Room – Main Dewatering Station

The 500 Level has the main dewatering station located on the south side of the main drift between the Production Shaft and Exhaust Shaft. This station has a dedicated electrical room that provides power to the main dewatering pumps and submersible pumps from the intermediate clean water sump. This electrical room has 13.8 kV power.

This electrical room has a 13.8 kV/4,160 V transformer, a 5 kV switchgear line up, two 4,160 V VFDs for the 500 hp pumps (one duty and one spare), a 4,160 V/600 V transformer, and a 600 V splitter rack to provide power to the submersible pumps and other small equipment.

Electrical Room – 620 Level Loading Pocket and Conveyor Drift

The 620 Level includes the loading pocket, loadout conveyor, and three feeder stations (one for ore and two for waste). This electrical room has a 600 V power distribution panel providing power to key equipment, such as the loading pocket hydraulic power unit (HPU), conveyor, feeder station for waste pass #1 (waste pass #2 and the ore pass stations are powered from the main electrical room near the shaft), and the shaft bottom sump pump.

Underground Grounding System

The UG grounding system will originate from the surface ground grid and be routed to the UG workings via two 4/0 bare copper ground wires installed onto separate bracket systems in the shaft. At the 500, 590, and 620 Levels, these grounding conductors will branch off from the shaft and be routed via the drift messenger systems to all levels, facilities, ramps, and escapeways (in the FARs).

Within each facility, a ground bus will be installed to provide a common connection point between the two points from the grounding system and facility equipment. All equipment will be appropriately grounded based on Canadian Electrical Code (CEC) requirements such that a ground loop is complete within the facility.

16.8.9 Communications and Instrumentation

The Rook I site will be employing proven technologies for the UG communication and automation systems. The following networks and systems have been included as part of this FS.

- Fibre optic network
- Programmable logic controller (PLC) network
- Fire alarm network
- Shaft signal system
- Leaky feeder radio system
- Vehicle and personnel tracking system
- Analog telephone system

16.8.10 Underground Fibre Optic Backbone

The site-wide fibre network covers all surface and UG communications and automation facilities. The main site control rooms located at the Production Shaft hoist building, mill control room, and the communications building are the main hubs on surface.

The UG communication and automation network will originate from the Production Shaft hoist building, and will be routed via two 96 core single mode fibre optic cables through the buried utilidor to the Production Shaft headframe. The network will then be routed down the Production Shaft to the 500 Level and 590 Level shaft station electrical rooms, where the cables are terminated in local network cabinets.

Considerations have been included in the design to provide a ring topology UG to allow for a self-healing network in the event a cable is damaged, or hardware fails.

Communication through the fibre optic network will support voice over internet protocol (VoIP) telephones located in all electrical rooms, refuge stations, maintenance facility offices, and other facilities as deemed necessary.

The fibre network will be designed to promote tele-remote operation from the surface control room for systems such as rock breakers and mobile equipment (e.g., LHDs, short run haul trucks).

PLC Network

In the UG mine, PLCs will be strategically located to provide monitoring and control of processes and facilities such as ventilation, dewatering, process water, and material handling.

Fire Alarm Network

The fire alarm network UG will be a critical component of the mine as it will provide instantaneous notification when a fire occurs. For the purposes of this FS, fire alarm panels (FAPs) have been planned for the following locations.

- The 500 Level to trigger fire alarms in the maintenance facility, fuel bay, rock breaker station, and dewatering station.
- The 590 Level to trigger fire alarms in the rock breaker stations.
- The 620 Level to trigger fire alarms in the loading pocket and along the loadout conveyor.

Shaft Signal System

The shaft signal system will be a communication system between personnel UG and the hoist operator. A sound device will be activated through cable pulls to allow communication from level to surface or level to level.

This system will be integrated into the hoist operator control room, with junction boxes and control panels in the Production Shaft headframe at the sheave decks, skip dump, and collar area. A shaft signal cable will be installed in the shaft to the 500, 590, and 620 Levels shaft stations where junction boxes and control panels will be located.

Leaky Feeder System

The coaxial leaky feeder system will run from the headframe down the shaft to the 500, 590, and 620 Levels via two-way splitters and line amplifiers. There will be a leaky feeder link between the headframe and hoist house. The leaky feeder system will be routed UG through all level drifts, ramps, facilities, and escapeways (in FARs) to provide radio-based communication throughout the mine.

Underground Vehicle and Personnel Tracking System

Radio frequency interference (RFI) modems will allow the mine to locate personnel, vehicles, and other mobile equipment. Tag readers will be powered from the leaky feeder cable. Each reader will be equipped with a radio modem, providing the radio frequency (RF) link back to the control room.

The system will be designed to increase safety, enabling safety officers to obtain real-time information on the numbers and whereabouts of personnel and vehicles UG.

Analog Telephone System

The analog telephone system will run from the Production Shaft hoist building, through the buried utilidor to the Production Shaft headframe, and down the shaft to telephone junction boxes located in the 500 Level and 590 Level shaft station electrical rooms.

From the 500 Level, the system will be routed along drifts and ramps to the following facilities.

- 500 Level maintenance facility office
- 500 Level refuge station
- 500 Level electrical rooms
- 410 Level refuge station

From the 590 Level, the system will be routed through the shaft and along drifts and ramps to the following facilities.

- 590 Level refuge station
- 590 Level electrical rooms
- 620 Level electrical rooms
- 680 Level refuge station

16.8.11 Water Supply

Fresh Water

Fresh water will be required for use in development and production drilling, dust control, washing floors and equipment, mixing shotcrete and concrete, fire protection, and other miscellaneous processes. Fresh water, combine with recycled water from the effluent treatment plan, will be distributed along the 500, 590, and 620 Levels from a surface water tank. The water will be delivered via the Production Shaft through a 150 mm diameter line.

Pressure reducing valves (PRVs) will be used to reduce water line head pressures on pipelines to the 500, 590, and 620 Levels. Upper levels of the mine will be fed from a high-pressure line that bypasses the 500 Level PRV station where the water pressure will be reduced locally.

Fire Water

UG fire-related services are required to meet National Fire Protection Association (NFPA) requirements. Fire water must be available UG for UG fire suppression, hose reels, and sprinkler systems. Fire water or foam systems will also be used in the main workshop areas, satellite workshops, and main fuel and lube bays. The fresh water storage tank located near the headframe will be sized to meet the delivery requirements for the UG fire sprinkler system.

Fire detection and suppression systems will interface with the emergency alarm system and will be included in areas at high risk of fire. These areas include the entire length of the conveyors (both above and below the top of the belt), main workshops (i.e., crane bays, service bays, welding bays, offices, and lube storage bays), main fuel and lubricant storage and distribution areas, and satellite workshops. Many of the individual operating systems will be equipped with self-contained fire suppression systems that are included by the original equipment supplier.

Fire water hose reels with 30 m long hoses will be located at 30 m intervals along the length of the conveyor.

Fuel and lube bays with self-contained units will be equipped with integral fire-suppression and will not require fire water.

Electrical mine load centers and sub-stations will require clean agent fire suppression systems, such as an FM-200 fire suppression system.

Potable Water

Treated potable water will be delivered via large plastic bottles. Potable water will be provided in UG workshops, permanent refuge stations, and bottle filling stations.

Fresh water will be piped to latrines for washing. The latrines will be equipped with sterilizing ultraviolet lights.

Personnel must fill appropriate water containers on surface or at a designated bottle filling station, and carry the water supply to work areas.

16.9 Underground Mine Facilities

16.9.1 Material Handling

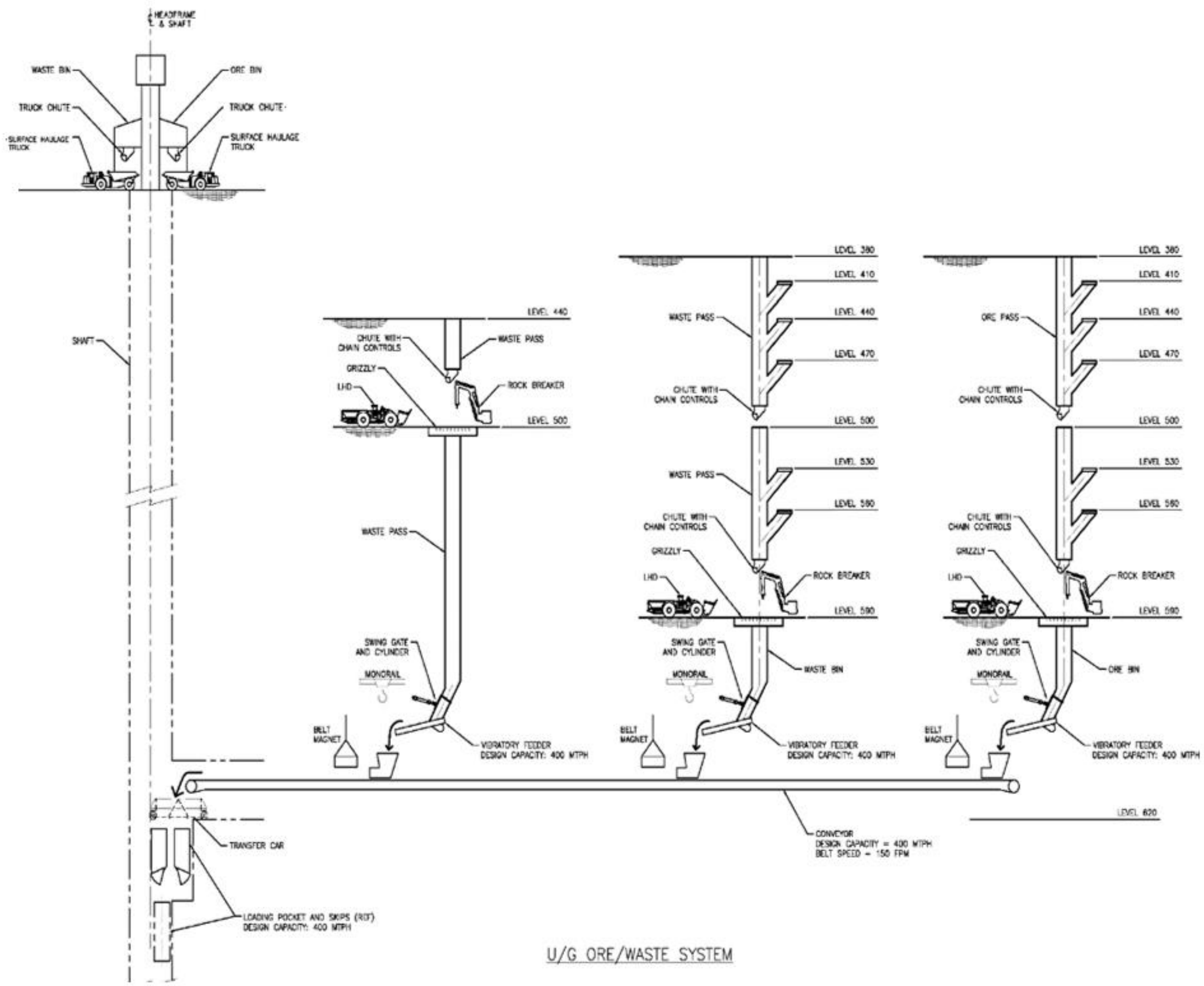
Each level above the 590 Level will have a waste pass and ore pass located near the resource on the FW side. A RAR will be connected to the passes to keep dust and other contaminants out of the fresh air circuit. Remucks for marginal ore, special waste, and waste will be located near the pass to provide capacity for batching.

Drilling and blasting from mining operations will produce a continual stream of broken rock. The rock will be mucked from the development headings and stopes to the nearest pass or remuck using a 17-tonne capacity LHD. All rock will be analyzed using scanners, which will measure the radioactivity of the load in the LHD bucket. Depending on the scan results, the load will be defined as ore, marginal ore, special waste, or waste. It will then be transferred to the nearest pass or remuck adjacent to the pass.

The ore and waste handling system will consist of three rock breakers: one on the 500 Level near the UGTMF, and two on the 590 Level dedicated to waste development and ore development and production. The rock breakers will be fed either directly using the LHD or via raises. Below the grizzlies, the rock will transfer through a raise and bins onto the 620 Level loadout conveyor.

A schematic of the ore and waste handling system is presented in Figure 16-26.

Figure 16-26: Schematic of Ore and Waste Handling System



Rock Breaker Facilities

The rock breaker facility dedicated to the UGTMF will be located on the 500 Level. The facility will be fed directly from an LHD or pass, with a dump on the 440 Level. On the 500 Level, material will discharge from the pass-through control chains and a hydraulic press frame onto a grizzly measuring 3.6 m × 3.6 m.

The sunken grizzly design will have a storage capacity of two LHD loads, and it will be protected by wear liners that are 75 mm thick. The sunken grizzly will have 400 mm × 450 mm square openings.

A hydraulic rock breaker will be equipped with two separate boom arms: one that will be used to manipulate a picking claw and a metal shear, and a second arm that will help break down oversized material so that it may pass through the grizzly. The hydraulic rock breaker will be located adjacent to the grizzly and perpendicular to the LHD dumping point. Undersized material will fall into a bin that extends to the 620 Level, where a swing gate will meter out material onto the vibratory belt feeder.

The HPU will be equipped with a self-contained fire protection system. The rock breaking facility will be equipped with closed-circuit television and automated controls so the rock breaker operators can conduct work from a separate area of the mine or the surface, and eliminate exposure to breaker noise and radon gas, in keeping with As Low as Reasonably Achievable (ALARA) safe work practices.

The other waste pass system and the ore pass system will have rock breakers located on the 590 Level. These rock breakers will be fed by passes connected to dump points on the 380, 410, 440, 470, 500, 530, and 560 Levels.

For both the ore and waste pass, material will discharge through a dedicated press frame gate, complete with chain controls, onto a grizzly measuring 3.6 m × 3.6 m. These rock breaking facilities will be identical to the facility servicing the UGTMF.

Undersized material from these facilities will fall into the bins that extend to the 620 Level, where a swing gate will meter out material onto the respective ore pass or waste pass vibratory feeder. Each vibratory feeder will be equipped an HPU with a self-contained fire protection system.

Vibratory Feeders and Conveyor

Hydraulic gates at the discharge of the bins will facilitate isolation of material flow for safe operations and maintenance activities. Once the gates have been opened, material will tumble onto the deck of the vibratory feeder until the flow of material is stopped by the natural angle of repose of the feed material.

Rock from each pass will be fed onto a single 1,500 mm wide conveyor belt that transports the material to a transfer car. The width of the belt has been selected based on its ability to handle the largest rock that may pass through the grizzly opening.

The conveyor belt, like the vibratory feeders, will operate at less than 40% of the design capacity based on volumetric handling capability. The conveyor belt will be equipped with low-friction impact mats at each feeder discharge that will assist to orient falling sharp material in the direction of conveyor movement, and to minimize the chances of puncturing the conveyor belt. Self-cleaning belt magnets will be used to capture any remaining tramp metal not collected at the grizzly.

The conveyor belt will be made from anti-static material that meets Canadian Standards Association (CSA) fire-retardant requirements. The belt will be equipped with a single belt scale located beyond the final waste rock load point. The main conveyor drive will be equipped with an ammeter, and the head and tail pulleys will be equipped with zero speed switches that are interlocked with the three vibrating feeder drives.

When operating, the conveyor belt will deliver 400 t of material per hour to the transfer car at the skip loading station. The transfer car will shuttle back and forth, powered by a hydraulic cylinder, to direct material discharging from the end of the conveyor into one of two skip loading flasks. Load cells will control the filling of the flasks.

The conveyor frame will be channel stringer fabrication complete with outboard guarding for personnel protection, and heavy-duty CEMA E sealed-for-life idlers. The conveyor will be equipped with emergency stop pull cords. The conveyor will be hung by chains from the drift’s back. It will also have a manual tension release for personnel safety.

Feeder control will be interlocked with swing gate position (i.e., open or closed) to minimize the potential of contaminating feed between the ore or waste rock streams. The conveyor will be equipped with a variable-speed drive.

16.9.2 Maintenance and Fuel

Main Workshop

The main workshop will be a multi-bay facility that can service up to six units. The main workshop will include the following.

- One service bay
- Two crane bays
- One wash bay
- One welding bay
- One office
- One hose shop
- One electrical substation
- One electrical shop

- One warehouse
- One clean room
- One tire storage
- Two latrines

In each crane bay, there will be two 25-tonne cranes so that multiple vehicles may be serviced at the same time.

In the service bay, there will be a ramp with removable grating for access to the underside of mobile equipment, trench drain, sump, and an oil water separator. The largest piece of equipment expected to be serviced in the main workshop will be an UG haul truck.

Additional equipment in the crane and service bays will include the following.

- Compressed air lines (from the surface)
- Service water hose reels
- Sinks
- Safety items
- Lube station
- Waste storage bins with lids
- Tool storage
- Work benches
- Parts storage

Ventilation for the crane bays and service bay will be flow-through ventilation to a nearby exhaust drift. Roll-up fire doors will be installed at the entrance to the workshop.

The welding bay will be located near the exhaust drift and will be equipped with welding tables, acetylene and oxygen bottles in a rack, a 5-tonne monorail crane, workbenches, storage cabinets, portable welding screens, and two exhaust hoods ducted to the exhaust drift.

The electrical substation supports the shop and offices. The purpose of the electrical shop will be to provide a clean area for repairs to electrical equipment.

A clean room will be located near the office and will be equipped with a sink with a water heater, lockers, and a boot washing station. The latrine will have portable latrines and a hand washing sink with water heaters.

The office will have tables and chairs, refrigerator, counter, microwave, first aid supplies, desks, computers, phone equipment for communication with the surface, and computer access to all necessary servers. A wall-mounted fan with a duct will ventilate the office.

The warehouse located in the main workshop with have shelving along one wall for equipment storage, such as small parts and tools. Pallets will be stored along the opposite wall. The warehouse will have space for a forklift to access the pallets.

The tire storage bay will be designed to accommodate multiple tire sizes for the UG mobile equipment. The bay will be sized to allow a forklift to access the tires.

The wash bays will be located adjacent to the main workshop area for cleaning vehicles prior to maintenance. The wash bay will have a high-pressure washer with a hot water heater, soap cubes, and a high-pressure hose reel. Process water will be supplied via piping routed through the internal ramp.

A fan with silencers will ventilate this area. Sloped concrete floors in each bay will promote gravity flow of water to a sump with a submersible pump. Oily water will be pumped from the sump to an oil water separator. The main fuel and lube station, permanent refuge, parking, and other storage bays will be located near the main workshop.

Satellite Workshop

A smaller single-bay satellite workshop will be located on the 590 Level near the Production Shaft. This workshop will have a 25-tonne crane, service water and compressed-air hose reels, communications, fire roll-up doors, and fire suppression sprinklers. The purpose of this satellite workshop will be to support servicing and minor repairs for limited-travel equipment.

Service water, compressed air, and fire water will be supplied from the surface to the satellite workshop via a piping-routed through the shaft and access drifts. A fan with silencers will ventilate this area.

A wash bay, satellite fuel and lubricant bay, permanent refuge station with latrines, and parking and storage areas will be located on the same level as the satellite workshop.

Fuel and Lubricant Facilities

Fuel and lubricants will be transported in bladders via the main service cage and transported using a forklift to the fuel and lubricant stations, located on the 590 Level, as well as near the 500 Level main workshop.

The main UG fuel and lubricant bays will be centrally located on the 500 Level near the main workshop. The facility will accommodate two vehicles at a time to access diesel fuel and lubricants such as glycol, transmission fluid, hydraulic fluid, and engine oil. This will be a drive-through arrangement connected to the exhaust circuit on the 500 Level.

The main fuel and lubricant bay will have two 4,500 L Sat Stat type storage tanks, five 2,200 L lubricant Sat Stat type storage tanks, lubricant hose reels, fuel pump, trench drain with a sump, means of instrumentation and controls, fire water hose reel, means of fire detection and suppression, and safety items such as a fire extinguisher, spare safety gloves, and spill absorbent.

Fire doors will be installed at the entrance and exit to the main fuel and lubricant bay. These doors will prevent any fires from spreading, and will prevent noxious gases from reaching workers. Fire detection and suppression equipment that interfaces with the complex's emergency alarm system will be included in the fuel and lubricant storage bays and the mobile equipment.

16.9.3 Explosives and Detonators Storage Facilities

UG storage magazines for explosives, detonators, and blasting accessories will be located on the 500 Level and 590 Level, away from the UG infrastructure and work areas. There will be separate magazines for explosives and detonators.

Explosive products be used for development and production mining will be securely handled and stored in the magazines, including ANFO, emulsion, detonators, and packaged explosives.

All explosives will be stored, stacked, and labelled to facilitate a first-in/first-out inventory control system. Each magazine will be designed with a locking gate. The location of the explosive or detonator facility will be a minimum of 100 m from any work area or blasting area, and at least 25 m from the main travel way.

Explosive and detonator materials will be transported from the surface via the main service cage to the UG magazines. The containers will be unloaded using monorails, and all other materials will be unloaded using boom trucks. Special trucks operated by trained and authorized individuals will be used to transport explosive materials from the UG magazines to the workplace.

16.10 Surface Mine Facilities

Surface facilities required to support operations are presented in Section 18.0.

16.11 Mining Equipment

The Arrow Deposit will be developed using a high degree of equipment mechanization. Each of the main pieces of equipment has the capability of operating remotely, and in some cases autonomously. Operators for the equipment are included in the cost estimate, except for the LHDs in production. One operator has been assumed for every two production LHDs.

All the equipment listed will be supplied by a major mining equipment manufacturer, in new condition, and purchased by NexGen. The mine development contractor will use the equipment provided by NexGen and will return the equipment to NexGen in full working condition.

During initial construction, there will be a single development crew on the 500 Level, until the connection is made to the Production Shaft. At that point, there will be sufficient air volumes for a second crew to commence work on the 500 Level. Once the Production Shaft is complete and commissioned, development crews will start on 620 Level and 590 Level.

In total, there will be four separate development crews—with the related gear—for the development activities. Once the lower mine is connected to the upper mine, there will only be a need for three development crews; therefore, it is assumed the additional gear purchased for the early development activities will be used as spare equipment. Additional costs have not been factored for spare equipment.

During the captive development phase, each development crew will have the following pieces of equipment.

- One two-boom jumbo
- One LHD
- One bolter
- One scissor lift

Once the upper and lower mines achieve peak development rates, an extra bolter will be added to support the proposed advance rates. As development progresses further from the 500 Level Exhaust Shaft station, an UG haulage truck will be added to the fleet.

Vertical development will be a combination of Alimak and raiseboring, and will be completed by contractors providing the equipment as required. This equipment is therefore excluded from the equipment list.

There will be approximately 50 pieces of equipment operating in the mine during peak production years. During steady state operations, the use of haul trucks will be minimal, as a series of ore and waste passes will be established for the movement of material. Once production progresses above the 380 Level and below the 620 Level, haulage trucks will be required, and a second truck will be added to the fleet.

Table 16-34 lists the equipment that will be required annually.

Table 16-34: Mobile Equipment Fleet

Equipment Type	Year												
	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Production / Development													
Two-Boom Jumbo	2	4	4	3	3	3	2	1	1	1	1	1	1
Rock Bolter	3	5	5	4	4	4	3	1	1	1	1	1	1

Equipment Type	Year												
	-2	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
LHDs – Development / Production	3	7	8	8	8	8	7	6	6	6	5	5	5
Scissor Lift	2	4	4	3	3	3	2	1	1	1	1	1	1
Production Drill	0	2	3	4	4	4	4	4	4	4	3	3	3
Block Holer	0	1	1	1	1	1	1	1	1	1	1	1	1
ANFO Trucks	0	2	2	2	2	2	2	2	2	2	2	2	2
UG Haulage Trucks	0	1	1	1	1	1	1	1	2	2	2	2	2
Subtotal Production / Development	**10**	**26**	**28**	**26**	**26**	**26**	**22**	**17**	**18**	**18**	**16**	**16**	**16**
Auxiliary Equipment Fleet													
Cable Bolt Jumbos	0	1	1	2	2	2	2	2	2	2	2	2	1
LHDs – Utility	0	0	1	1	1	1	1	1	1	1	1	1	1
Fuel Delivery Truck	0	0	1	1	1	1	1	1	1	1	1	1	1
Boom Truck	0	1	1	1	1	1	1	1	1	1	1	1	1
Transmixer	0	2	3	3	3	3	3	3	3	3	3	3	2
Shotcrete Sprayer	0	1	2	2	2	2	2	2	2	2	2	2	1
Personnel Transport Vehicles	0	0	0	1	1	1	1	1	1	1	1	1	1
Scissor Lift – Construction	1	1	1	1	1	1	1	1	1	1	1	0	0
Supervisor / Engineer Trucks	1	4	5	5	5	5	5	5	5	5	5	5	3
UG Grader	0	0	1	1	1	1	1	1	1	1	1	1	1
Leaders Basket Truck with Backhoe	2	2	3	3	3	3	2	1	1	1	1	1	1
Mechanics Truck	1	1	1	1	1	2	2	2	2	2	2	2	2
UG Forklifts	0	2	2	2	2	2	2	2	2	2	2	2	2
Subtotal Auxiliary Fleet	**5**	**15**	**22**	**24**	**24**	**25**	**24**	**23**	**23**	**23**	**23**	**22**	**17**
Total UG Mobile Equipment	**15**	**41**	**50**	**50**	**50**	**51**	**46**	**40**	**41**	**41**	**39**	**38**	**33**

Equipment operating hours have been estimated based on first principles. The requirement for equipment specific to development or production will fluctuate with changes to annual throughput, while some equipment will operate at a certain rate regardless of the tonnage or metres being developed.

A mobile equipment rebuild and replacement schedule was developed based on the operating hours required and the equipment available. The total number of rebuild and replacement quantities are presented in Table 16-35.

Table 16-35: Total Number of Rebuilds and Replacements by Equipment Type

Unit	Total Number of Rebuilds	Total Number of Replacements
LHDs – Utility	1	1
UG Haulage Trucks	0	0
Boom Truck	1	0
Supervisor / Engineer Trucks	2	1
Mechanics Truck	1	1
Leaders Basket Truck with Backhoe	1	1
Personal Transport Vehicle	1	1
Transmixer	0	0
Shotcrete Sprayer	1	0
Fuel Delivery Truck	0	0
Scissor Lift	0	0
UG Grader	1	1
UG Forklift	1	1
Scissor Lift – Construction	0	0
Production Drill	1	0
Block Holer	2	2
Cable bolt Jumbo	1	1
LHDs – Development / Production	7	3
ANFO Trucks	0	0
Rock Bolter	0	0
Two-Boom Jumbo	1	0

16.12 Comments on Section 16

The mine plan will apply conventional mining techniques and conventional equipment. The proposed implementation of a UGTMF is innovative for uranium mines, as it will facilitate permanent storage of process tailings UG and active reclamation of the mine.

17.0 RECOVERY METHODS

17.1 Process Flow Sheet

A zero-based design approach was used for the mill process design. The design aims to achieve the required throughput while minimizing redundancy. Health, safety, and environmental aspects will not be compromised.

There are only two instances where circuit design capacity is planned to be greater than nominal. Grinding capacity has been increased by 20% more than nominal to allow for higher maintenance requirements. The effluent treatment plant has also been designed for a greater-than-nominal flow rate, due to the possibility of having mine inflow and weather-related surges in effluent treatment requirements that must be considered.

Process design has been directed by the metallurgical test program results, knowledge from literature, and Wood's experience with existing successful process methods.

The proposed process block diagram is shown in Figure 17-1. Table 17-1 shows the production design requirements used to develop the process flows and mass balance for the processing plant.

17.2 Plant Design

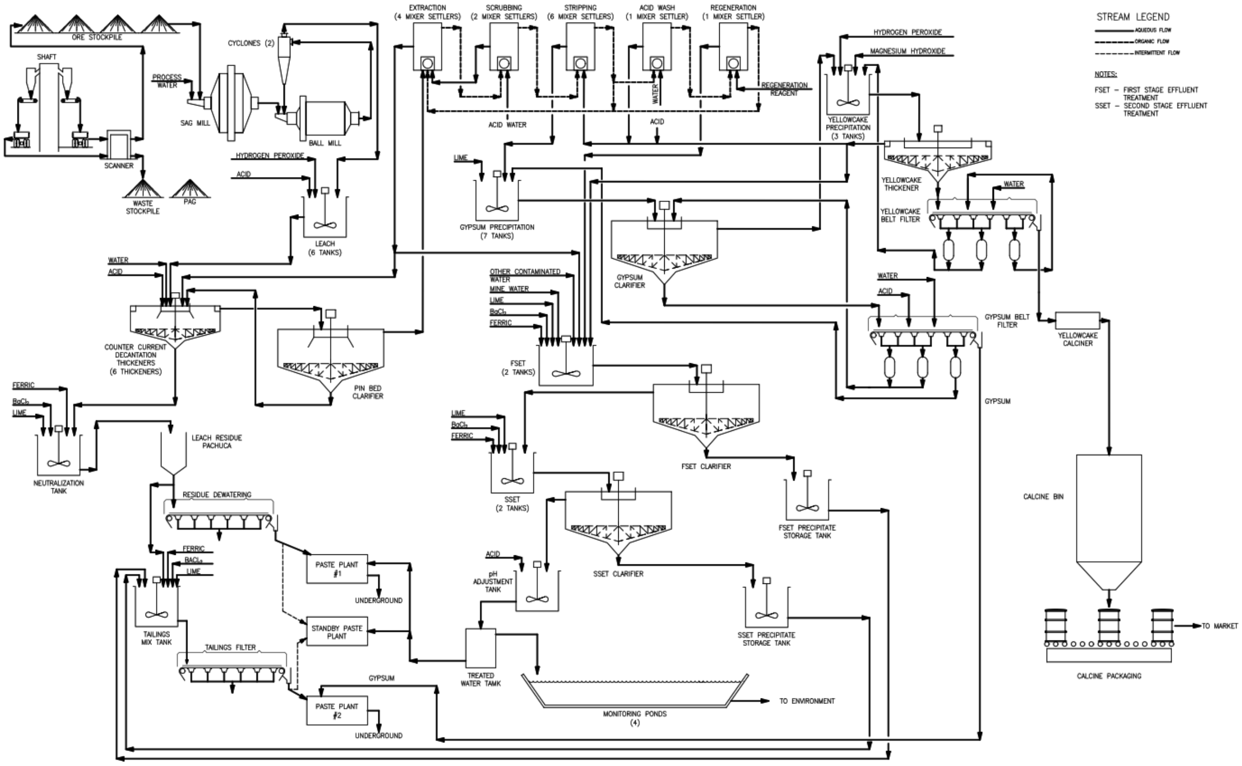
17.2.1 Ore Sorting and Storage

Ore will be sized UG using grizzlies and rock breakers. The hoisted ore will be loaded into an ore truck at the headframe. The truck will drive through a radiometric scanner to confirm the ore grade and the delivery location of the ore to the ore pad. Different ore grades and types can be stored in different piles.

17.2.2 Grinding

A loader operator will deliver ore to the ore feed hopper. Traffic in the ore storage and reclaim area will be restricted, to minimize ore contamination in the site area. A variable-speed feeder belt will feed ore from the hopper into the SAG mill at a prescribed rate similar to the ground ore feed rate to be fed to mill leaching. The ore will be weighed on the belt and will pass through a gamma radiation scanner to check its uranium content.

Figure 17-1: Process Flow Diagram



Note: Figure prepared by Wood, 2020.

Table 17-1: Production Design Requirements

Production Criteria	Units	Quantity
Ore feed rate	t/a	456,300
	t/op day	1,300
Design ore feed grade	%U_3O_8	3.05
Nominal ore feed grade	%U_3O_8	2.37
Maximum feed grade to mill	%U_3O_8	5.0
Plant uranium recovery	%	97.64
Production rate – design ore grade	lb U_3O_8/a	30,000,000
Production rate – nominal ore grade	lb U_3O_8/a	23,300,000
Operating time	hr/a	8,000
	d/y	351
Availability	%	95

Water will be added into the SAG mill feed and ore to provide the target percent solid content in the mill. SAG mill discharge will be transferred via gravity to feed the ball mill. The ball mill will also be fed recycled oversize particles from a classification cyclone situated above the ball mill. Water can be added to the ball mill feed and/or discharge, to maintain the target percent solid composition of slurries in the circuit. Ball mill discharge will be transferred to a pump box that will pump the ore slurry to the classification cyclone / cyclones.

The overflow stream of the cyclone is designed to have the target particle size of 100% passing 300 µm and the target 50% solid composition. The overflow stream of the cyclone will be transferred via gravity to a ground ore storage tank.

The ore storage tank will be mechanically agitated. It will provide surge capacity between the grinding circuit and leaching circuit and some degree of ore blending.

The grinding circuit will have tonnage capacity greater than required for leach feed. This will allow the grinding circuit to fill the ore storage tank. When full, the grinding circuit may be shut down to provide short periods (i.e., up to three hours) of grinding circuit maintenance, without disruption of feed rate to leaching.

17.2.3 Leaching

The first leach tank will be fed by a variable-speed centrifugal pump that will feed the rate of prescribed solids to the leaching circuit. The leaching circuit will consist of six mechanically-agitated tanks, connected in a series. Flow between each tank will occur by gravity. The discharge of each tank will be from a baffled upcomer, to mitigate short circuiting of solids in the tanks. In total, the tanks will provide the target 10-hour residence time to oxidize and dissolve the uranium.

The tanks will be heated with steam spargers to maintain the 50 °C leach temperature. Most of the sulphuric acid required will be fed into the first two to three tanks. This will also be the case with the hydrogen peroxide oxidant that will be fed to the leaching tanks. Sulphuric acid will be added to maintain the target minimum of 25 g/L to 35 g/L acid content in the leaching tank discharge, while the peroxide will be added to maintain the target greater-than 450 meV oxidation reduction potential (ORP).

If low iron ore is being leached, ferric sulphate will be available to provide supplemental ferric iron, which is required for faster oxidation of U^{+4} (non-soluble in the acidic solution) to U^{+6} (soluble in the acidic solution).

It is expected that 99.3% of uranium in ore will dissolve in the leaching circuit.

17.2.4 Counter-Current Decantation

A variable-speed pump will be used to pump slurry from Leach Tank 6 (the last tank of the leaching train) into a mix tank. Overflow solution from the thickener CCD 2 will also be transferred into this mix tank via gravity. The mixed slurry will be pumped to the center well feed of CCD 1, along with a flocculant flow to enhance settling. The overflow of CCD 1 (i.e., pregnant aqueous solution) will be transferred to a pumpbox and pumped to feed a pin bed clarifier. Underflow from CCD 1 will be removed by a variable-speed pump. The variable-speed pump will be controlled by the density of the underflow stream and by the load level of solids in the thickener. Underflow will be pumped to a small mix tank, where it will be mixed with the gravity overflow from CCD 3. This mixed solution will be pumped to the feed well of CCD 2, where it will be treated with flocculant. Similarly, the CCD underflows will be pumped to feed the next CCD (i.e., CCD 3 to feed CCD 4 until underflow of CCD 6 [the final CCD in the train]). CCD 6 underflow will be pumped to the residue neutralization tank.

In the residue neutralization tank, lime will be added to adjust the acidity of the slurry to pH 9, to neutralize the leach residue. Barium chloride and ferric sulphate will also be added to reduce the concentrations of dissolved elements such as radium (Ra), molybdenum, and arsenic. Neutralized residue will then be pumped to the residue storage pachuca tank.

Wash water will be fed into the feed mix tank of CCD 6. The wash water will be made up of the following.

- Priority 1: a percentage of SX raffinate flow (recycle as much acid as possible).
- Priority 2: acidized settling pond water.
- Priority 3: acidized fresh water.

The acid content of the total slurry solution must be maintained, to ensure that uranium is not reprecipitated in the CCD circuit. Enough wash water will be fed to the feed mix tank of CCD 6 to meet the target uranium content of the pregnant aqueous solution that will be overflowing from CCD 1.

The overflow of CCD 6 will transfer via gravity to the mix tank feeding CCD 5. The solutions will pass from one thickener to the next, in the opposite direction as the slurry of solids (i.e., from CCD 6 to feed CCD 5).

Circuit performance will be based on a combination of the following.

- The amount of uranium in the feed solution.
- The amount of wash water to be added to CCD 6 feed.
- The settled densities in the CCD thickeners.

Greater than 45% solids are expected in the CCD underflow. It is estimated that 99.5% of dissolved uranium will be washed out of the leached residue solids when using the train of six CCD thickeners.

17.2.5 Pregnant Solution Clarification

The overflow from CCD 1 will feed a pin-bed clarifier that will remove turbidity from the pregnant aqueous solution. The feed to the clarifier will be treated with a small quantity of flocculant to aid settling / clarification. The overflow of the clarifier will flow to the SX feed tank via gravity. The feed tank will have capacity to feed the SX circuit for approximately 2 hours. The small amount of underflow solids will be pumped back, to feed the CCD 1 thickener.

17.2.6 Solvent Extraction

The organic in the SX circuit will be made up of the following three components.

- A tertiary amine that selectively forms a bond with uranyl sulphate. Enough amine will be added into the solution to hold the design g/L U_3O_8. This is typically approximately 6%–12% amine reagent by volume).
- Isodecanol that will be introduced into the solution to enhance the separation of the pregnant aqueous from the organic after mixing ceases. Isodecanol is typically added to approximately half the volumetric concentration of the amine.
- A kerosene-type organic as the main carrier solvent.

In laboratory testing the proportions were 12% amine, 6% isodecanol, and 82% kerosene.

There will be four extraction mixer-settler units. Clarified pregnant aqueous solution will be pumped from the SX feed tank and into the extraction mixer 1, where it will be mixed with organic solution from extraction settler 2. As the organic and the pregnant aqueous are thoroughly mixed, the tertiary amine in the organic will bind to the uranyl sulphate and remove it from the pregnant aqueous.

After mixing, the mixer will discharge the solutions into a settler unit, where the solution will separate (i.e., the lighter organic will float on top of the heavier pregnant aqueous). Some of the organic in an extraction settler will be returned to its mixer; therefore, the ratios between organic and pregnant aqueous can be controlled in the mixer by recirculating the organic.

The pregnant aqueous that has settled out from extraction settler 1 will be fed to extraction mixer 2, where it will be met by the organic flowing counter-current from extraction settler 3. In this countercurrent flow, pregnant aqueous will be fed into extraction mixer 1 and will discharge as barren raffinate from extraction settler 4. Conversely, barren organic will be fed into extraction mixer 4 and will discharge from extraction settler 1 as loaded organic (i.e., high uranium content organic). Barren raffinate from extraction settler 4 will be pumped to the raffinate tank. Periodically, the organic that accumulates on the raffinate tank surface will be skimmed off and returned to the extraction circuit.

Much of the raffinate will report to the CCD 6 feed tank, where it will be recycled. As much raffinate as possible will be recycled, to capture the acid that it contains. However, recycling of raffinate will increase the circulating load of contaminant elements. This elevation of contaminant levels will require some of the raffinate to be bled to the effluent treatment circuit. The raffinate tank will be able to hold about 2 hours of raffinate generation.

At this point, the loaded organic is expected to contain 99.9% of the uranium that has been fed in the pregnant aqueous. The organic will be washed in two mixer settlers, with a small amount of acidic water flowing counter-current to the organic. The acid solution will wash some elements (e.g., arsenic) from the loaded organic as well as any small “bubbles” of pregnant aqueous that may have escaped extraction mixer settler 1 with the loaded organic. In both acid wash mixer settlers, pregnant aqueous will be recirculated to obtain the target organic-to-pregnant aqueous ratio in the mixers. The scrubbed loaded organic will have a high concentration of uranium and much lower concentrations of contaminating elements. However, some elements (e.g., molybdenum) can go with the uranium into the organic flow, to an extent.

There will be six strip mixer-settlers. In stripping, barren aqueous strip solution will be used to strip uranium off the organic. The stripping solution will be a strong acid solution that contains 400 g/L sulphuric acid. Scrubbed loaded organic will feed the strip 1 mixer, where it will be mixed with aqueous stripping solution from the strip 2 settler. The mixed solution separates in the strip settler. Much of the aqueous in the strip settler will be recirculated back to the strip 1 mixer, to maintain the target organic to aqueous ratio in the mixer. The remainder of the loaded strip solution will be pumped to the loaded strip tank. The loaded strip tank will be able to hold about 4 hours of loaded strip as it is generated. The loaded strip will be concentrated in uranium at 150 g U_3O_8/L.

Organic from strip 1 settler will feed the strip 2 mixer, where it will be mixed with aqueous from strip settler 3. This will be a counter-current arrangement, with the uranium reporting to strip 1 aqueous discharge as loaded strip, and the barren stripped organic discharging from strip 6 settler. The barren strip reagent solution will be fed into the strip 6 mixer and will move counter-current to settler 1.

The barren organic exiting stripping can contain droplets of aqueous that contain strong acid. The wash mixer settler will wash the organic with water and recover the acid that might be lost. The wash aqueous will be pumped to the strong-acid, strip solution make-up tank, where more acid will be added to the aqueous before it is used to strip the loaded organic.

Most of the washed organic will report to the barren organic tank; however, a portion will report to the mixer of the regeneration unit. A dilute solution of sodium carbonate will be used to keep the aqueous in the regeneration unit at an approximate pH of 9. This will strip the barren organic of elements (e.g., molybdenum) that otherwise could recirculate with the organic, and build up in concentration to reduce the organic loading capacity for uranium. The proportion of organic reporting to regeneration will be as low as possible, to obtain low contamination concentration levels in the circulating organic. If there is uranium in the barren organic sent to the regeneration unit (i.e., from incomplete strip performance), much of it will be lost to the spent regeneration solution. The spent regeneration solution will be transferred to the effluent treatment circuit. It is expected that the stripping of the loaded organic into the loaded strip will be 99.6% efficient (i.e., uranium lost to the regeneration stream will be approximately 0.4%).

17.2.7 Gypsum Precipitation and Washing

Lime will be added to increase the pH of the loaded strip solution, so as to remove the acid in the strip solution in preparation for precipitating uranium. Loaded strip solution will be pumped from the loaded strip tank into the first reactor tank of a train of seven tanks. Flow from one tank to the other will occur via gravity. Lime slurry will be added to each tank to gradually (i.e., in small steps) increase the pH to a target value of 3.0. As lime is added, it will react with the strong acid solution to precipitate gypsum. Gradual addition into high agitation will ensure that precipitation happens as slowly as possible, so as not to trap uranium in the gypsum particles as they are being precipitated. Precipitation will remove sulphate to the low levels that are required in the next uranium precipitation step. The total residence time in the gypsum precipitation circuit will be 4 hours.

Gypsum solids will be present at a relatively low percentage, as it is pumped from the last reactor tank (i.e. tank 7) to the gypsum clarifier. The concentrated underflow stream will be fed onto the gypsum belt filter. Gypsum will be dewatered on the gypsum filter.

When the gypsum is dewatered to a high degree, acidized fresh water will be added to the top of the cake, to wash the solution from the cake. Once the cake has been dewatered, another quantity of acidized fresh water will be added for a second wash. Following the acidized freshwater washes, there will be two more freshwater washes. The gypsum cake will then be fed to the paste backfill feed conveyor.

17.2.8 Yellowcake Precipitation and Washing

Purified loaded strip from the gypsum clarifier overflow will report to YC precipitation tank 1. In this tank, hydrogen peroxide will be added to precipitate uranyl sulphate as uranyl peroxide. The peroxide will be dispersed into a well-agitated slurry to minimize very fast localized precipitation. As the reaction progresses, the pH will begin to drop. A slurry of magnesia will be added to maintain a pH between 2.8–3.5. Tank 1 will flow by gravity to YC precipitation tank 2. In tank 2, the reaction will complete, and time will be given for the precipitate particles to grow. Total residence time in the YC precipitation tanks will be 4 hours.

YC precipitation tank 2 will overflow into YC wash tank 1, where the YC slurry will be mixed with barren strip filter backwash and excess YC belt filter filtrate. Wash tank 1 discharge will be fed to the YC wash thickener 1.

The thickened underflow of YC wash thickener 1 will be pumped to the YC belt filter. Flocculant will be added to the line to the pumpbox, so as to assist dewatering on the

YC belt filter. The dewatered cake on the belt filter will have two fresh water washes. Good washing of the dissolved solids from the YC on the belt filter will increase the YC product quality.

Belt filter filtrate will be recycled to precipitation tank 1 to dilute the slurry. This will reduce the viscosity of the slurry and assist in mixing in the peroxide reagent. Some of the YC solids from the thickener underflow will be recycled back to precipitation tank 1, to ensure that solid particles will be available to precipitate on as reagents induce further precipitation. This will assist crystal growth and make dewatering and washing more efficient.

The cake will discharge the belt filter at 70% solids and will report to a conveyor that will feed the calciner.

Barren strip will be removed from the circuit as overflow from YC wash thickener 1 and will report to the barren strip tank. Good solid-settling performance will be required in YC wash thickener 1, to ensure that YC solids do not escape with the barren strip to feed the effluent treatment system and cause uranium recovery loss. The barren strip tank can contain approximately 4 hours of barren strip generation.

17.2.9 Yellowcake Drying / Calcining and Packaging

YC from the YC belt filter will be fed to the YC pre-dryer by the YC conveyor. The pre-dryer will heat the YC to reduce the contained moisture feeding the calciner to minimize corrosion in the calciner. The pre-dryer will circulate hot oil flow to indirectly heat the YC and dry it to a temperature of about 120 °C. The water vapor discharged from the dryer will be scrubbed to remove particulates before the vapor will be discharged to the environment. Dried cake from the pre-dryer will report to the YC calciner.

The calciner will be an indirectly-heated, rotary type. The calciner will be heated by natural gas. The combustion gas flow that heats the dryer drum will be kept uranium-free and will discharge through a stack. A small ventilating air stream will pass through the calciner to ensure that no gasses are concentrated in the calciner. Upon exiting, the gas will pass through a scrubber to remove any particulates. The liquid discharge of the scrubber will report to the YC wash tank 2. The gasses will discharge via a stack to the environment.

In the calciner, the damp uranyl peroxide will be dried, molecular water driven off, and uranium peroxide oxidized to produce a U_3O_8 product. The design temperature will be 840 °C with a solids residence time of one hour. As well as oxidizing the YC, a small amount of volatile contaminants will also be driven out of the calcine (e.g., F).

The calciner discharge screw conveyor will be designed to cool the calcine to about 200 °C before discharging it to a calcine storage bin. The discharge bin will be able to hold about one day of maximum production.

The calcine bin will feed a packaging system that will load the calcine into 200 L steel drums. The drums will be sampled manually before lids are fitted and seal rings applied. The drums will then be washed thoroughly and dried. After being weighed, an identification label will be attached that includes the drum tare and total weight as well as the net weight of the product contained. The normal net weight of a drum will be about 400 kg. Typically, there will be about 100 drums packaged per mill operating day. Some empty drums will be stored in the packaging area. There will also be room in the packaging area for at least two days of production or about 200 drums. Lots will be loaded into truck vans and will be transported to the designated delivery point.

17.2.10 Neutralized Leach Residue Storage and Dewatering

The slurry from the leach residue will be pumped to the leach residue storage pachuca tank. The pachuca tank will be air agitated and store up to three hours of slurry. The discharge from the pachuca tank will be pumped to either the residue dewatering belt filter or the tailings mix tank.

About two-thirds of the NLR solids will be fed to the residue dewatering belt filter. Flocculant will be added to the slurry feeding the belt filter. The filter will dewater the residue to an estimated 70% solids. The dewatered cake will be fed to the paste plant to produce CPB.

The other third of the neutralized leach residue solids will be pumped to the tailings mix tank.

17.2.11 Tailings Storage, Mixing, Neutralization and Dewatering

First stage effluent treatment (FSET) clarifier underflow will be pumped to the FSET precipitate storage tank. This tank will be agitated to keep the slurry homogenous. The tank will provide storage for about 4 hours of precipitate generation. Similarly, second stage effluent treatment (SSET) precipitate slurry will be pumped and stored in the SSET precipitate storage tank with a similar 4 hour holding capacity. These tanks will decouple the effluent treatment circuit from paste backfill requirements of the UG mining and waste handling systems.

In the proportion that they are accumulated, streams of neutralized leach residue, FSET and SSET precipitate slurries will be fed into the tailings mix tank. A small quantity of lime will be added to ensure that the pH is maintained at 9. Barium chloride and ferric sulphate reagents will also be added to the mix tank to precipitate some solids that are present as dissolved solids in the slurry solution. A ratio of ferric iron to As + Mo will be maintained at greater than an excess ratio of at least 4:1. These reagents can also stabilize the pastes made in the subsequent process. Flocculant will also be added to the tank. The mix tank will have a 1.5-hour retention time. Slurry will then be pumped to the tailings belt filter.

The tailings belt filter will dewater the slurry to produce a cake and which will be fed to the paste plant to produce CPT. Some of the combined belt filtrate will be pumped to the paste plant to dilute the paste mix as required in the paste recipes. The remaining combined filtrate will be pumped to the effluent treatment plant.

17.2.12 Paste Backfill Plant

The cake from the residue dewatering belt filter will be mixed with water and binder (50% OPC / 50% Slag) to create the CPB. The CPB forms a structural backfill that will be used for all mine backfill purposes as well as UGTMF plugs and caps. The cake from the tailings belt filter will be blended with water and binder (100% OPC) to create the CPT. The CPT will be stored in the UGTMF permanently.

The paste plant is a continuous backfill plant that will consist of three replicate modules. One each will service the CPB and CPT requirements and the third will blend either CPB or CPT and will act as a standby to the two operational modules. Each backfill module will consist of a paste mixer, paste hopper and 100 bar hydraulic piston type paste pump. The three modules will be fed by a series of conveyor belts.

Filtered cake will be discharged into the continuous mixer(s) where rheology control water and binder will be added. The binder and water will be dosed into the continuous mixer(s) by a PLC to ensure that a consistent paste recipe is maintained. The paste recipe target yield stress will vary depending on the region of the UG mine or UGTMF receiving the material.

A binder system consisting of silos, weigh belts and screw conveyors will be used to feed binder to the continuous mixer(s). Binder will be delivered by bulk truck to two silos located next to the paste plant. Each silo will have a 500-t capacity complete with dust collectors. Binder will be weighed and fed continuously from the silos into the mixer feed chute(s).

The paste in each continuous mixer will discharge by overflowing into a dedicated paste hopper. Each hopper will provide for a continuous flow of paste to the UG. Paste will discharge out of each paste hopper to a 100-bar rated hydraulic piston type paste pump which will pump the paste to the UDS.

The CPB and CPT will be pumped down one of two surface boreholes, ranging from 65° to 70° in inclination from horizontal. A third borehole will be drilled and cased in Year 1 of operations and will serve as a standby. The surface boreholes will be drilled down to 440 Level to be used in the initial years of mine production. Later in the mine life, when upper levels of the mine are developed, the rock around the surface boreholes will be blasted on 380 Level and the casing pipe will be cut and routed so CPB and CPT can be delivered on 380 Level, as well as lower levels.

The surface boreholes will be cased with ceramic-in-epoxy lined steel pipe to provide enhanced wear protection. The surface boreholes will breakthrough into individual cutouts for geotechnical stability reasons. Diverter valves will be used as dump valves, one on each borehole, which will divert backfill to a sump area near the cut-outs in case of process upsets or emergency. The initial rerouting of piping for CPB to be diverted to the UGTMF will happen manually by means of a removable piping elbow.

By year two of UG operations a total of three boreholes and seven automated diverter valves will be required at the breakthrough location to fully automate the backfill flow from the three boreholes to the stopes and UGTMF respectively. One under each borehole as an emergency dump valve and four diverter valves are required to route the three boreholes to the two distinct areas of the mine. One borehole will be fully dedicated for the UGTMF, while the other two boreholes can go to either the mine stopes or the UGTMF.

Two side-by-side diameter normal (DN) 100 pipeline systems will be run to all major areas of the UGTMF. A twinned system is specified to prevent a potential operational issue that would occur if two boreholes were used at the same time and their flows merged UG at the change over station. One DN 100 pipeline system will be run to all areas of the mine stopes.

17.2.13 Effluent Treatment

Feed water will report to the first of two first-stage water treatment reactor tanks. Much of the mill effluent will be acidic and even when combined with slightly basic mine effluents, the pH will normally be lower than the target operating pH of 4.5. Lime slurry will be added to the reactor tanks to maintain the pH at 4.5. The free acid will react with the lime and a resulting gypsum precipitate will be formed. The metals in the effluent will begin to precipitate with the hydroxide that will be added by the lime to form metal hydroxides. Iron, arsenic, and molybdenum are the main metals of interest that precipitate; however, all existing metals will begin precipitating.

The raffinate added will normally have significant levels of ferric iron. If raffinate is not present or in low supply, ferric sulphate can be added to ensure an adequate ratio of ferric iron to arsenic and molybdenum (approximately 4:1). Much of the arsenic, molybdenum, selenium (Se) will be precipitated in first-stage water treatment conditions. These elements can co-precipitate with precipitates or be adsorbed onto surfaces of precipitate of iron compounds such as ferrihydrite and ferric and manganese hydroxides.

Generally, there will be a high enough oxidation reduction potential in the first-stage water treatment to keep arsenic in an arsenate form. This will make arsenic precipitation more efficient. It is also beneficial to inject air into the reactors to provide oxygen (O) to the system provide a relatively oxic environment. Air will be injected into the agitator blade area, will help to fluidize any radon from the effluents and ensure that all generated CO_2 is stripped and removed before the pH is increased in the second-stage water treatment. If present, CO_2 can make uranium more soluble in a higher pH solution.

Some barium chloride will be added in the first-stage water treatment reactors. Barium (Ba) will react with the sulphate that is plentiful in the first-stage water treatment to form barium sulphate. The Ra in the effluents will act similarly to Ba and much will be co-precipitated in first-stage water treatment.

The two reactor tanks will have a total residence time of one hour at design flow and one and a half hours at nominal flow. All the reagents can be added into either the first or second reactors as prescribed.

Elements precipitated in the first-stage water treatment reactor tanks will feed with the water into the first-stage water treatment clarifier. The clarifier will settle the precipitates and will provide a clarified stream that will flow to the second-stage water treatment reactors.

The solids in the underflow stream will be removed from the first-stage water treatment clarifier and will report to the first stage effluent treatment precipitate storage tank. The underflow slurry from this tank is pumped to the tailings mix tank where it is mixed with the tailings streams and neutralized.

In the two second-stage water treatment reactor tanks, more lime will be added to increase the pH to 10.5. As the pH is increased, iron, arsenic, and molybdenum, as well as other metals present, will be precipitated. More ferric sulphate as well as barium chloride will be added to precipitate more of the oxyanions as well as Ra. Sulphuric acid will be available for pH control or id additional sulphate is required. All reagents can be added in either of the two second-stage water treatment reactor tanks. As with the first-stage water treatment, the residence time in the reactors will be a minimum of 1 hour. At the nominal flow rate residence time will be approximately two hours. Precipitated solids will be removed from the second stage effluent treatment precipitate storage tank. As with the first stage precipitate, the underflow slurry from this tank is pumped to the tailings mix tank where it is mixed with the tailings streams and neutralized. The second-stage water treatment clarifier will overflow into the pH adjustment tank. Dilute sulphuric acid will be added to adjust the effluent pH to 6.5 (for release requirement) before it is pumped to the treated water tank.

The treated water tank will be a source for the treated water distribution pump that supplies recycled treated water to mine, mill process and paste plant uses. Use of this water will reduce the amount of fresh water that will need to be used and therefore the amount of effluent discharged into the environment. The treated water tank will overflow into one of the four monitoring ponds. The overflow system of this tank will ensure that there is constant source of treated water for recycling.

17.2.14 Feed and Effluent Monitoring Ponds

The mine sump pumps will discharge to a surface feed settling pond. The pond can contain 4–5 days of nominal mine water discharge. As water is retained in the pond, suspended solids will settle.

Water that runs off from potentially-contaminating uses such as mine and mill dry facilities and maintenance shops, will discharge to the feed settling pond.

Potentially contaminated runoff will be collected in two site runoff ponds as described in Section 20.2.1.

A contingency pond will be available to accommodate pond cleaning and maintenance activities as well as to provide additional capacity for handling or storing contaminated water.

Water will be pumped from the feed settling pond to feed either the grinding circuit, CCD wash circuit or the first-stage water treatment circuit. The flow of water to the first-stage water treatment circuit will be maintained at a prescribed flow rate. Using settling pond water in the mill circuits reduces fresh water use and the volume of treated effluent discharged to the environment.

Effluent monitoring ponds will allow storage of treated effluent until water parameters are assayed and confirmed to meet discharge criteria. As a pond receives water from the treated water tank, the flow will be sampled. Once a monitoring pond is full, the composite sample that represents the full pond will be taken to the on-site laboratory and assayed. If all the assays are within the acceptable ranges, approval will be given for the pond to be discharged to the environment. As the pond is discharged to the environment, another composite sample will be taken that will be representative of the discharge to the environment. The assays of this composite sample will be reported as required to the control agencies. If assays are outside the acceptable ranges for the pond fill composite, the pond contents will be pumped back to the feed settling pond for reprocessing in the effluent treatment plant. At nominal fill rates a monitoring pond will hold about 18 hours of treated effluent.

17.3 Uranium Recovery

Uranium recovery was estimated by evaluating the recovery of the individual circuits and combining these into an overall recovery. Total net uranium metallurgical recovery is forecast to be 97.6% as shown in Table 17-2.

Table 17-2: Uranium Metallurgical Recovery by Unit Operation

Circuit	Recovery (%)
Leach extraction	99.3

Circuit	Recovery (%)
CCD washing efficiency	99.5
SX efficiency	99.9
Stripping / regeneration efficiency	99.6
Gypsum circuit total efficiency (including washing)	99.9
YC precipitation efficiency	99.9
Barren strip, calciner, packaging	99.9
Unaccounted losses	0.4
Next Overall Mill % U Recovery	**97.6**

17.4 Energy, Water, and Process Materials Requirements

17.4.1 Water

Fresh water consumption was estimated for the different mill processes and totals about 123 m^3/h. Fresh water reduction opportunities include using mine water from the settling pond in grinding and CCD residue washing circuits. Internal use of process waters includes using filtrates and treated water in the paste plants. Treated water will also be available for further use for washing and flushing applications in the mill and mine. Opportunities to reduce fresh water consumption identified to date total approximately 147 m^3/h (this has already been accounted for in the water balance and has resulted in the fresh water consumption of 123 m^3/h).

17.4.2 Reagents

Reagents will include the following.

- Sulphur
- Sulphuric acid (94% H_2SO_4)
- Unslaked lime (CaO)
- Hydrogen peroxide (H_2O_2)
- Flocculant
- Kerosene
- Tertiary amine ($N\text{-}R_3$)
- Isodecanol ($C_{10}H_{21}OH$
- Sodium carbonate (Na_2CO_3)
- Magnesia (MgO)
- Barium chloride ($BaCl_2$)
- Ferric sulphate ($Fe_2(SO_4)_3$)
- Ordinary Portland cement (OPC)
- Slag

17.4.3 Energy

Energy requirements for the Project are discussed in Section 18.8. The process plant is estimated to require 7.4 MW. The paste plant is estimated to require 0.9 MW.

17.5 Comments on Section 17

The proposed process flowsheet is conventional for the uranium industry and will use conventional equipment.

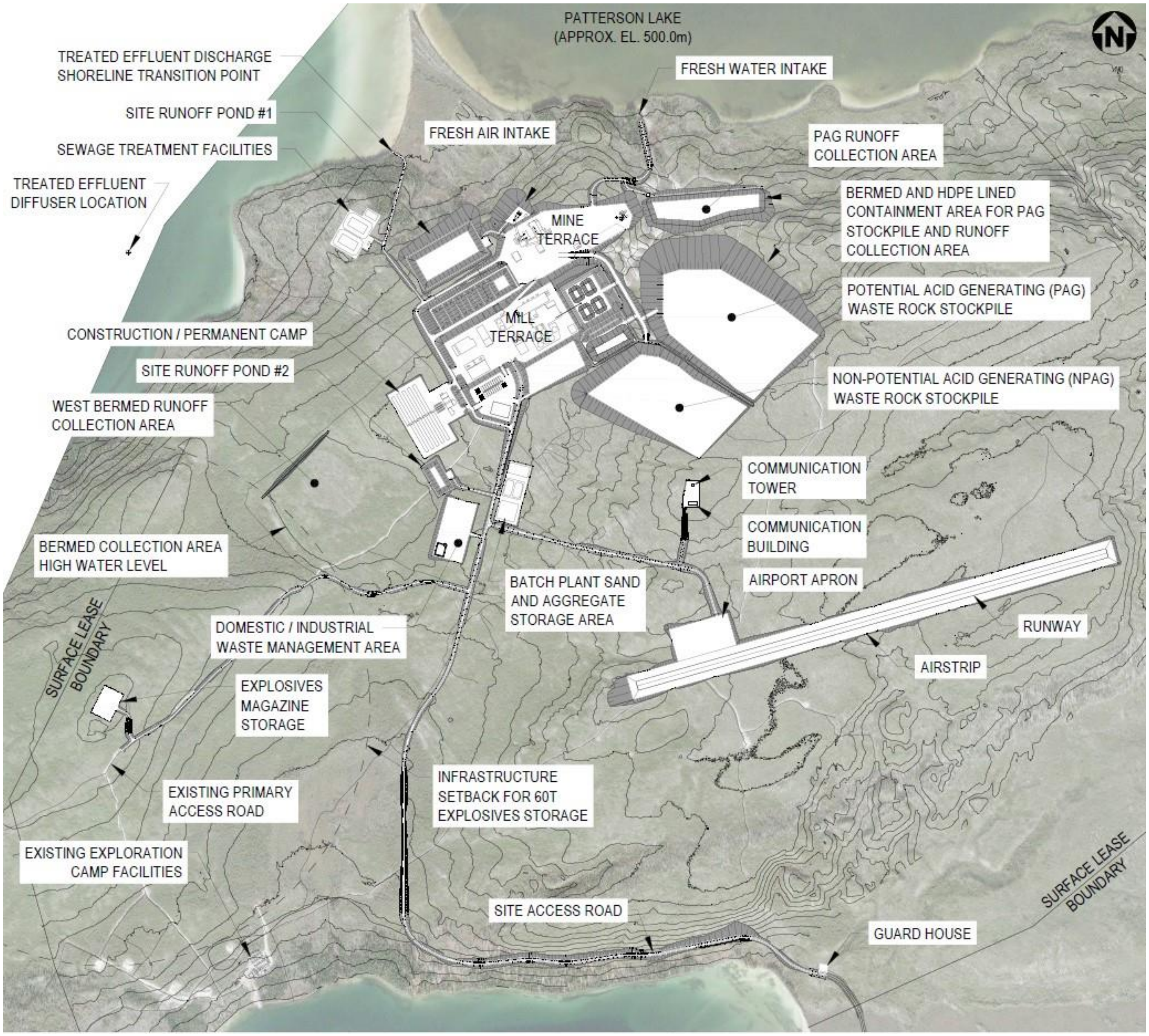
18.0 PROJECT INFRASTRUCTURE

The key infrastructure planned for the Project include the following.

- UG mine with two vertical shafts
- UG infrastructure, including:
  - Material handling systems
  - Maintenance facilities
  - Fuel bay
  - Explosives magazine
  - Ventilation
  - Paste backfill and paste tailings distribution system
  - Electrical and communications facilities
  - UG water supply
  - Dewatering facilities
- UGTMF
- Surface support infrastructure for the mine, including:
  - Headframe and hoist facilities
  - Surface explosives magazine
  - Ventilation fans
- Surface support infrastructure for the mill, including:
  - Process plant
  - SX plant
  - Effluent treatment plant
  - Acid plant
- Site support infrastructure, including:
  - Accommodation camp
  - LNG facilities
  - LNG power plant
  - Mine and mill dry facilities
  - Analytical and metallurgical laboratory and maintenance facility
  - Warehouse
  - Security buildings
- Surface ore storage stockpile facility
- Waste rock storage facilities for PAG, NPAG, and special waste materials
- Water management facilities, including:
  - Two storm water runoff ponds
  - Six contact water process ponds
  - Conveyance and diversion structures
- Domestic / industrial waste management areas
- Airstrip

The layout of the planned surface infrastructure is shown in Figure 18-1. UG mine layouts are discussed in Section 16.0 of this report.

Figure 18-1: Proposed Surface Infrastructure Layout Plan



Note: Figure prepared by Wood (2020).

18.1 Roads and Logistics

Construction of the access road from Highway 955 to the site began in 2016 and was completed in 2017 with improvements to the surface composition. The road alignment varies to best fit the existing land and to avoid steep slopes and excessive embankment fills. As a result, there are sharp turns and blind spots that will create hazards for traffic to and from the site. Some corners will be cleared to minimize safety concerns. The road may need to be sprayed with water in dry weather to avoid rutting, dust, and surface rework.

The site road plan is designed to allow for intermittent closures while maintaining access to all major infrastructure. As of 2021, there is an access trail from the existing camp site to the future plant site to accommodate exploration activities. This road may be used by the site preparation contractors during the first year of construction. However, it will not be considered for use in the final design.

Access roads will be 8 m wide (4 m per lane). Service roads will be 6 m wide where two-way traffic is required, and 5 m wide with pull-out lanes where one-way traffic is required. Haul roads will be 12 m wide. The use of haul roads will be restricted to heavy haul truck traffic.

18.2 Stockpiles

Stockpiles are discussed in Section 20.2.1.

18.3 Waste Storage Facilities

The planned waste storage facilities are discussed in Section 20.2.1. Any radiologically-contaminated waste will be disposed of in the UG workings.

18.4 Tailings Storage Facilities

No surface tailings management facilities are included in the FS design. According to the project plan, all processed waste UG will be stored in a purpose-built UGTMF. Refer to Section 16.5 for further information.

18.5 Water Management

Water management is discussed in Section 20.2.2.

18.6 Built Infrastructure

The buildings and structures that will be required for the site are summarized in Table 18-1.

Table 18-1: Buildings and Structures

Item	Location	Comment
SAG Mill Feed Conveyor	Mill Island	The belt conveyor feeding the SAG mill will be housed in a fully-enclosed prefabricated modular gallery.

Item	Location	Comment
Process Plant Building	Mill Island	Stick-built, completely enclosed steel structure with a ridged sloping roof. The building will have personnel access doors, overhead access doors, air intake louvers, and wall exhaust fans.
Headframe – Production Shaft	Mine Island	Headframe design includes a collar house, sub-collar, ventilation plenum, head sheaves, skip dump, ore bin, and waste bin.
Hoist House – Production Shaft	Mine Island	Hoist house will include two permanent double drum hoists, an auxiliary single drum hoist, compressor room, control booth, electrical room, and an overhead crane.
SX Building	Mill Island	Pre-engineered, completely enclosed steel structure with a ridged sloping roof. The building will have personnel access doors, overhead access doors, air intake louvers, and wall exhaust fans.
Effluent Treatment Building	Mill Island	Pre-engineered, completely enclosed steel structure with a ridged sloping roof. The building will have personnel access doors, overhead access doors, air intake louvers, and wall exhaust fans.
Acid Plant Building	Mill Island	Pre-engineered, completely enclosed steel structure with a ridged sloping roof. The building will have personnel access doors, overhead access doors, air intake louvers, and wall exhaust fans.
Drum Storage Building	Mill Island	Pre-engineered, completely enclosed steel structure with a ridged sloping roof.

Item	Location	Comment
Maintenance and Warehouse Building	Mill Island	Insulated pre-engineered fabric building. The maintenance shop will occupy half of the building. This area will provide sufficient space to rebuild and repair process equipment, as well as fabricate items to support the operations of the site. The maintenance shop will include a drive-through maintenance bay that will be located on the west end of the building, with a 10 t overhead crane. The warehouse side of the building will be used to stock supplies and equipment required for the ongoing plant operation and maintenance. The warehouse will include a truck receiving platform and a 7.5 t overhead crane.
Wash Bay Building	Mill Island	Insulated, pre-engineered fabric building. The wash bay building will have two drive-through wash bays.
Administration	Mill Island	A single-storey, prefabricated modular building. There will be office and cubical space for approximately 58 people in the administration building. There will also be two meeting rooms, two training rooms, a kitchen, and a lunchroom.
Analytical and Metallurgical Lab	Mill Island	A single-storey, prefabricated modular building that will house the analytical and metallurgical labs.
Mine Dry Facility	Mine Island	A two-storey, prefabricated modular building. The female and male dry areas will be sized for 27 and 114 workers, respectively. It is anticipated that there will be office and cubical space for approximately 27 people in the mine dry.
Mill Dry Facility	Mill Island	A single-storey, prefabricated modular building. The female and male dry areas will be sized for 17 and 75 workers, respectively.
Intake Fans with Heaters	Mine Island	Twin surface air fans, installed in a horizontal arrangement with natural gas direct-fired heating systems.
Exhaust Fans	Mine Island	Twin exhaust air fans, arranged horizontally, with the exhaust outlet discharging upwards to provide better dispersion and to prevent the outlet from freezing.

Item	Location	Comment
Valve Houses	Various	There are ten valve houses throughout the site. Each will be a prefabricated modular building.
Explosives Magazine	Southeast of Plant Site	Prefabricated modular shipping-container-style building.
Domestic Waste Incinerator Building	Domestic Waste Pad	Pre-engineered, completely enclosed steel structure with a ridged sloping roof.

As discussed in Table 18-1, the following three building design types will be used.

- **Pre-engineered**: all process and internal platforms / structures inside these buildings will be stick-built and either supported independently of the shell structure or tied to the pre-engineered columns, where possible.
- **Stick-built**: each building and its internal platforms / structures will be designed as one structure.
- **Modular**: standalone structures fabricated off-site that will be shipped to the site as a single unit or multiple sections, and supported on independent foundations on-grade or on elevated structural platforms.

All facilities will include the required electrical, HVAC, and fire protection services, as well as any other required services.

18.6.1 Mine Island

The term "mine island" refers to the prepared area surrounding the Production Shaft and Exhaust Shaft that will house most of the surface infrastructure required to support UG mining. The surface of the pad will be approximately 200 m × 80 m.

18.6.2 Mill Island

The term "mill island" refers to the prepared area surrounding the process plant that will house the surface infrastructure required to support the mill and some of the surface infrastructure required to support the site. The surface of the pad will be approximately 400 m × 175 m.

18.6.3 Site Security

The site gatehouse will be a 4 m × 20 m modular building that will include washroom facilities and water storage for security personnel. Gate arms will be used to control site access.

18.6.4 Fire Protection

A standard, deep-buried, interconnected firewater loop will be installed. It will encircle the process plant and the Production Shaft. Fire is an inherent risk that must be managed in SX plants. The SX plant will have its own specially-designed fire suppression system.

18.7 Camps and Accommodation

The NexGen Rook I site camp will be a modular, single-storey facility. It will provide a comfortable home for all users (i.e., NexGen employees, consultants, contractors, and other Rook I personnel that will be staying on-site). The camp will be located on the west side of the site where the main facilities are located, as shown in Figure 18-1.

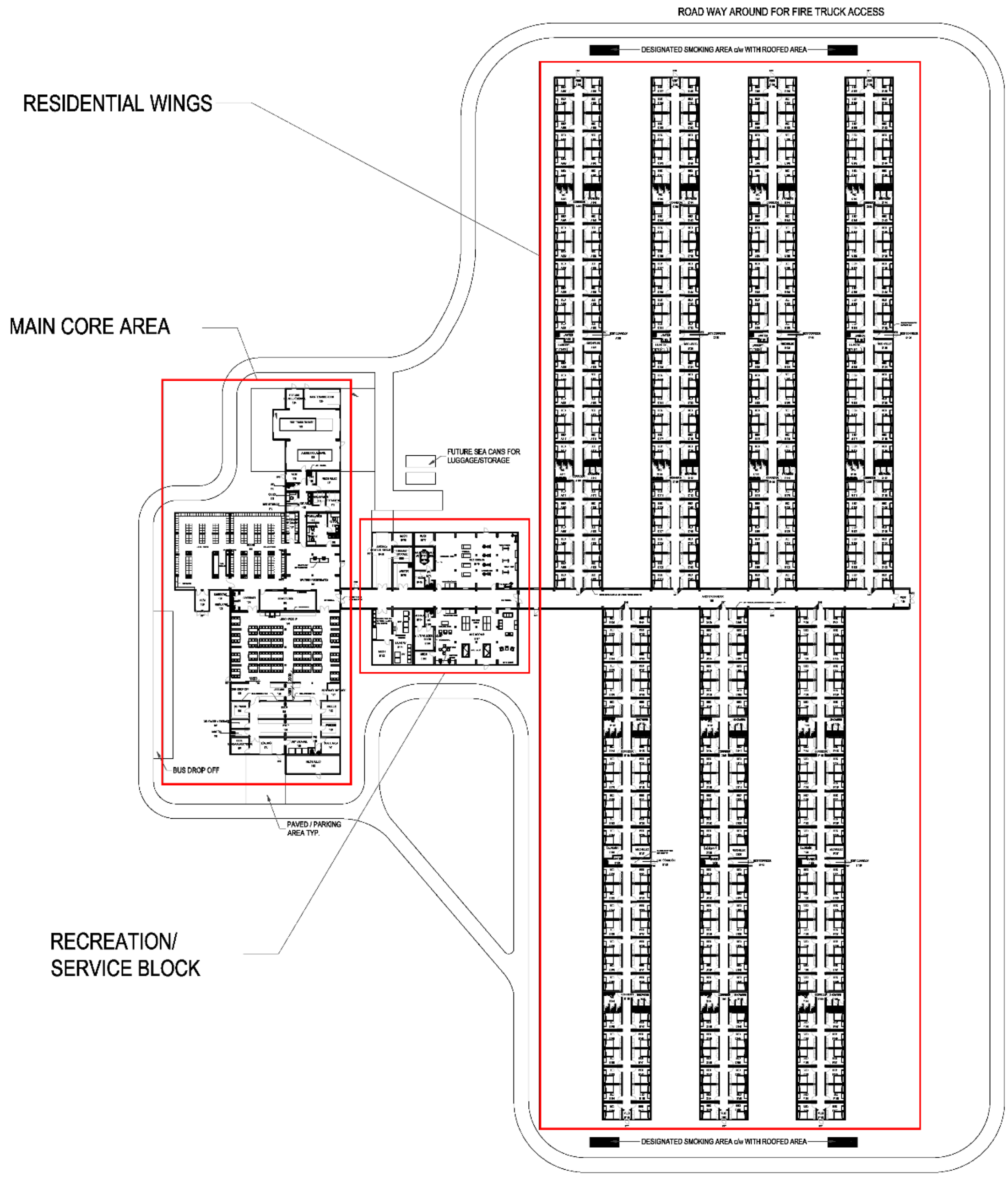
Users will be able to access the camp via the mine access road that runs from the guard house to the intersection at the roads to the mill terrace and mine terrace. The camp will be on the west side of the mine access road, opposite the construction office complex.

The maximum capacity of the camp will be 350 users during the construction phase. As construction contractors complete work on-site, and permanent users begin to start shift rotations, the population will reduce to approximately 300 users. One residential wing will be removed as the population decreases throughout the operation phase of the mine.

Camp users will require minimal parking since they will mostly travel to the site by small plane to the on-site airstrip. Forty electrified parking spaces and two barrier-free spaces will be provided for users travelling to the site by different means.

The camp design includes three major sections: the main core area, the recreation / service block, and the residential wings (see Figure 18-2).

Figure 18-2: Camp Layout



The camp design accomplishes the following.

- Provides semi-private spaces, such as individual rooms for one user that will be shared a rotating basis.
- Provides amenity spaces for different activities, including dining and recreation.

The approach to creating the camp design focused on the following four key concepts.

- Provide users with choices for activities during their time off while at site.
- Create a connected residence that provides access to all areas.
- Provide private spaces.
- Provide effective and efficient building services.

18.8 Power and Electrical

The Project is located in a region of northwest Saskatchewan with road access, but the area is devoid of other infrastructure. There is a 14.4 kV, single-phase power line approximately 95 km from the site; however, it is of insufficient capacity for the Project's scope. The nearest sub-station to the site with sufficient capacity for the Project's powered requirements is approximately 200 km away.

From a study completed during the PFS, it was determined that the NexGen Rook I site would be powered by an on-site generation plant, due to a lack of existing power infrastructure and a high cost for the installation of a new transmission line.

Power will be generated by an LNG electrical power plant. The 13.8 kV power plant will house nine generators sized at 3.329 MW, where eight generators will be operating with one stand-by unit. The total planned capacity of the plant is 26.5 MW, based on a nominal demand of 24.1 MW. Table 18-2 summarizes the projected power requirements.

This power plant will be modular so that power can be installed in phases, based on scheduling requirements. Discussions with LNG power plant vendors have confirmed that a portion of the plant (i.e., four or five generators) can be installed early to support construction activities (e.g., shaft sinking, camp, and administrative / office complex). This installation is currently scheduled to provide power early in Year -3, with the remaining portion of the plant being installed to provide power in Year -1.

The plant design includes LNG storage and filling facilities with the fuel being trucked to the site.

Table 18-2: Power Requirements Projection

Description	Power Consumption (MW)
Mine	13.4
Process/Mill Terrace	8.6
G&A	2.1
Total	**24.1**

18.9 Water Supply

The site water system will be required to draw raw water from a single location at Patterson Lake. Once drawn from the lake, the fresh water will need to be distributed to yard hydrants and fire suppression systems for use as fire water, to surface, fill trucks, and storage tanks for use as potable water, and to the mine terrace facilities for use as process water.

Each use will require equipment for pumping, storage and conveyance, and recirculation. The water will be reused if possible, depending on quality and quantity requirements, regulatory requirements, costs, and feasibility.

The freshwater distribution system for the mine and camp area will require raw water intake from the north side of Patterson Lake. This water will be used as process, fire, and potable water for the UG mine, and as fire and potable water for the camp area.

The water distribution system will comprise the following.

- A raw water intake structure
- A freshwater pump station (FWPS) and pumping gallery
- Freshwater storage tanks
- Piping to the water treatment plant
- Processing facilities and utility facility buildings

18.10 Comments on Section 18

Based on the information provided in this section, Wood and Stantec believes that the infrastructure considerations outlined are sufficient to support the proposed mine plan.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Overview

The information in this section regarding the uranium industry is from the World Nuclear Association website, and it has not been independently verified by the Project's QP.

As of 2021, production from world uranium mines supplies 90% of the worlds power utilities' uranium requirements. Primary production of uranium from mines is supplemented by secondary uranium supplies. Primary production from mines is supplemented by secondary supplies, formerly most from ex-military material, but now the products of recycling and stockpiles built up in times of reduced demand.

The spot prices quoted until 2007 only applied to day-to-day marginal trading, and represented a small portion of the world's uranium supply. Since 2008, the portion of the material traded at spot prices has approximately doubled in the last decade, to approximately one-quarter of the world's uranium supply.

Uranium is mostly traded via 3-15-year term contracts. Producers sell uranium directly to utilities at a higher price than the spot market, which reflects the security of the supply. However, the specified prices in these contracts are often based on the most recent spot price at the time the contract is established and signed; although slightly higher, the contract prices are therefore similar to the spot prices.

Over two-thirds of the world's uranium production from mines is from Kazakhstan, Canada, and Australia. An increasing quantity of uranium—50% as of 2021—is produced via in situ leaching. Table 19-1 presents a breakdown of the top uranium-producing countries.

Table 19-1: Production of Uranium Worldwide (2019)

Country	Tonnes Mined in 2019	Percentage of World Production (%)
Kazakhstan	22,808	42%
Canada	6,938	13%
Australia	6,613	12%
Namibia	5,476	10%
Uzbekistan (estimate)	3,500	6%
Other	9,417	17%
Total (worldwide)	54,752	100%

Source: World Nuclear Association website, January 2021.

The following excerpt is from the World Nuclear Association website (2021).

> About 440 reactors, with [a] combined capacity of over 390 [gigawatt electrical (GWe)], require some 79,500 tonnes of uranium oxide concentrate, containing about 67,500 tonnes of uranium (tU) from mines (or the equivalent from stockpiles or secondary sources) each year. This includes initial cores for new reactors coming online.
>
> The capacity is growing slowly, and at the same time, the reactors are being run more productively, with higher capacity factors and reactor power levels. However, these factors' increasing fuel demand are offset by a trend for increased efficiencies, so demand is dampened—[from 1970 to 1990], there was a 25% reduction in uranium demand per [kilowatt hour] output in Europe due to such improvements, which continue today.
>
> Because of the cost structure of nuclear power generation, with high capital and low fuel costs, the demand for uranium fuel is much more predictable than with probably any other mineral commodity. Once reactors are built, it is very cost-effective to keep them running at high capacity and for utilities to make any adjustments to load trends by cutting back on fossil fuel use. Demand forecasts for uranium thus depend largely on installed and operable capacity, regardless of economic fluctuations. However, this picture is complicated by policies which give preferential grid access to subsidised wind and solar photovoltaic (PV) sources.

The World Nuclear Association website notes that mineral price fluctuations are related to demand and perceptions of scarcity. The price cannot indefinitely stay below the cost of production, nor can it remain at a very high price for longer than it takes for new producers to enter the market and for supply anxiety to subside.

19.2 Current Market Activity

Information in this sub-section is from a UxC report prepared for NexGen, entitled UxC Special Report (November 2020).

19.2.1 Uranium Demand

This section outlines UxC's latest base case forecast for the world's reactor count and nuclear generating capacity (GWe net) through 2035. This section also lists total base case uranium requirements for each of the forecast years.

For reference, as of October 2020, there are 437 operable reactor units in 31 countries with approximately 388 GWe capacity, which translates to base requirements of 165.2 Mlb U_3O_8, as presented in Table 19-2.

Table 19-2: Global Nuclear Power Data in Late 2020

Number of Countries	Number of Reactors	Total Capacity (GWe Net)	Total Uranium Requirements (Mlb U_3O_8)
31	437	388.0	165.2

Table prepared by UxC, 2020.

For the 2022 starting forecast, UxC anticipates 33 countries will have nuclear energy capacity, with a total of 442 reactors world-wide, and a total capacity level of approximately 393.6 GWe net. In 2022, total base case uranium requirements will decrease slightly to 159.5 Mlb U_3O_8.

By 2025, as aging reactors are taken offline at a rate faster than replacement units are added, UxC projects that 34 countries will operate 434 reactors (396.2 GWe) and have a base case requirement of 173.0 Mlb U_3O_8. By 2030, UxC forecasts that 34 countries will have 464 reactors (434 GWe), and a base case requirement of 191.8 Mlb U_3O_8.

Finally, by 2035, UxC expects that 36 countries will have 468 operating reactors (458 GWe), and a base case requirement of 204.7 Mlb U_3O_8.

Figure 19-1 presents the uranium market requirements in Mlb U_3O_8, which are derived from UxC's Uranium Requirements Model (URM) of the commercial nuclear fuel cycle for the base, high, and low requirements scenarios, respectively.

Figure 19-1: Uranium Requirements Model 2018-2035

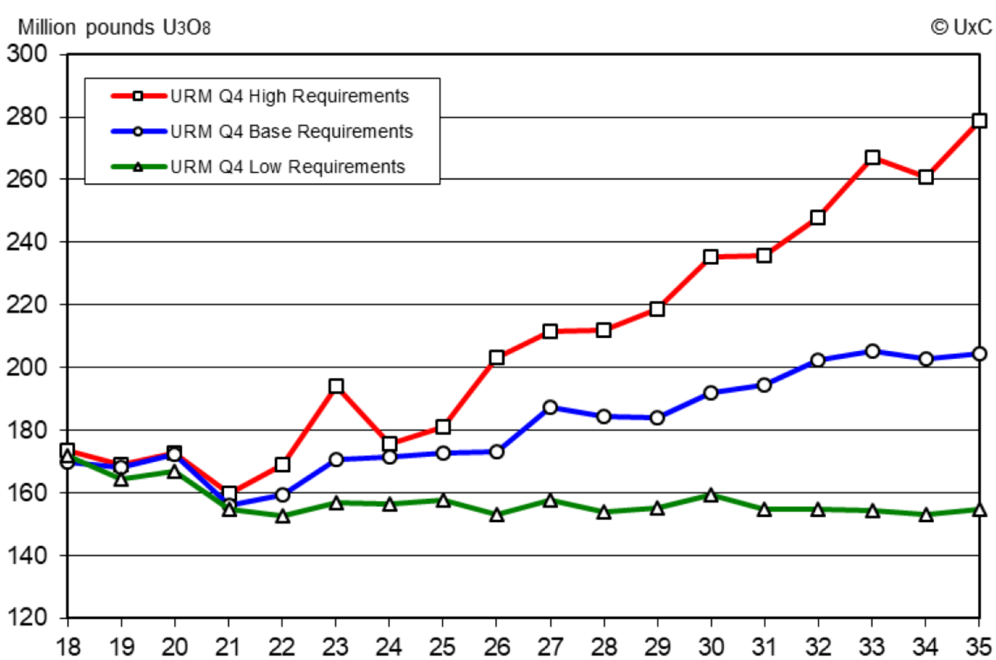


Figure prepared by UxC, 2020.

In the base case scenario, requirements increase from 159.5 Mlb U_3O_8 in 2022, to 173.0 Mlb U_3O_8 in 2025. By 2030, requirements rise to 191.8 Mlb U_3O_8 before ascending to 204.7 Mlb U_3O_8 in 2035.

The growth over this period is attributable to new reactor capacity in Asia, mainly from China, where requirements are expected to grow 188%, from 23.3 Mlb U_3O_8 in 2022 to 67.1 Mlb U_3O_8 by 2035.

Furthermore, growth is also expected in Eastern Europe, Africa, and the Middle East, which all have ambitious reactor build plans that will require increasing uranium supplies to meet the demand. In the base case scenario, total requirements will increase approximately 28% from 2022 to 2035.

19.2.2 Uranium Supply

UxC estimates that 2020 world uranium production will total approximately 122 Mlb U_3O_8, which is 12% lower than the 139 Mlb U_3O_8 produced in 2019. Looking ahead to 2021, world production is expected to increase 10% to 135 Mlb U_3O_8. In 2022 through 2025, world production is forecasted to increase from 135 Mlb U_3O_8 to 142 Mlb U_3O_8.

However, in 2026 to 2028 production is forecasted to increase to 160 Mlb U_3O_8, before declining to 147 Mlb U_3O_8 in 2029 as several uranium mines exhaust their reserves. In the mid- to late-2020s, new uranium projects are needed as secondary supplies will decline significantly during this period.

From 2030 through 2035, global production is forecasted to decline from 154 Mlb U_3O_8 per year to 118 Mlb U_3O_8 per year, which is significantly below projected global demand in the range of 199-211 Mlb U_3O_8 per year.

19.2.3 Supply and Demand Scenarios

UxC analyzed current production rates for existing and planned uranium production projects. These rates were based on company plans, where known; otherwise, these were based upon UxC's estimates of future production potential. As these relate to planned or potential production, the rates may overstate or understate the eventual level of production, but this will ultimately be determined by the market.

Figure 19-2 presents broad mid-case supply and demand requirements and the market demand range. The information in Figure 19-2 was sourced from the UxC proprietary URM.

Figure 19-2: Market Demand versus Mid-Case Production Sources, 2008-2035

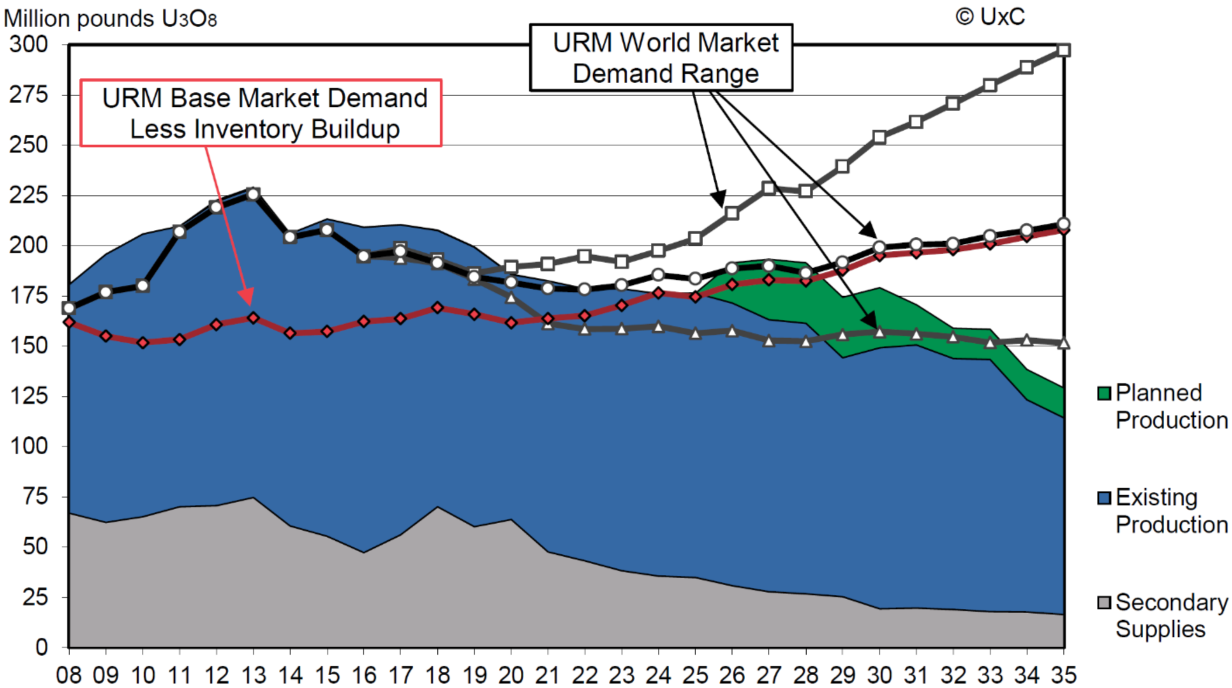


Figure prepared by UxC, 2020.

As the supply and demand differentials indicate, a potential supply shortfall will begin in 2022 compared to the base demand case. This could possibly begin as early as 2021 if the high demand forecast proves to be accurate.

19.3 Price Forecast

Information in this sub-section is from a UxC report prepared for NexGen, entitled UxC Special Report (November 2020).

This section presents uranium spot price forecasts developed using UxC's proprietary U-PRICE® model, which is an econometric simulation model of the uranium market. The U-PRICE model was designed to consider key factors that influence the uranium market. The structure of the model allows for an integrated simulation of uranium prices and related market variables.

The model incorporates additional information regarding the historical relationships of specific factors in the market. Therefore, the U-PRICE model provides an enhanced tool to quantify much of UxC's existing analysis and published indicator information.

Using various input assumptions (refer to Table 19-3), UxC used the U-PRICE model to develop three price forecasting scenarios: the Mid-Price, High Price, and Low Price scenarios. The forecasting timeframe is 18 years, from 2018 to 2035. Due to market uncertainties and the potential for unpredictable events that may occur during the forecasting period, interval forecasts were developed for each price scenario using 70% and 90% statistical confidence bands. Specifically, the confidence band was statistically determined and calculated based on variations of historical prices.

Table 19-3: Key Assumptions Used to Develop Spot U_3O_8 Price Scenarios

Factor	Descriptions: Factors and Assumptions
Demand Outlook	This is projected demand for uranium using the URM based on UxC's latest forecasts of the total number of reactors and electric capacity.
Market Outlook and Perception	Factors included in this category mainly reflect the psychological impacts of significant events on market participants' general perception of the uranium market.
Primary Production	Both primary uranium production from existing mines and potential production from new projects are addressed in this supply-side factor.
Secondary Supplies	This category includes non-traditional sources of supplies such as uranium produced via underfeeding and tails re-enrichment, U_3O_8 in Enriched Uranium Product (EUP) inventories, and mixed-oxide fuel (MOX) / reprocessed uranium (RepU).
Separative Work Units (SWU) Market Developments	This factor examines the impact of changes in the SWU market on the price of uranium due to the substitutability between produced uranium and uranium from enrichment programs.
Exchange Rates	The potential impacts of macroeconomic factors on uranium prices (such as the strength of the US dollar, monetary policies, and oil prices) are analyzed and included in the price projections.

Table prepared by UxC, 2020.

The projected composite prices presented in Figure 19-3 were developed using a probability-weighted average of the three price scenarios.

Figure 19-3: UxC Price Forecast Comparison

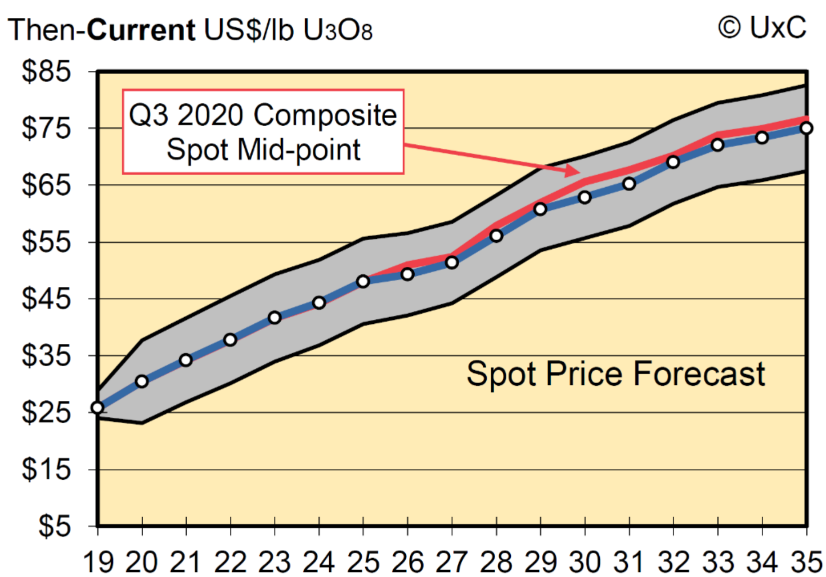


Figure prepared by UxC, 2020.

The probabilities were determined using monthly spot prices recorded during three distinct periods.

- The period from January 2004 to June 2007 when the spot price exhibited an increasing trend.
- The persistent price declining period from June 2011 to December 2016.
- The period from February 1999 to October 2003 when the spot price fluctuated within a relatively narrow range.

Beyond the medium term, improved market fundamentals helped trigger a sustained price recovery. However, the key factor driving the upward momentum and the sustainability of the uranium price continues to be the growth of nuclear power.

As with the spot uranium price forecasts, there are three scenarios for long-term (LT) contract base price projections: Mid LT Base, High LT Base, and Low LT Base. The two most commonly used pricing approaches in LT uranium contracts are base-escalated and market-related pricing. The key assumptions used to develop each scenario were consistent with those used in forecasting the spot prices.

The term that was price-projected using the U-PRICE model is the base price of uranium in LT contracts signed in any given year. The projected base price is not identical to the average delivery price of that year because the average delivery price is an average of delivery prices derived from contracts signed at different points in time. The average delivery price also includes prices under market price contracts.

19.3.1 Composite LT Base Price: Probability Analysis

Similar to the method used to derive the composite spot price, the composite LT base price is defined as a probability-weighted average of the three prices. Table 19-4 summarizes the detailed forecasts of the three LT base price scenarios. For comparison purposes, Table 19-2 also includes the composite price scenario.

19.3.2 Forecasts

Consistent with UxC Mid LT Base guidance, commodity price forecasts used in the financial model in Section 22 assume the following.

- Uranium price of US$50/lb U_3O_8 based on LT forecasts, and the net of YC transportation fees (estimated at approximately US$0.50 per pound).
- One hundred percent of uranium sold at an LT price of US$50/lb U_3O_8.
- The projected exchange rate is C$1.00 = US$0.75.

Table 19-4: UxC Annual LT Base Price Projections, 2020–2035 (US$/lb U_3O_8)

Scenario	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
High Constant	$32.23	$37.00	$42.14	$46.41	$50.00	$52.81	$54.11	$54.30	$58.34	$61.70	$62.07	$65.02	$65.40	$65.05	$65.56	$66.38
Mid Constant	$30.20	$33.36	$35.84	$38.93	$40.39	$43.18	$43.52	$44.77	$47.21	$49.99	$51.34	$51.47	$53.69	$55.92	$56.27	$56.46
Low Constant	$29.02	$29.82	$30.62	$31.25	$31.25	$33.00	$34.29	$35.01	$36.58	$39.68	$41.79	$42.58	$43.07	$43.35	$43.83	$43.20

Note:
Table prepared by UxC, 2020.
Average price forecast for Mid Constant 2027 to 2035 $51.90 /lb

19.3.3 Term Contracts

LT contracts are an integral part of uranium market transactions, and many uranium producers rely on base-escalated prices, or at least have base-escalated floors if the mine is indexed to spot prices. While supplies in the spot market are largely driven by available inventories (including secondary supplies of uranium from enrichers and government agencies), LT contracts are mainly offered by uranium producers able to commit supplies for multiple years.

In this regard, the LT base price provides an indicator of future supply availability. In some ways, it is counterintuitive to use a spot price to determine future supply because it is an expression of today's inventory spot market dynamics.

The Ux U_3O_8 Price (Spot Price) represents the most competitive uranium price. This price considers a delivery timeframe of less than or equal to three months, a quantity of uranium greater than or equal to 100,000 lb U_3O_8, as well as origin considerations. The Ux LT U_3O_8 Price (LT Price) considers escalation, delivery timeframe greater than or equal to 36 months, and quantity flexibility (up to ±10%).

There are two general approaches to LT contract uranium pricing: floating (or market) pricing, and base-escalated (or fixed) pricing. Floating pricing typically references the prevailing market price (such as the Ux U_3O_8 Price [Spot Price]) as the delivery price at the time of delivery in the future; the contract is therefore based on the prevailing market price at the time of sale.

For fixed pricing, the base price is typically set at a premium of the then-prevailing spot price when the contract is signed; it is therefore subject to escalation or indexing for inflation, such as the GDP-Implicit Price Deflator (IPD) at the future time of delivery. In common practice. when determining contract prices, utilities often use a combination of these two pricing options to achieve cost minimization and price stability.

Floor prices in LT contracts are typically determined based on the production cost of uranium. In theory, production costs should be below the floor price. This ensures cost recovery (including a reasonable rate of return) for a producer.

Long Term Demand Over Next 12 Months

As presented in Figure 19-4, estimated term volume for 2020 is below that reported over the past six years. Previous years have also included sizeable term volumes attributed to a few utilities each year with sizeable awards. Moreover, the past two years have included some unique, larger-volume deals that included integrated fuel supplies for new reactors.

These types of awards are not included in current volume estimates. Therefore, if any unique, larger-volume deals are concluded, term volume could be pushed to new heights. However, given that most utilities are well covered over the next several years, fewer utilities may need to enter the term market over the next 12 months.

Figure 19-4: Utility LTC Volume Over the 1990-2020 Period

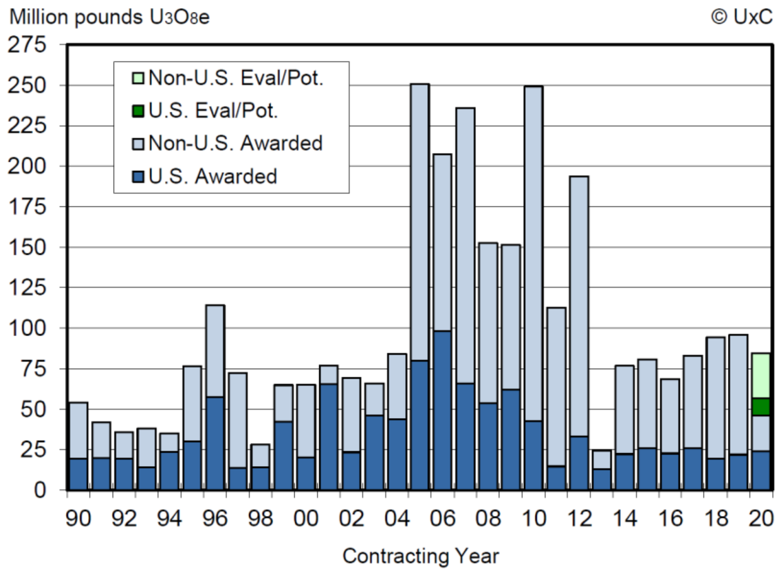


Figure prepared by UxC, 2020.

Also, with the volatility in this year's (2020) spot price, utilities may shift to mid-term contracting to cover nearer-term needs, while others could shift to lock in longer-term supply based on the view of future supply / demand balances and potential upward price movements. Some utilities had also been waiting for COVID-19 factors to dissipate before deciding to commence term procurement activities.

19.4 Contracts

NexGen is considering selling production from the Project via LT contracts with buyers. It is expected that such contracts would follow industry norms.

LT contracts have currently not been signed for the Project.

19.5 Comments on Section 19

Uranium price assumptions are based on LT forecasts. LT contracts have not been signed for the Project as of the effective date of this report.

The QP believes the information in this section regarding marketing and metal price forecasts is acceptable to support the financial analysis in Section 22.0.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

NexGen began collecting baseline data and information in 2015, with the majority of field studies commencing in 2018. Where necessary, some studies continued into 2019 and 2020 to complete the baseline data and information collection requirements, with some work ongoing into 2021. As of the effective date of this report, NexGen has enough data to complete a comprehensive EA. Results from NexGen baseline studies are summarized in Table 20-1.

As of the effective date of this report, several environmental and social baseline studies have been completed, and some are still ongoing. While the 2020 field programs have concluded, reports are not yet completed; therefore, the results of the 2020 NexGen baseline studies and programs were not available for use in this report.

Provincial permits and licenses for exploring and maintaining the work camp were approved in previous years; many of these permits have been renewed and are currently active.

The EA process for the Project is in progress as of the effective date of this report, and preparation of a Draft EIS is underway. Similarly, the CNSC licensing process is in progress. NexGen submitted a project description to the CNSC and ENV on 14 February 2019, and a revised project description was subsequently submitted in April 2019.

In accordance with REGDOC-3.2.2, Aboriginal Engagement (CNSC, 2016), NexGen also submitted a preliminary Indigenous Engagement Report (IER) on 14 February 2019.

The CNSC formally commenced the EA process on 02 May 2019 by issuing the Notice of Commencement, which served as an invitation for the public to comment on the project description. The CNSC Record of Decision (DEC 19-H112) regarding the scope of the EA for the Project was published on 20 February 2020.

As development of the Draft EIS and licensing applications are in progress, related findings—including any notable issues that could materially impact NexGen's ability to extract the Mineral Resources—are not yet available for inclusion in this report.

Furthermore, no recommendations from the EA or licensing processes for future monitoring and/or management of environmental and social aspects of the Project have been determined. Therefore, any considerations regarding specific monitoring and management plans are not included in this report.

As of the effective date of this report, NexGen has not applied for mine development licenses, permits and/or authorizations. A summary of both federal and provincial regulatory requirements for the EA phase of the Project are included in Section 20.7.

NexGen introductions to local communities commenced in 2013, and NexGen's relationships with these communities have evolved since the submission of the 2018 NI 43-101 report. A summary of NexGen's Indigenous and community engagement activities from 2013–2020 is provided in Section 20.8; Section 20.8 also discusses a conceptual Indigenous and community engagement plan for 2021.

20.1 Baseline Studies

NexGen conducted baseline studies to gather information regarding the existing conditions for the biophysical, cultural / heritage, and socioeconomic components of the Project. Some baseline studies have been completed, while other studies are currently in progress. A summary of studies that were completed from 2017 to 2020 is provided in Table 20-1, including the purpose of each study, relevant background information, and a brief description of the findings. Findings from 2020 baseline studies that are still in progress or are not yet finalized have been omitted.

All baseline studies, regardless of technical discipline, have the following objectives.

- Establish and characterize the existing conditions of the area, both within a local study area and a regional study area.
- Provide a baseline against which potential effects from the Project can be assessed.
- Help identify potential areas of concern or interest from local communities regarding the Project.
- Help with early identification of monitoring and management plan requirements.

Other discipline-specific baseline study objectives are described in Table 20-1.

The baseline studies form part of the comprehensive EA. As the EA development is ongoing, potential effects of proposed project activities have not yet been identified. Identifying potential impacts of the Project and determining their level of significance are key objectives of the EA, and they will be concluded in the EIS.

NexGen will be required to mitigate the negative impacts of project activities as much as practicable. Mitigations proposed are required to reflect industry standard techniques that have been recognized and approved by regulatory agencies on other projects that have taken place in similar climates, terrains, and conditions. Project-specific monitoring requirements will be determined during the EA process, and confirmed in the conditions of the Project's EA approval, as well as in various permits that will be required by both federal and provincial jurisdictions.

The baseline study results to date have not identified any known environmental or social issues that could materially affect NexGen's ability to move forward with the Project.

Table 20-1: NexGen Baseline Study Results

Study	Purpose and Study Design	Background Information	Findings
Air Quality and Climate Reference: Golder (March 2020)	**Purpose** The atmospheric baseline program was undertaken to complete the following objectives: • Identify the meteorological conditions that will influence dispersion of air emissions. • Describe the historical and existing air quality. **Study Period** Baseline air quality and climate monitoring at the project site has been ongoing since 2018. The baseline study report will include data through to 2020. **Support of the EA** The identified historical and existing meteorological and air quality conditions will be used to contribute to the assessment of potential effects of the Project on air quality. The meteorological baseline data will be used to validate the meteorological inputs to the air dispersion model. The baseline data will be used by other disciplines in support of their assessments.	• Meteorological information regarding temperature, precipitation, wind speed and direction, relative humidity, and solar radiation was available from studies completed within the anticipated project area. • There are limited long-term meteorological and air quality monitoring data available for the local study area; therefore, surrogate stations outside of the local study area were used to support the characterization of existing conditions. The characterization from the surrogate stations was considered representative of the conditions within the local study area. • Long-term measurements of temperature, precipitation, wind speed and direction, and relative humidity were available from two regional long-term climate monitoring stations. • Baseline air quality parameters included: particulate matter (PM), nitrogen dioxide (NO_2), sulphur dioxide (SO_2), dustfall (including metals), metals deposition, and radon. • Regional air quality monitoring at several remote locations was used to supplement data collected in the local study area for completion of the characterization of baseline conditions. • Baseline data were compared to air quality standards, where applicable.	• Although 2020 data have been collected, they have not yet been integrated into findings. • There were no exceedances recorded during the study for total suspended particulates (TSP), with concentrations remaining well below the provincial standard. • The baseline data were generally below both the 24-hour and annual Saskatchewan Ambient Air Quality Standards (SAAQS), except for some events which were likely due to forest fires. • Local and regional NO_2 and SO_2 baseline data generally indicate concentrations below annual ambient air quality criteria for each compound. • Monthly dust fall deposition measured in the anticipated project area were below the residential and industrial dust fall guidelines from Alberta Environment and Parks (2019), which were used as surrogate criteria in the absence of a Saskatchewan dust fall standard. • Most of the metals analyzed from dust fall were not observed above the lower detection limit (LDL). • Ambient radon exposure and concentrations in the anticipated project area were generally below the reportable detection limit (RDL). The radon concentrations measured were within the range of background concentrations in Canada (WHO, 2016), and were low compared to previous studies for outdoor radon concentrations in Saskatchewan.

Study	Purpose and Study Design	Background Information	Findings
Aquatic Resources References: CanNorth (2019a, 2019b)	**Purpose** Baseline studies were conducted to complete the following objectives: • Characterize water quality, local, and regional aquatic fauna. • Provide context for which potential effects to the aquatic ecosystems caused by the Project should be assessed in the EIS. **Study Period** Baseline aquatic studies commenced in 2018 and were completed in 2020; however, the 2020 field season results are currently being finalized. **Support of the EA** Water, sediment, aquatic macrophyte, fish habitat, and fish tissue chemistry studies provided data that will be used for the following: • Assessment of potential effects of the Project on water quality, aquatic flora and fauna, and aquatic ecosystems. • Environmental risk assessment. • Predictive modelling as part of the EA.	• Aquatic baseline studies included studies regarding lake morphometry, water and sediment quality, plankton and benthic invertebrate communities, aquatic macrophyte chemistry, fish spawning and habitat, and fish tissue chemistry. • The study areas included waterbodies in close proximity to the proposed mine infrastructure, situated downstream of the proposed treated effluent discharge location in Patterson Lake, and far-field from the site that can be used as reference areas once the mine is operational.	• Water quality in the study areas contained adequate dissolved oxygen (for all biota in general conditions), near-neutral pH, and low levels of nutrients, ions, metals, and radionuclides. • Fish diversity was abundant in the study areas, with northern pike found in all sampled waterbodies, along with many other species such as lake whitefish, walleye, yellow perch, and lake trout. There were no fish species at risk identified. • The background iron levels exceeded the Saskatchewan Environmental Quality Guidelines (SMOE, 2014) in some waterbodies. • Mean concentrations of arsenic, vanadium, and polonium-210 exceeded guidelines in some sediment samples. • Several metals and radionuclides in Patterson Lake were higher than in other sampled areas; however, metal and radionuclide concentrations in northern pike and lake whitefish flesh, and bone samples were frequently below laboratory detection limits. • The fish winter habitat surveys completed at the proposed freshwater intake and treated effluent discharge locations in Patterson Lake indicated that these areas would provide suitable overwintering habitat for large-bodied fish and forage fish.

Study	Purpose and Study Design	Background Information	Findings
Geomorphology Reference: Golder (July 2019)	**Purpose** The purpose of the 2018 geomorphology study was to characterize existing geomorphology of lake outlet channels and shorelines that could be influenced by the Project. The study focused on Patterson Lake, and the Clearwater River outlet of Patterson Lake to Forrest Lake. **Study Period** The baseline geomorphology study commenced and was completed in 2018. **Support of the EA** The baseline study will help identify areas that may be of concern for sediment and erosion as a result of Project activities. The study provides a basis to help identify future monitoring programs and site-specific mitigations in the EA, to manage flows and reduce potential for erosion and channel stability during Project phases.	The study included a desktop review, field data collection, and preliminary analysis for potential erosion susceptibility.	• Several shoreline segments along Patterson Lake were identified as being potentially sensitive to erosion and sediment transport processes such as accretion resulting from longshore drift and ice-thrust. These areas are expected to be most sensitive to changes in the lake hydrologic regime. • The surveyed reach of the Clearwater River between Patterson Lake and Forrest Lake has an active sediment transport regime capable of transporting mostly fine- to medium-sand-size materials.
Heritage Resources References: Bison Historical Services Ltd. (October 2018), CanNorth (October 2018)	**Purpose** The purpose of the heritage resource survey was to determine the presence of any culturally or archaeologically significant sites or artifacts within the vicinity of the Project. **Study Period** A heritage resource survey was completed in 2018 and submitted to the Heritage Conservation Branch. **Support of the EA** As no heritage resources were identified in the study area, there is no requirement to include this topic in the EA.	The local study area included the northern shore of Patterson Lake; a large, level upland area; and the southern shore of Patterson Lake.	• No heritage resources were identified within the local study area. • The Saskatchewan Heritage Conservation Branch confirmed in a letter to NexGen (November 2018) that all Heritage Resources Impact Assessment regulatory requirements had been met and that the province has no concerns regarding the Project proceeding on this basis.

Study	Purpose and Study Design	Background Information	Findings
Hydrogeology Reference: Golder (November 2020)	**Purpose** The purpose of the baseline hydrogeology monitoring study was to collect data to characterize typical spatial and temporal variability within the groundwater system to support both the EA and the design of future monitoring programs. **Study Period** Baseline groundwater monitoring data at the site have been collected since 2017 and up until the end of Q3 in 2020 for the purposes of the EA; data were collected on a quarterly basis. **Support of the EA** The baseline study will help identify areas that may be of concern for groundwater as a result of Project activities. The study provides a basis to help identify future monitoring programs and site-specific mitigations in the EIS to manage potential groundwater issues.	Baseline data for groundwater quality findings included physical and visual properties, ionic concentrations, dissolved and total metals, nutrients and organic compounds, isotopes, and radionuclides.	• In general, the groundwater within the bedrock is predominantly of calcium-chloride type. The groundwater within the surficial glacial deposits is predominantly of magnesium-bicarbonate type. The other water types identified included sodium-chloride, mixed sodium / calcium-chloride, and sodium-bicarbonate. • While the groundwater chemistry data have yet to be examined for trends, it is reasonable to consider that sufficient data have been collected to identify seasonable variability in water quality, if such variation exists. • Future monitoring of the baseline is under consideration, which would further establish the existing characterization of the groundwater, particularly seasonal trends.

Study	Purpose and Study Design	Background Information	Findings
Hydrology References: Golder (February 2019), Golder (August 2019)	**Purpose** The purpose of the hydrology study was to characterize existing hydrological conditions for local and regional water bodies and water courses. This included an assessment of the hydrological and climate variability in the region. **Study Period** Baseline studies commenced in 2018 and are ongoing as of 2021. **Support of the EA** The baseline hydrology conditions will: • Provide context for the assessment of potential effects on hydrological conditions resulting from the Project. • Assist in the development of a water balance for Patterson Lake to be used in the EA and to support Project decision making. • Help establish a framework for future monitoring, if necessary.	The studies focus on the Clearwater River watershed to the Naomi Lake outlet, and the Clearwater River watershed above confluence with the Mirror River. Studies that were conducted included the following: • Meteorological data collection to characterize seasonal variations. • Snow surveys to determine snow accumulation available for melt. • Hydrometric monitoring. • Sediment transport characteristics. • Mixing study of the North Basin of Forrest Lake. Meteorological and hydrological conditions during the monitoring period were compared to long-term records in the region. Site-specific observations and measurements included air temperatures, precipitation, seasonal water levels, snow accumulation, timing of spring freshet, and discharges and water level changes during rainfall events.	The hydrology baseline study provided the following information: • Characterization of seasonal and interannual fluctuations of water levels and discharges over the monitoring period. • Measurements of hydrological conditions in both wetter-than-normal and drier-than-normal years. • Characterization of sediment transport dynamics over a range of discharge conditions in the Clearwater River downstream of Patterson Lake. • Characterization of mixing between Clearwater River and north basin of Forrest Lake for a specific monitoring period. • Characterization of lake currents at select points in Patterson Lake during the monitoring period.
Light Reference: Golder (August 2020)	**Purpose** The purpose of the light survey was to complete the following objectives: • Characterize the existing local and regional light conditions. • Provide a baseline against which potential light effects from the Project can be assessed. **Study Period** Field studies were conducted in September 2018 and March 2020. **Support of the EA** Information from the light baseline studies will: • Provide context for the assessment of potential effects on light levels resulting from the Project. • Support the development of Project-specific mitigation strategies, if required.	• The focus of the study was on light trespass and sky glow levels. • In the absence of federal or provincial guidance, baseline light trespass and sky glow levels were characterized using thresholds from light assessment literature.	• Baseline light trespass levels for all nine measurement stations were observed to be less than assessment thresholds from light assessment literature (International Commission on Illumination [CIE] 2003). • Baseline sky glow for seven of the nine measurement stations was observed to exceed assessment thresholds from light assessment literature (Narisada and Schreuder 2004). The observed elevated sky glow is likely from the combined influence of aurora activity during the field study, and the presence of Fort McMurray approximately 180 km southwest of the Project.

Study	Purpose and Study Design	Background Information	Findings
Noise Reference: Golder (August 2020)	**Purpose** The purpose of the noise baseline study was to complete the following objectives: • Establish the existing acoustic environment. • Provide a baseline against which potential noise effects from the Project can be assessed. **Study Period** Field studies were conducted in September 2018 and March 2020. **Support of the EA** Information from the noise baseline studies will: • Provide context for the assessment of potential effects on noise levels resulting from the Project. • Support the development of Project-specific mitigation strategies, if required.	• The focus of the study was on baseline noise levels. • In the absence of Saskatchewan-specific guidance, baseline noise levels were characterized using assessment thresholds from Environment and Climate Change Canada, Health Canada, and the Alberta Energy Regulator (AER).	• Baseline noise levels for all three measurement stations were observed to be less than assessment thresholds set out in the Environment Canada, Health Canada, and AER guidance documents (Environment Canada 2009; Health Canada 2017; AER 2007). • Based on AER criteria, there were no low frequency noise (LFN) issues at any of the measurement stations.
Socio-Economic	**Purpose** The purpose of the socio-economic baseline study is to describe the existing socio-economic characteristics of the communities near the Project, including the local labour force and the economy, infrastructure and services, and overall community well-being. **Study Period** Data collection commenced in 2018 and is ongoing as of 2021. **Support of the EA** The characterization of the socio-economic status of the communities near the Project will help inform the understanding of potential socio-economic impacts and benefits discussed in the EA.	• Data collection occurred through desktop research, data from Statistics Canada, and key person interviews. • The key person interview program solicited detailed information from key representatives in the community, such as those with expertise in education, health care, social services, local businesses, and community leadership. People were interviewed to gain a better understanding of the conditions people live in and how the Project may affect them.	• The communities near the Project are predominately Indigenous, and include three First Nations communities (Buffalo River Dene Nation, Birch Narrows Dene Nation, and Clearwater River Dene Nation) and three northern villages / hamlets that are predominately Métis (La Loche, Buffalo Narrows, and Turner Lake). • Communities have an interest in the Project and its associated opportunities, including employment and contracting. Maximizing local participation may require support for training, education, and business capacity development. • Concerns such as transportation of materials through the region and environmental impacts to waterways were frequently noted through the key person interview program.

Study	Purpose and Study Design	Background Information	Findings
Terrain and Soils Reference: Golder (March 2020)	**Purpose** The purpose of the terrain and soils baseline program was to describe the existing terrain and soil characteristics at the project site, including landform classification, soil chemistry, and soil sensitivities. **Study Period** The terrain and soils baseline study commenced in 2018 and was completed in 2019. **Support of the EA** Information from the terrain and soils baseline program will provide context for the assessment of potential effects to terrain and soils resulting from the Project. The characterization of the terrain and soils will also help inform reclamation efforts during the construction, operations, and decommissioning Project phases.	• Soil chemistry was compared against Soil Quality Guidelines for the Protection of Environmental and Human Heath (Canadian Council of Ministers of the Environment 2014). • Radionuclide soil samples were compared to Canadian Guidelines for the Management of Naturally Occurring Radioactive Materials (Canadian NORM Working Group 2013).	• There are three terrain areas in the anticipated project area, including upland landscape positions for well-drained soils; depressional landscape positions for very poorly drained soils; and transition landscape positions (between upland and wetland positions). • Soil inspections indicate that the project site predominantly consists of loamy sand textured soils. • Soil chemistry indicates that concentrations of metals that were analyzed did not exceed the upper limits of the Soil Quality Guidelines. • Radionuclide analysis of soils samples collected in 2019 had no detectable levels of lead-210, thorium-228, thorium-230, or thorium-232. Polonium-210 and Ra-226 were detected at the sites. None of the radionuclide values exceeded the upper limits of the Canadian Guidelines.
Traditional Land and Resources Use	**Purpose** The purpose of traditional land and resource use studies was to describe the holistic, cumulative, dynamic, and intergenerational knowledge of land and resource use understood by the Indigenous peoples in the area. **Study Period** Data collection commenced in 2018 and is ongoing as of 2021. **Support of the EA** The information collected and provided by Indigenous communities will: • Facilitate the understanding of existing uses in the region, as well as the cultural context. • Support the integration of Indigenous knowledge into the EA.	• Information regarding traditional land and resources use was supported by community-led studies that characterized the historic and current uses of the area and the importance of these activities to the communities' well-being and culture. • Additional data collection regarding commercial and recreational uses in the region was supported through desktop research and key person interviews.	• Currently, there is a diversity of traditional and modern resource activities throughout the northwestern region of Saskatchewan. • Traditional land uses in the region include hunting, trapping, fishing, and gathering. These activities support cultural expression in the communities, which is integral to community well-being. Opportunities to spend time on the land are considered highly important to communities. • Commercial and recreational uses in the region include commercial fishing, trapping, lodges, and outfitting. Many of these activities are important contributors to the local economy.

Study	Purpose and Study Design	Background Information	Findings
Traffic Reference: InterGroup (November 2018)	**Purpose** The purpose of the baseline traffic study was to characterize the existing traffic volumes and trends on highways leading to the project site, specifically Highway 155 and Highway 955. **Study Period** The traffic study was conducted in 2018. **Support of the EA** Characterizing the existing traffic volumes and trends will contribute to the understanding of how the Project may affect these conditions during all Project phases.	• The regional traffic overview study was a desktop review of data from the Saskatchewan Highways and Infrastructure for the year of 2016. • The study describes the 2016 average annual daily traffic, with a focus on truck traffic along Highway 155 and Highway 955. Traffic-related accidents were also included in the study. • Highway 155 is an all-weather paved highway. Highway 955 is an all-weather highway that is almost entirely unpaved with the exception of approximately 4.5 km.	• Overall, Highway 155 had higher traffic volumes than Highway 955. Highway 155 has three large service centres along its route, while Highway 955 does not have any service centres. • The portion of Highway 155 with the highest average annual daily traffic (1,510) is found immediately south of the community of Buffalo Narrows. The portion with the highest average annual daily traffic on Highway 955 (2,000) is located in the town of La Loche. • Highway 155 had more traffic accidents as compared to the Saskatchewan average on provincial highways. It also had a greater collision rate than all rural municipalities. Fifty-eight collisions causing property damage were reported, along with 11 collisions causing personal injury and two collisions causing fatalities. In total, 22 injuries and three deaths were reported. • Highway 955 had fewer traffic accidents when compared to the Saskatchewan average on provincial highways. It also had fewer collisions than the collision rate for all rural municipalities. In 2016, seven collisions causing property damage were reported and no collisions causing personal injury or deaths. No injuries or deaths on the highway were reported.

Study	Purpose and Study Design	Background Information	Findings
Vegetation References: CanNorth (2019, 2020), Omnia (2019)	**Purpose** The purpose of the vegetation baseline studies was to complete the following objectives: • Collect information regarding vegetation and wetlands occurring in close proximity to the site. • Document the ecological land classification and anthropogenic disturbance, including fire disturbance. **Study Period** Baseline vegetation studies commenced in 2018 and were completed in 2019. **Support of the EA** Information from the baseline vegetation studies will: • Provide context for the assessment of potential effects to vegetation resulting from the Project. • Support the development of Project-specific mitigation strategies. • Help establish a framework for future environmental effects monitoring.	• A list of plant species with conservation concerns was compiled from a database search. The list was supplemented with data collected through terrestrial and aquatic vegetation inventory surveys. • Sensitive plant species are identified and tracked by the Saskatchewan Conservation Data Centre.	• A total of 114 plant species were identified in the terrestrial and aquatic vegetation inventory surveys. • The dominant habitats on the project site consist of regenerating and recently-burned jack pine stands. • Other vegetation communities present include wetlands and moist, mixed-wood / deciduous forests. • Eight provincially tracked vascular plant species were identified. An additional 22 provincially tracked moss and lichen species were also detected. No federally listed at-risk species were detected. • Wetland classifications identified a total of 15 wetlands in the local study area of the Project.

Study	Purpose and Study Design	Background Information	Findings
Vegetation Chemistry Reference: Golder (March 2020)	**Purpose** The vegetation chemistry baseline study was undertaken to establish the existing concentrations of metals and radionuclides in selected plant species (lichen and blueberry). **Study Period** The vegetation chemistry baseline study was conducted in 2019. **Support of the EA** The baseline study will provide context to help determine which potential effects from the Project on vegetation, wildlife, and people will be assessed in the EA.	• Lichens were chosen because they are estimated to account for approximately 90% of the diet for caribou. Caribou are a federally-listed species, important to Indigenous people and government regulators, and they are a valued component for the Project wildlife assessment. • Lichens also effectively bioaccumulate airborne contaminants; this provides a conservative scenario for the assessment of risks to caribou. • Blueberry was selected to represent local and Indigenous use of plant resources. Blueberry samples include stems, leaves, and fruit. • Samples were analyzed for 30 different metals, and four radionuclides (lead-210, polonium-210, Ra-226, and thorium-230). • There are no current guidelines for concentrations of metals or radionuclides for lichen or vascular plants to compare to the field data. However, pre-Project disturbance samples were obtained from reference sites and will form the baseline for evaluating changes in metals concentrations in vegetation during Project construction and operation, if required.	• A high degree of variability was observed for several of the metals and radionuclides across plant tissue types. • Arsenic, beryllium, cadmium, molybdenum, selenium, silver, thallium, and tin were observed at or near the non-detect level across the majority of plots. • Radionuclide concentrations were generally observed to be most elevated in lichen tissue samples for lead-210, polonium-210, and thorium-230, and in blueberry stems for Ra-226.

Study	Purpose and Study Design	Background Information	Findings
Wildlife References: CanNorth (2019a, 2019b), Omnia (2019)	**Purpose** The wildlife monitoring studies were conducted to document information regarding select wildlife occurring in close proximity to the project site. **Study Period** Baseline studies were conducted in 2018 and 2019. Avian and bat monitoring studies continued in 2020. **Support of the EA or Permitting** Information from the wildlife baseline studies will: • Provide context for the assessment of potential effects to wildlife resulting from the Project. • Support the development of Project-specific mitigation strategies. • Inform predictive modelling used in the EA.	• The studies included desktop literature review and extensive field programs. • The baseline studies focused on amphibians, birds, bats, waterfowl, terrestrial and semi-aquatic furbearers, small mammals, and boreal woodland caribou.	• Most of the bird species detected during the baseline studies are protected under the Migratory Bird Convention Act and/or the provincial Wildlife Act, and several species with conservation concern were identified including four species listed under the Species at Risk Act (SARA). • Seven additional species with provincial activity restriction guidelines were observed during the breeding bird surveys. • A total of 19 sensitive species were detected during targeted field surveys and incidental observation opportunities. Six of these species are listed federally under SARA, along with boreal woodland caribou (threatened), and bat species (potentially two species listed as endangered). • A second baseline study, conducted independent of the first, identified 13 different sensitive or at-risk wildlife species during field surveys.

20.2 Requirements for Waste Rock and Tailings Management

Management of waste rock and tailings during site preparation, construction, operation, and decommissioning of a new uranium mine or mill is a regulatory requirement by the CNSC. Waste rock and tailings management general requirements are described in the Uranium Mines and Mills Regulations (Section 3[c]), under the NSCA. Regulatory document REGDOC-2.11.1, Waste Management, Volume II: Management of Uranium Mine Waste Rock and Mill Tailings (CNSC, 2018) details the CNSC requirements and expectations for the management of waste rock and tailings for all mine phases.

As per REGDOC-2.11.1, NexGen will require a documented plan for managing waste rock and tailings. REGDOC-2.11.1 also outlines CNSC's expectations regarding the mine waste management options. The license application for the uranium mine and mill will require the results of the EA and a description of the waste rock and tailings management plans.

The FS project design and details regarding the UGTMF plan are discussed in Sections 16.0 and 18.0.

20.2.1 Ore and Special Waste Stockpiles

There will be an ore stockpile on-site with four piles of differing grades. Each pile will have a capacity of approximately 6,500 m^3.

It is estimated that about 1% of the waste rock brought to surface will be mineralized, but will not be a high enough grade to warrant being processed through the mill. Therefore, it will not be stockpiled in the raw ore stockpile area; this material will instead be stored in the special waste rock stockpile area, which will have an anticipated pile volume of 60,000 m^3.

Both the ore and special waste stockpiles will be dual-lined with HDPE, and will be self-contained facilities capable of withstanding a full PMP 24-hour event.

20.2.2 Waste Rock Storage Facilities

Approximately 5.9 Mm^3 of waste rock will be generated over the course of the LOM. Of this total 4.59 Mm^3 (78%) is potentially acid generating (PAG) and 1.33 Mm^3 is non-PAG (NPAG). The PAG and NPAG waste rock will have separate storage areas. The PAG and NPAG waste rock will be stockpiled with 2:1 side slopes and the top of the finished stockpile will tie into the hill to the south and the overall height will not exceed the highest nearby topography.

Figure 20-1 shows the waste rock storage facility layout included in the mine design for the Project.

Figure 20-1: Waste Rock Storage Facility Layout Plan

20.2.3 PAG Waste Rock Storage Area

The PAG WRSA will be lined with HDPE. Clearing and grubbing will be required in the area, and personnel will need to prepare to install an HDPE liner, including sand bedding and crushed rock cover to protect the liner.

All precipitation in the PAG WRSA will be captured and diverted to the PAG runoff collection area, which will be located at the topographical low point. Water management infrastructure for the PAG WRSA will comprise a perimeter berm and/or collection ditches. The runoff collection area, like the WRSA base, will be HDPE-lined.

The PAG WRSA and runoff collection area must be able to capture and retain all precipitation from a PMP 24-hour event; it will therefore require a retention capacity of 141,670 m^3, accounting for 1 m of freeboard. Captured runoff will subsequently be piped via an overland HDPE pipeline to the settling pond.

20.2.4 NPAG Waste Rock Storage Area

The NPAG WRSA will not be lined with HDPE. Water management infrastructure for the NPAG WRSA will comprise a perimeter berm and/or collection ditch. Precipitation in the NPAG WRSA will be captured and diverted to site runoff pond #2.

20.3 Underground Tailings Management Facility

The UGTMF is discussed in Section 16.0.

20.4 Requirements for Water Management

20.4.1 Applicable Requirements

General requirements and expectations related to surface water management are listed in REGDOC-2.9.1: Environmental Protection: Environmental Principles, Assessments and Protection Measures, Version 1.2 (CNSC, 2020). Water management infrastructure will be designed to divert non-contact surface runoff water away from the site. Water on-site and in the WRSAs will be captured and controlled.

As per REGDOC-2.9.1, NexGen will be required to have a water management plan.

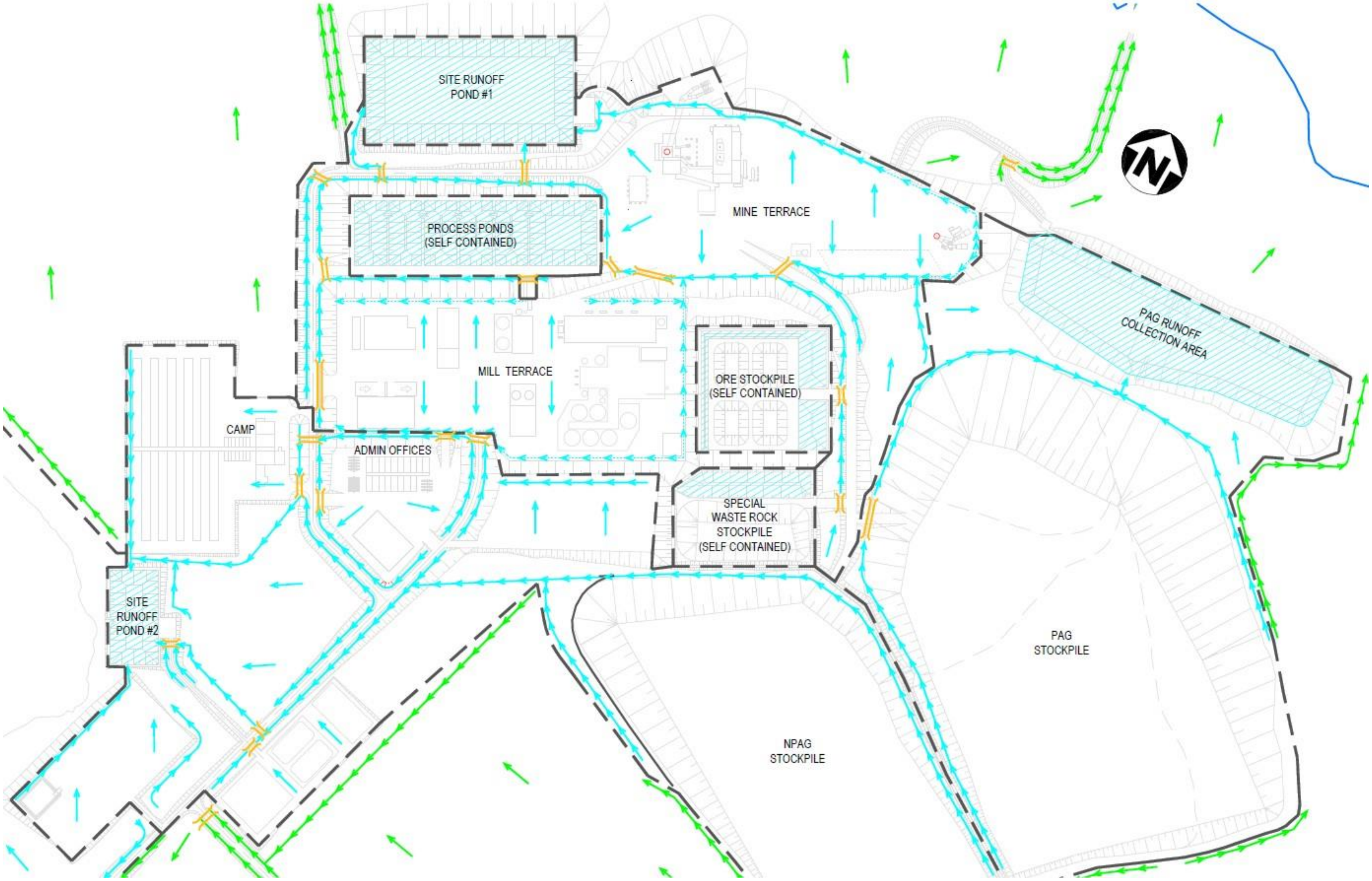
As of the effective date of this report, a site-wide water balance and water quality model for the Project is in development. The model will be used to support the development of a water management plan, evaluate engineering design decisions, and support the assessment of the effects of Project activities on the environment.

20.4.2 Water Management

Water management infrastructure will be designed to meet the requirements of the Environment and Climate Change Canada's Code of Practice for Metal Mines (ECCC 2009) and the Saskatchewan Construction Guidelines for Pollution Control Facilities at Uranium Mining and Milling Operations (SERM 2000).

The site surface drainage network will either collect or divert water. Where practical, all reasonable efforts will be made to divert non-contact site surface runoff away from any developed features. Precipitation and snow melt runoff that come in contact with infrastructure areas or contact zones will be captured, collected, and diverted to impound areas identified as site runoff ponds, or to collection areas. These impound areas are highlighted with blue cross-hatching in Figure 20-2.

Figure 20-2: Site Drainage Diversion and Collection Layout



Note: Figure prepared by Wood, 2020

From a surface drainage perspective, self-contained structures are defined as water management infrastructure that fully contain specific precipitation events. No other precipitation source is diverted to these structures, nor is the specific precipitation event diverted elsewhere until it is subsequently pumped to the settling pond.

Self-contained locations on-site will include the following.

- Process pond cluster of monitoring ponds, contingency pond, and settling pond.
- Ore stockpile area.
- Special waste rock stockpile area.
- PAG runoff collection area.

The remainder of the site will incorporate water management features designed to capture water from zones that are not within the self-contained areas. This water will be collected and diverted to one of the two site runoff ponds.

The primary criteria (i.e., the specified design precipitation event) for the two capture zones of the site runoff ponds differ depending on the specific anticipated precipitation event. However, both site runoff ponds will have the following features in common.

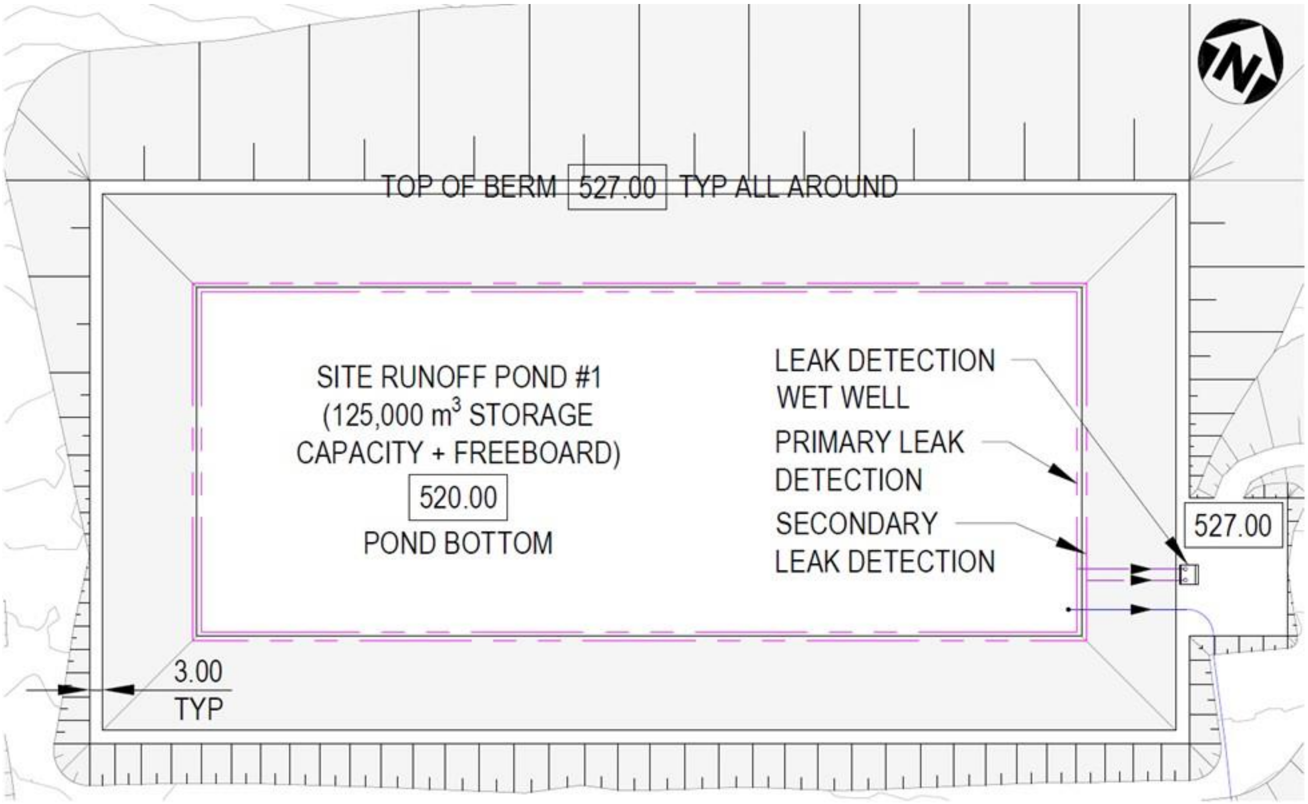
- Dual 80 mil HDPE-lined ponds with primary liner and secondary liner containment.
- Dual leak detection for the two-liner system, with primary leak detection and secondary leak detection.
- Leak detection monitoring wells on the top edge of the pond.
- One-metre freeboard above the pond high water level (HWL) containment capacity.
- Adjacent graded pad for pond access and intake sump pump deployment.

Site Runoff Pond #1

The following features are specific to site runoff pond #1 (see Figure 20-3 for a schematic of the pond).

- Capacity of 125,000 m^3, plus 1 m freeboard
- Capture zones with potential contact with mineralized or radiologically contaminated material. Major contributors will be the ore stockpile, special WRSA, and PAG runoff collection area.
- Capture zone collection and retention of a PMP 24-hour event.
- Draw-down achieved via sump pump intake that diverts collected precipitation to the site settling pond for re-use or treatment.
- Pond depth of 7 m overall, and an operating depth of 6 m to pond HWL.

Figure 20-3: Site Runoff Pond #1



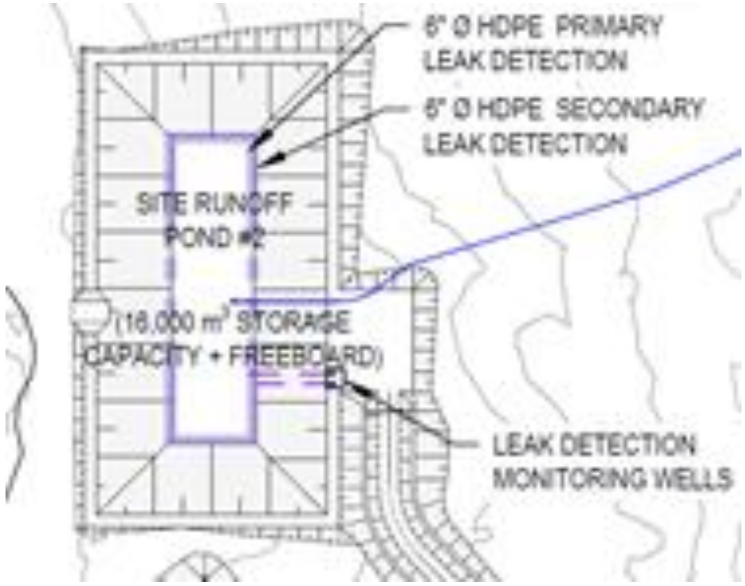
Note: Figure prepared by Wood, 2020

Site Runoff Pond #2

The following features are specific to site runoff pond #2 (see Figure 20-4 for a schematic of the pond).

- Capacity of 16,000 m^3, plus 1 m freeboard.
- Capture runoff area collection and retention of a 1:100 year 24-hour precipitation event. Sized to capture a PMP 24-hour event for precipitation directly intercepted by the pond itself.
- Pond contents will be tested, and if they are suitable for release, they will be discharged to the environment through the west bermed collection area. If the contents are not suitable for release, contained water will be pumped to the site settling pond.
- In the case of a PMP 24-hour precipitation event, the runoff pond will capture and collect runoff to site runoff pond #2 prior to diverting the remaining additional precipitation to the west bermed runoff collection area.
- The site runoff pond #2 overflow weir will help control overflow when maximum pond capacity is reached.

Figure 20-4: Site Runoff Pond #2



Note: Figure prepared by Wood, 2020

The west bermed runoff collection area will receive runoff from the local contributing area. This area will also receive overflow from the site runoff pond #2 in the case of a PMP 24-hour precipitation event.

The west bermed runoff collection area will be benefitted by a local natural low that will provide substantial storage capacity through the construction of a berm on the north side of the low area. This containment area will be used for precipitation events beyond the design capture events for the site. Its purpose will be to prevent suspended solids in the runoff from entering Patterson Lake. The area will not be lined to allow for natural dissipation of suspended solids.

Site ditching, either diversion ditching or collection ditching, will be sized to accommodate the specific precipitation event predicted for the respective capture area. Where flow upset conditions will likely result in a release to the environment, primary (i.e., critical) ditches will be sized to accommodate a full PMP 24-hour precipitation event. Secondary (i.e., non-critical) ditches will be sized to accommodate a 1:100 year 24-hour precipitation event.

To maintain ditch integrity, both diversion ditching and collection ditching will be constructed with erosion control measures reflective of ditch slopes and flows rates, where required.

Site swales will be situated on surface graded pads where ditches are not possible, and where the initial anticipated contributing precipitation does not warrant a full ditch.

The storage areas for mineralized ore and mineralized / special waste rock will be dual-contained with HDPE liners, and will be used to store water from a PMP 24-hour event. These areas will contain separate leak detection pipe under-drain systems.

Six contact water storage ponds are planned, including four monitoring ponds for treated effluent, one contingency pond, and one feed settling pond. Each monitoring pond and the contingency pond will be sized for a capacity of 5,000 m^3, and will maintain 1 m of freeboard as contingency for a PMP 24-hour event.

The feed settling pond will have a capacity of 16,000 m^3, with 1 m freeboard. Approximately 1,100 m^3 of the feed settling pond capacity will be reserved for a 1:100 year 24-hour precipitation event, which will include runoff that collects around the Production Shaft and in the pipe containment corridor.

All ponds will be double-lined with a HDPE liner, and will have 300 mm of sand between both layers.

All other water conveyance and containment structures have been designed to accommodate a 1:100 year, 24-hour storm event, and the anticipated volumes of water generated under routine and non-routine operating conditions.

20.5 Site Monitoring

Requirements and expectations related to environmental monitoring and management of the Project generally fall under the NSCA regulations.

Project-specific monitoring requirements will be determined during the EA process, and confirmed in the conditions of the Project's EA approval, as well as in various permits that will be required by federal and provincial jurisdictions. These monitoring requirements will be documented in the EA follow-up monitoring program that will be developed for the Project.

The EA follow-up monitoring program will be required to be consistent with CSA Standards N288.4-10 (Environmental Monitoring Programs at Class I Nuclear Facilities and Uranium Mines and Mills [CSA Group, 2010]), N288.5-11 (Effluent Monitoring Programs at Class I Nuclear Facilities and Uranium Mines and Mills [CSA Group, 2011]), and N288.7-15 (Groundwater Protection Programs at Class I Nuclear Facilities and Uranium Mines and Mills [CSA Group, 2015]), as applicable. NexGen is also planning to seek input from the public and local Indigenous communities in the development of the EA follow-up monitoring program.

As part of licensing requirements for the Project, NexGen will be required to develop an Environmental Protection Program (EPP) that outlines measures to protect and monitor the environment for radiological and non-radiological contaminants. The EPP will be supported by the following plans.

- Groundwater Protection Plan
- Air Protection Plan
- Wildlife and Biodiversity Protection Plan

20.6 Closure Plan

20.6.1 Regulatory Requirements

Prior to the completion of mining and milling activities, a detailed decommissioning plan will be required in accordance with the provincial and federal requirements listed at the beginning of this section. Once finalized, the plan and an application for approval to decommission will be submitted to provincial and federal authorities. Following approval, decommissioning activities will commence.

Mine closure requirements (i.e., regarding decommissioning and reclamation) are listed in the following documents.

- NSCA, Uranium Mine and Mills Regulation, Section 3(a)(viii).
- CNSC's Uranium Mines and Mills Regulations REGDOC-3.5.1 Licensing Process for Class 1 Nuclear Facilities and Uranium Mines and Mills (Version 2) (CNSC, 2017).
- CNSC's REGDOC 2.11.2 Decommissioning (CNSC, 2019).
- The Mineral Industry Environmental Protection Regulations, 1996 Section 14(1) (Government of Saskatchewan, 1996).
- Saskatchewan EPB 381 Northern Mine Decommissioning and Reclamation Guidelines (Government of Saskatchewan, 2008).

Closure requirements include the following.

- Plan for decommissioning and reclaiming the mine site.
- Description of the proposed methods and procedures and time frames for monitoring the mine site for physical and chemical stability, and detecting the release of pollutants during and after decommissioning and reclamation.
- Preliminary estimate of the cost required to carry out the closure plan, and the cost of monitoring the mining site after decommissioning and reclamation, in accordance with provincial requirements,
- Proposal for an assurance fund to ensure completion of the closure plan.
- Proposal for the management and administration of the assurance fund.
- Proposal respecting the release of all or portions of the assurance fund during decommissioning and reclamation of the mine site.

Decommissioning activities and the achievement of end-state objectives will be completed prior to final closure of the site. An application to abandon the property will be required by provincial and federal authorities before the site is transferred back to the province through the Institutional Control Program, in accordance with The Reclaimed Industrial Sites Act and The Reclaimed Industrial Sites Regulations.

20.6.2 Decommissioning Process

Orderly cessation of operations and transition of the operation into a safe inactive state will be completed prior to decommissioning. Production mining will be completed, and active mining areas backfilled and secured. The mill processing circuits will be systematically shut down, flushed, and cleaned. Surface facilities, infrastructure, and equipment will be cleaned, scanned, and prepared for decommissioning.

Wherever practical, surface, and UG infrastructure, and equipment and materials not required during the decommissioning phase that meet radiological criteria for off-site removal will be salvaged, sold, or transferred off-site for recycling or disposal. Remaining infrastructure, equipment, and materials will undergo final decommissioning on-site.

Surface Infrastructure

Surface infrastructure, equipment, and materials identified for on-site decommissioning—including the mill, mine surface, and ancillary facilities—will undergo sequential demolition starting with those not required to support decommissioning. Surface infrastructure, equipment, and materials will be demolished, staged, and transferred UG, where they will be treated as backfill during mine decommissioning.

Any remnant ore or special waste stockpiled on surface will be returned to the mine, and backfilled along with associated liners, berms, and fill material. Surface and process ponds will be dewatered, and sediment, liners, and fill material will be transferred UG and backfilled in the mine. These areas will then be subject to testing and radiological surveys to ensure conditions meet established criteria.

Radiological surveys will be required for roadways, storage pads, building foundations, ditches, berms, and other earthworks components prior to decommissioning. Materials that do not meet the requirements of decommissioning criteria will be removed and backfilled as part of mine decommissioning, with areas to be re-surveyed until the criteria are met. These areas will then undergo contouring, scarification, and revegetation with approved native vegetation species.

A portion of the stockpiled NPAG will be used to backfill mine shafts and as supplemental fill where required for earthworks. The remaining NPAG stockpile will be contoured in place to establish slope stability and appropriate drainage, covered with soil and organic material preserved during construction, and revegetated using approved native vegetation species.

The effluent treatment circuit and associated infrastructure will be retained until the final stage of decommissioning, when the mine is backfilled, and decommissioning of the first shaft is complete. This will allow for collection and treatment of water from the mine and surface facilities. A modular effluent treatment system will then be established, and the effluent treatment circuit and associated infrastructure will be demolished and backfilled in the second shaft, along with any remaining surface infrastructure and fill material.

Underground Workings

Mine decommissioning will occur in parallel with surface decommissioning. Designated surface materials will be transferred UG and backfilled into the lateral portions of the mine, along with mine infrastructure, equipment, and materials. Backfilled material will be placed in available space UG until all designated waste has been removed from surface. During this period, the mine will continue to be dewatered and ventilated, and required infrastructure will be maintained.

Shafts will be decommissioned sequentially following completion of backfilling in the lateral portions of the mine. The lower portion of the shaft (i.e., from the bottom of the shaft to bottom of the shaft liner) will be backfilled with remnant waste material and clean waste rock. A concrete plug will then be placed to seal the shaft below the bottom of the liner.

The remainder of the shaft (i.e., from the concrete plug to shaft collar) will be filled with clean fill material removed from berms, roadways, or other surface earthworks, and remaining stockpiled overburden retained from shaft sinking. Each shaft will then be sealed with a shallow, reinforced concrete plug at surface.

Mine dewatering and treatment will be maintained until the first shaft is decommissioned, at which time the water treatment system will be decommissioned and stored in the bottom portion of the second shaft during backfilling. All other openings to surface will be filled with a low conductivity, impermeable material, and will be sealed at surface during decommissioning.

The UGTMF will be progressively decommissioned during mine operation, and active decommissioning will not be required during the decommissioning phase. Facility performance and environmental monitoring criteria established during mine operation will be utilized to ensure UGTMF end-state objectives are achieved, and that processed waste is safe and stable prior to decommissioning access drifts and associated infrastructure.

20.6.3 Closure Costs

NexGen has allocated closure costs as part of the capital cost estimate in Section 21.1.

No closure or reclamation bonds are in place for future development or operations. No closure or reclamation bonding is required in support of the drill program discussed in Section 1.0.

20.7 Permitting

20.7.1 Federal and Provincial Regulatory Environmental Assessment Framework

Several federal and provincial regulatory approvals are required for all new uranium mine and mill development.

Federally, under the authority of the NSCA, proponents wishing to carry out mining and milling in Canada must first obtain a licence from the federal nuclear regulator (i.e., the CNSC). Before the CNSC can make a licensing decision, proponents are required to complete an EA of the proposed project. The federal Impact Assessment Act came into effect in August 2019; however, the Project commenced when the Canadian Environmental Assessment Act (CEAA, 2012) was in effect, and will continue to be reviewed under that law, as confirmed by the CNSC in a letter to NexGen on 29 August 2019.

According to CEAA 2012, EAs are required for designated projects, which are defined under the Regulations Designating Physical Activities (the Project List). The construction of a new uranium mine and mill is considered a designated activity, as defined by the Project List, and the CNSC is designated as the responsible authority for the oversight of the EA.

In Saskatchewan, new uranium mines and mills are subject to the Environmental Assessment Act. The Environmental Assessment Act requires the completion of an EA, and approval from the Saskatchewan Minister of Environment before proceeding. Developments that are approved are also required to obtain approval under the Environmental Management and Protection Act, 2010.

As the Project falls under both federal and provincial jurisdictions, the CNSC and ENV EA Branch will each require an EA prior to Project approval. The CNSC and ENV EA Branch are coordinating a harmonized EA under the guidance of the Canada-Saskatchewan Agreement on Environmental Assessment Cooperation (2005). This agreement reduces regulatory duplication and allows the regulatory agencies to work together to ensure that a thorough process is followed to limit environmental effects.

Throughout this collaborative process, the CNSC will act as the responsible EA authority and the lead agency overseeing the EA process. The CNSC will coordinate activities with the provincial and other federal agencies so that all regulatory considerations are taken into account in the review of the EA and the final decision.

20.7.2 Licensing and Approval Process

The proponent of a new uranium mining and milling project is required to obtain a license from the CNSC under the NSCA prior to development. NSCA regulations list the information applicants must submit to the CNSC as part of their license applications.

The CNSC's licensing process for uranium mines and mills typically follows the lifecycle of a project, and includes the following four general licensing phases.

- License to Prepare Site and Construct
- License to Operate
- License to Decommission
- License to Abandon

Under provincial legislation, the ENV is responsible for protecting and managing Saskatchewan's environmental and natural resources. Following completion of an EA, approvals under The Environmental Management and Protection Act, 2010 must be secured prior to construction, and adhered to throughout the project lifecycle.

The Mineral Industry Environmental Protection Regulations (MIEPR), 1996 govern the permitting and operation of mines and mills in Saskatchewan. Under the MIEPR, NexGen will require approval to construct, install, alter, or extend a pollutant control facility prior to commencement of construction. In addition, approval to operate a pollutant control facility will be required prior to mine operation, and will be subject to regular review and renewal throughout the life of the Project.

Another important approval requirement is the development and maintenance of a preliminary decommissioning plan and preliminary decommissioning cost estimate. Financial assurance will be required to cover the costs associated with executing the decommissioning plan, and the assurance will require approval by both the CNSC and the ENV.

20.7.3 Exploration Permits

NexGen will require the following provincial permits to support exploration drill programs and camp operations.

- Exploration drilling permits, including:
  - Aquatic habitat protection
  - Forest protection
  - Work authorization
- Industrial lease permit
- Water use permit

20.8 Community Engagement Requirements and Plans

20.8.1 Engagement Requirements

Public and Indigenous participation are important aspects of the EA process. Public and Indigenous engagement will be required to comply with the Government of Saskatchewan Proponents Guide: Consultation with First Nations and Metis in

Saskatchewan Environmental Impact Assessment (Government of Saskatchewan, 2014), CNSC REGDOC-3.2.1 Public Information and Disclosure (CNSC, 2018) and CNSC REGDOC-3.2.2 Indigenous Engagement (CNSC, 2019).

More specifically, NexGen will be required to meet requirements outlined in the Project's terms of reference. The following is an excerpt from Section 4.0 of the Rook I Project's Terms of Reference (April 2019).

> An overview of NexGen's Indigenous, public, and regulatory engagement plans will be provided in the EIS. In preparing the EIS, NexGen will demonstrate how it has engaged with Indigenous communities, the general public, and communities of interest that are likely to be affected by the proposed Rook I Project.
>
> This section will include a description of engagement activities, including documentation of meetings; discussion topics and outcomes; outstanding concerns and any relevant agreements. The EIS will discuss engagement activities with the federal and provincial regulatory agencies and will identify how NexGen will continue to engage with Indigenous communities and the public.

Further to these requirements, NexGen must consider the following regulatory guidance documents in its engagement approach.

- First Nation and Métis Consultation Policy Framework (Government of Saskatchewan 2010).
- Proponents Guide – Consultation with First Nations and Métis in Saskatchewan Environmental Impact Assessment (Government of Saskatchewan 2014).
- Proponent Handbook – Voluntary Engagement with First Nations and Métis Communities to Inform Government's Duty to Consult Process (Government of Saskatchewan 2013).
- Generic Guidelines for the Preparation of an Environmental Impact Statement pursuant to the Canadian Environmental Assessment Act, 2012 (CNSC 2016).
- Considering Aboriginal Traditional Knowledge in Environmental Assessments Conducted Under the Canadian Environmental Assessment Act, 2012 (Canadian Environmental Assessment Agency 2015).

20.8.2 Engagement Approach

Since exploration on the Project began in 2013, NexGen has engaged regularly and established relationships with local communities and Indigenous groups. NexGen's objectives when engaging include the following.

- Build sustainable relationships based on mutual trust and respect.
- Communicate clearly with using appropriate language and agreed-upon formats and platforms.
- Provide timely and accurate information regarding the Project, including information about potential environmental effects for all phases.

- Understand how the proposed development of the Project may impact Indigenous and/or treaty rights.

NexGen has focused on meaningful engagement based on continuing to build trust, respect, and confidence with Indigenous groups and local communities that could potentially be impacted by the Project.

Methods of engagement have included notification letters, meetings with leadership, and community information sessions. Specific to engagement with Indigenous groups, NexGen has established Joint Working Groups (JWGs) for detailed discussions and provided funding for traditional land use studies. NexGen has committed to listening to Indigenous groups and responding to questions and concerns appropriately. NexGen acknowledges that engagement is a dynamic process, and has expressed its intent to maintain flexibility to ensure engagement remains meaningful.

NexGen has further stated that the company is seeking to engage in this dialogue to include Indigenous knowledge, traditional land use, and other items of value to Indigenous groups in the EA, project design, and licensing programs. NexGen is also committed to facilitating opportunities to participate in the identification, development, and review of mitigation measures.

Feedback received during engagement activities—including issues and concerns raised—is documented and attributed to the appropriate Indigenous group, and NexGen responses or follow-up actions are noted. Engagement records are stored on a secure server with restricted access and summarized in engagement logs and feedback tracking tables. Engagement is active and ongoing, and it will reflect the stage of project development, including the EA stage.

Four JWGs were established with Indigenous groups for NexGen to work directly with representatives identified by Indigenous leadership to support the gathering and incorporation of Indigenous knowledge throughout the EA process. The JWGs are endorsed by the leadership of each Indigenous group and include representation from both NexGen and each respective Indigenous group. The JWGs meet regularly, based on community availability, with meetings scheduled to continue throughout the Project's EA.

20.8.3 Agreements

In the second half of 2019, NexGen entered into Study Agreements (Agreements) with the following four Indigenous groups.

- Clearwater River Dene Nation
- Métis Nation – Saskatchewan (MN-S), including as on behalf of the Locals of MN-S Northern Region II
- Birch Narrows Dene Nation
- Buffalo River Dene Nation

The Agreements provide a framework for working collaboratively to advance the EA and exchange information that will be used to inform the Crown as the Crown undertakes its duty to consult.

The Agreements provide funding to each Indigenous group and outline a collaborative process for formal engagement to support the inclusion of Indigenous knowledge in the EA. The Agreements also outline processes for identifying potential effects to Indigenous rights, treaty rights, and socio-economic interests, and avoidance and accommodation measures in relation to the Project.

Through the Agreements, NexGen provides funding to each of the Indigenous groups for a community-led traditional land use study, participation in a JWG with NexGen, and engagement of technical expertise to provide support as required. In addition, the Agreements formally reflect the continued commitment of the community initiatives and programs NexGen has conducted since 2013. Furthermore, the Agreements commit to negotiating an Impact Benefit Agreement (IBA) in good faith, as early in the regulatory process as possible. The IBAs are centered around the following pillars.

- Environmental protection and assurance
- Culture and traditional values and community engagement
- Economic participation, including employment, training, and contracting opportunities
- Financial payments
- Agreement implementation

In addition, in Q3 2020 NexGen entered into a Study Funding Agreement with Ya'thi Néné Land and Resource Office (YNLR) to undertake a Traditional Knowledge, Land Use and Occupancy Study in relation to the Project. YNLR is responsible for holding Traditional Knowledge, Land Use and Occupancy information on behalf of the Athabasca Denesųliné First Nations (including Fond du Lac Denesųliné First Nation and Black Lake Denesųliné First Nation).

20.9 Comments on Section 20

Baseline studies have been completed to support Project permitting. NexGen will need to comply with a number of Federal and Provincial acts during construction and operations. NexGen has developed and is implementing a Project-specific plan for engaging the public and Indigenous communities on the Project.

The EA process for the Rook I Project is in progress as of the effective date of this report, and preparation of a Draft EIS is underway.

21.0 CAPITAL AND OPERATING COSTS

Capital and operating cost estimates were prepared by Stantec, Wood, and Patterson & Cooke, with contributions from NexGen.

21.1 Capital Cost Estimates

21.1.1 Basis of Estimate

The capital cost estimate meets the criteria to be classified as a Class 3 estimate, as defined by AACE International. It has an approximate accuracy of ±15%. All costs included in the estimate are reported in Q4 2020 Canadian dollars.

The capital cost estimate reflects a detailed bottom-up approach based on key engineering deliverables that define the project scope. This scope was described and quantified within material takeoffs (MTOs) in a series of line items.

The pre-production capital costs are defined as all costs incurred from Year -4, up to Year 1, while sustaining capital costs are costs incurred from Year 1 through to the end of mine life.

Pre-production capital costs have been divided into the following.

- Pre-commitment early works Capital
  - Clearing and grubbing.
  - Site levelling and road construction.
  - Batch plant construction.
  - Initial camp construction.
  - Shaft-sinking preparations, including freeze hole drilling, freeze plant installation, and sinking plant installations).
  - Contingency
- Project capital
  - Areas of UG mining (including shaft sinking, development, and infrastructure)
  - Site development (clearing / grubbing and civils)
  - Processing
  - On and off-site infrastructure
  - Indirect expenses
  - Capitalized Owner's costs (Year -4 up to Year 1)
  - Contingency

Sustaining capital costs are related to the following.

- UG mine mobile equipment
- Development

- Infrastructure construction
- Process plant maintenance
- Infrastructure maintenance
- Mine closure.

21.1.2 Pre-Commitment Early Works

NexGen is preparing a pre-commitment early works program that will encompass all scheduled activities planned for Year -4 Month 1 through Month 6. This plan will advance certain elements of the overall scope and mitigate project risks. The program includes work and the associated costs that NexGen intends on expending prior to an FID.

The scope of the pre-commitment early works program includes the following (at a high level).

- Clearing and grubbing.
- Site levelling and road construction.
- Batch plant construction.
- Initial camp construction.
- Shaft-sinking preparations, including freeze hole drilling, freeze plant installation, and sinking plant installations).

Stantec estimates the pre-commitment early works program will cost approximately $157.9 million.

Area	Units	Cost
Mining	$ million	51.9
Site Development	$ million	8.4
On-Site / Off-Site Infrastructure	$ million	49.1
Owners Costs	$ million	6.2
Indirect Costs	$ million	26.8
Contingency	$ million	15.6
Total pre-commitment early works	**$ million**	**157.9**

1. Pre-commitment Early Works contains contingency separate from the project contingency.
2. Totals may not sum due to rounding.

21.1.3 Direct Mine Capital Costs

The project period mine capital costs primarily include shaft sinking, lateral, and vertical mine development, and stationary mine infrastructure. Mine mobile equipment used for the Project—which is typically another major cost area—will likely be purchased on a “lease to own” basis.

Stantec assumed there will be two main contracts associated with the access and development of the mine: a Shaft and Headworks Construction Contract, and an UG Lateral Development and Installations Contract. The shaft and headworks contractor will be responsible for erecting both temporary and permanent headworks, the concurrent sinking of both Production Shaft and Exhaust Shaft, and the subsequent operation of the shafts from Year -4 to Year 1.

The UG lateral development and installations contractor will be responsible for the UG ramp, lateral and vertical (raising) development (in waste-rock only) starting from the shaft station(s), and construction of stationary mine infrastructure from Year -2, Month 4 to Year 1.

All UG ore development and pre-production mining will be executed by the Owner's labour force. The mine capital cost estimate is presented in Table 21-1.

Table 21-1: Direct Mine Cost Estimate

Area	Units	Cost
Surface Infrastructure	$ million	76.0
Shaft Sinking & Infrastructure	$ million	82.2
Underground Development	$ million	31.3
Underground Mine Equipment	$ million	14.3
Underground Infrastructure	$ million	29.5
UGTMF Access Development	$ million	3.2
UGTMF Infrastructure	$ million	3.4
Total	**$ million**	**240.0**

Note: Totals may not sum due to rounding.

21.1.4 Direct Process Plant Capital Costs

Process plant costs include the construction of the entirety of the process plant facility and contained entirely within the project period. All process capital costs are summarized in Table 21-2.

Table 21-2: Process Capital Cost Estimate

Description	Units	Cost
Ore Handling & Stockpiles	$ million	5.5
Grinding	$ million	7.4
Leaching	$ million	3.5
Liquid / Solids Separation	$ million	25.7
Solvent Extraction	$ million	43.1
Precipitation	$ million	15.1

Description	Units	Cost
Tailings Neutralization	$ million	9.5
Product Drying and Packaging	$ million	15.0
Plant Building & Services	$ million	91.6
Total Process Capital Costs	**$ million**	**216.4**

Note: Totals may not sum due to rounding.

21.1.5 Direct Infrastructure Capital Costs

The project period infrastructure costs include site development and on-site and off-site infrastructure including the following.

- LNG power plant
- Site power distribution
- Paste backfill plant
- Water management systems
- Waste management systems
- Permanent camp
- Administrative facilities
- Dry facilities
- Maintenance shop
- Fuel storage
- Airstrip
- Information technology (IT) and communications systems
- Surface support mobile equipment

All infrastructure capital costs are summarized in Table 21-3 and Table 21-4.

Table 21-3: Site Development Capital Cost Estimate

Description	Units	Cost
Pad Civil Work	$ million	5.4
Site Roads	$ million	1.8
Borrow Pit & Crushing Operations	$ million	0.0
Site Water Management	$ million	8.8
Overburden Dumps and Stockpiles	$ million	0.0
Airstrip, Apron and Airport Facilities	$ million	6.0
Waste Rock Stockpiles	$ million	5.7
Total Site Development Capital Costs	**$ million**	**27.7**

Note: Totals may not sum due to rounding.

Table 21-4: On-site / Off-site Infrastructure

Description	Units	Cost
Power Supply & Distribution	$ million	43.6
Paste Backfill Plant	$ million	16.4
Water Management Systems	$ million	27.5
Waste Management Systems	$ million	2.4
Ancillary Facilities	$ million	18.4
Bulk Fuel Storage & Distribution	$ million	0.0
IT & Communications	$ million	3.7
Support Mobile Equipment	$ million	4.2
Off-Site Roads	$ million	2.5
Total Site Development Capital Costs	**$ million**	**118.9**

Note: Totals may not sum due to rounding.

21.1.6 Indirect Capital Costs

The project period indirect costs include the following.

- Temporary construction facilities
- Construction support contracts and equipment
- Contractor indirects
- Start-up and commissioning
- Logistics and freight
- Integrated project / construction management team

Engineering and procurement costs were excluded from the capital cost estimate; Stantec anticipates that subsequent phases of engineering will be financed by other means. All indirect capital costs are summarized in Table 21-5.

Table 21-5: Indirect Capital Cost Estimate

Description	Indirect (% of Project Capital)	Indirect Cost ($ million)
Construction Support Contracts & Equipment	2.8	28.7
Contractor Indirects	16.3	167.9
Temporary Facilities	0.2	2.3
Start-up & Commissioning	1.4	14.1
Logistics & Freight	1.6	16.2
Project & Construction Management	5.3	54.6
Tax	4.2	42.7
Total Indirect Capital Costs	**–**	**326.5**

Note: Totals may not sum due to rounding.

21.1.7 Owner Capital Costs

The project period Owner capital costs included in the capital cost estimate pertain to pre-production ore and special waste development, pre-production mining, pre-production processing, tailings / paste costs, and G&A costs incurred during the pre-production period. The Owner's costs are summarized in Table 21-6.

Table 21-6: Owner Capital Cost Estimate

Description	Units	Cost
Project Ore Development	$ million	1.1
Project Mining	$ million	14.9
Project Processing	$ million	7.6
Project G&A	$ million	17.9
Permitting & Environmental	$ million	0.0
Support & Infrastructure	$ million	56.4
Decommissioning/Closure (Net of Salvage Value)	$ million	0.0
Total Owner's Capital Costs	**$ million**	**97.9**

Note: Totals may not sum due to rounding.

21.1.8 Capital Cost Estimate Contingency

Table 21-7 summarizes the project period capital cost estimate contingency for the Project.

Table 21-7: Capital Cost Estimate Contingency

Description	Total Costs ($ million)	Contingency (%)	Contingency ($ million)
Mining	240.0	11.5	27.5
Processing	216.4	10.8	23.5
Site Development	27.7	11.7	3.2
On-Site / Off-Site Infrastructure	118.9	11.7	13.9
Indirect Costs	326.5	11.0	35.9
Owner's Costs	97.9	11.0	10.7
Total Contingency Capital Costs	**1,027.2**	**11.2**	**114.8**

Note: Totals may not sum due to rounding.

21.1.9 Capital Cost Estimate Summary

Table 21-8 summarizes the pre-production estimated capital costs associating with, supplying, constructing, and pre-commissioning the Project.

Table 21-8: Total Pre-production Capital Cost Estimate

Description	Units	Cost
Pre-commitment early works	**$ million**	**157.9**
Project Capital		
UG Mining	$ million	240.0
Processing	$ million	216.4
Site Development	$ million	27.7
On-Site / Off-Site Infrastructure	$ million	118.9
Subtotal Project Direct Costs	***$ million***	***602.9***
Project Indirect Costs	$ million	326.5
Project Owner's Costs	$ million	97.9
Subtotal Project Direct, Indirect & Owner's Costs	***$ million***	***1,027.2***
Project Contingency	$ million	114.8
Total Project Capital	**$ million**	**1,142.0**
Pre-production Capital Cost (Pre-Commitment & Project)	**$ million**	**1,299.9**

Note:
1. Total shown includes Pre-commitment Early Works, which does not get carried forward into Section 22.0.
2. Pre-commitment Early Works contains contingency separate from the project contingency.
3. Totals may not sum due to rounding.

21.1.10 Sustaining Capital Cost Estimate

Table 21-9 summarizes the sustaining and reclamation capital cost estimates for the Project.

Table 21-9: Sustaining and Reclamation Capital Cost Estimates

Description	Units	Total
UG Mining Equipment	$ million	62.4
UG Mine Development	$ million	107.7
UGTMF	$ million	39.5
UG Infrastructure	$ million	78.2
Process Plant	$ million	14.8
Surface Infrastructure	$ million	28.0
Indirect Costs	$ million	31.9
Total Sustaining Capital Costs	*$ million*	362.4
Reclamation and Closure (less salvageable equipment)	$ million	69.5
Total Sustaining and Reclamation	**$ million**	432.0

Note: Totals may not sum due to rounding.

Underground

Sustaining capital costs incorporates all capital expenditures after the pre-production period of Year -4 to Year -1. Mine sustaining capital costs include some capital spending that will not be completed until Year 1, as well as certain pieces of equipment that will need to be rebuilt over the LOM, and ongoing mine development. Sustaining capital for the mine infrastructure was also included, specifically for the RAR fans, ore / waste bins and handling infrastructure, maintenance shop, and other mine capital items that will incur costs on an ongoing basis.

The UGTMF will be constructed in phases; the first phase will be completed during the pre-production period. Subsequent expansions of the UGTMF will occur during the production period, on an as-needed basis.

Surface

Sustaining capital costs will be used to rebuild and replace worn-out equipment on surface, and to maintain surface facilities for overall operating efficiency. In addition to facility maintenance, the sustaining capital costs for the surface include costs associated with the lease-to-own mobile equipment program.

Closure and Reclamation

Stantec has prepared an estimate to decommission and reclaim the site. Reclamation costs totalling $69.5 million have been estimated, which include a deduction of $9.1 million for salvageable equipment have been included for Year 12 through Year 16.

21.2 Operating Cost Estimates

21.2.1 Basis of Estimate

Stantec, Wood, and Paterson & Cooke developed operating cost estimates to determine the annual production costs. Unit costs are expressed in $/tonne processed and $/lb U_3O_8. All costs included in the estimate are reported in Q4-2020 Canadian dollars. Operating costs were allocated to one of mining, processing, and G&A.

21.2.2 Labour

The total estimated on-site labour for the Arrow Deposit project is shown in Table 21-10. The totals include the surface and UG construction labour, supervision, staff, maintenance, G&A, the integrated management team plus an allowance for visitors, contractors, and consultants. The numbers are reflective of an annual peak, but will vary throughout each year depending on the ongoing activities.

Table 21-10: Total On-site Labour – Annual Peak

Labour Type	Year of Mine Life														
	Y-4	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
UG – Operating/Sustaining															
Staff	0	1	1	3	31	31	31	31	31	31	31	31	29	28	19
Hourly	0	0	0	44	132	136	134	132	124	106	105	99	93	90	81
UG – Shaft Sinking / UG Capital															
Staff	13	23	57	26	0	0	0	0	0	0	0	0	0	0	0
Hourly	75	106	86	164	0	0	0	0	0	0	0	0	0	0	0
Process Plant Labour															
Staff	0	0	1	2	2	2	2	2	2	2	2	2	2	2	2
Hourly	0	0	0	57	62	62	62	2	62	62	62	62	62	62	62
Surface Construction															
Staff	76	31	112	0	0	0	0	0	0	0	0	0	0	0	0
Hourly	25	32	26	3	0	0	0	0	0	0	0	0	0	0	0
G&A															
Staff	0	0	0	3	6	6	6	6	6	6	6	6	6	6	6
Hourly	0	0	0	10	23	23	23	23	23	23	23	23	23	23	23
Integrated Management Team															
Total	18	40	49	28	28	0	0	0	0	0	0	0	0	0	0
Visitor, Contractors, and Consultants															
Allowance	10	12	17	17	14	13	13	10	12	11	11	11	11	11	10
Total															
Staff	116	106	236	79	81	52	52	49	51	50	50	50	48	47	37
Hourly	100	137	112	277	216	220	218	157	208	190	189	183	177	174	165
Combined	216	243	348	356	297	272	270	205	259	240	239	233	225	221	202

Note: Totals may not sum due to rounding.

21.2.3 Mine Operating Costs

UG mining will occur from Year -2 through Year 11 (in Year -2 and Year -1, UG mining costs will be capitalized). UG mining will begin with capital development in Year -2, and the capitalized development will continue through Year 10. All UG mine operating costs are summarized in Table 21-11.

Table 21-11: Underground Mine Operating Cost Estimate

Description	LOM Cost ($ million)	Unit Cost ($/t processed)	Unit Cost ($/lb U_3O_8)
Labour	336.1	73.47	1.44
Mine Consumables	98.7	21.58	0.42
Equipment Operations and Maintenance	172.7	37.75	0.74
Power Consumption	72.1	15.76	0.31
In-fill Drilling	11.6	2.53	0.05
Total	**6,91.3**	**151.09**	**2.96**

Note: Totals may not sum due to rounding.

21.2.4 Process Plant Operating Costs

The process plant operating costs are primarily comprised of labour, power consumption, treated process water, and consumables. Consumables consist of reagents, grinding media, mill liners, and LNG. An allowance was included for annual maintenance. All process plant operating costs are summarized in Table 21-12.

Table 21-12: Process Plant Operating Cost Estimate

Description	LOM Cost ($ million)	Unit Cost ($/t processed)	Unit Cost ($/lb U_3O_8)
Labour	159.7	34.89	0.68
Power Consumption	95.0	20.77	0.41
Water	0.3	0.06	0.00
Grinding Media, Liners, & Wear Parts	14.1	3.08	0.06
Reagents	326.8	71.43	1.40
Maintenance Materials	49.7	10.87	0.21
Laboratory	1.4	0.30	0.01
Total	**647.0**	**141.41**	**2.77**

Note: Totals may not sum due to rounding.

21.2.5 Tailings Facility / Paste Plant Operating Costs

The tailings facility / paste plant operating costs are primarily comprised of labour, power consumption, and binder. An allowance was included for annual maintenance. All tailings facility / paste plant operating costs are summarized in Table 21-13.

Table 21-13: Tailings Facility / Paste Plant Operating Cost Estimate

Description	LOM Cost ($ million)	Unit Cost ($/t processed)	Unit Cost ($/lb U_3O_8)
Labour	9.2	2.00	0.04
Power Consumption	11.3	2.46	0.05
Binder	119.0	26.00	0.51
Maintenance Materials	4.6	1.00	0.02
Total	**144.0**	**31.46**	**0.62**

Note: Totals may not sum due to rounding.

21.2.6 General and Administrative Operating Costs

G&A costs include labour, camp and catering costs, flights to and from site, insurance premiums, and general maintenance of the surface buildings. All G&A costs are summarized in Table 21-14.

Table 21-14: G&A Operating Cost Estimate

Description	LOM Cost ($ million)	Unit Cost ($/t processed)	Unit Cost ($/lb U_3O_8)
Labour	67.2	14.68	0.29
Camp Costs	57.0	12.45	0.24
Flights and Logistics	25.1	5.49	0.11
Miscellaneous (e.g., service contracts, emergency / medical, environmental, HR, insurance, external consulting)	41.8	9.14	0.18
Equipment Maintenance and Fuel	56.9	12.44	0.24
Infrastructure Maintenance	25.4	5.56	0.11
Power Consumption	14.2	3.09	0.06
Total	**287.5**	**62.84**	**1.23**

Note: Totals may not sum due to rounding.

21.2.7 Operating Cost Estimate Summary

LOM operating costs are estimated to be $1,769.8 million. LOM operating costs are summarized in Table 21-15.

Table 21-15: Operating Cost Estimate Summary

Description	LOM Cost ($ million)	Average Annual ($ million)	Unit Cost ($/t processed)	Unit Cost ($/lb U_3O_8)
Mining	691.3	64.6	151.09	2.96
Processing	646.9	60.5	141.41	2.77
Tailing Facility / Paste Plant	144.0	13.5	31.46	0.62
General & Administration	287.5	26.9	62.84	1.23
Total	**1,769.8**	**165.4**	**386.80**	**7.58**

Notes
1. Average annual cost based on 10.7 years.
2. Totals may not sum due to rounding.

21.3 Comments on Section 21

Total capital costs—including pre-commitment, project, sustaining, and reclamation costs total $1,741.0 million.

LOM operating costs are estimated to be $1,769.8 million.

22.0 ECONOMIC ANALYSIS

22.1 Cautionary Statement

The results of the economic analysis represent forward-looking information that is subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented in this section.

Forward-looking statements in this report include statements regarding future uranium prices, the estimation of Mineral Resources and Mineral Reserves, the estimated mine production and uranium that will be recovered, the estimated capital and operating costs, and the estimated cash flows that will be generated from mine production. Actual results may be affected by the following factors.

- Differences in estimated initial capital costs and development time from what has been assumed in the 2021 FS.
- Unexpected variations in quantity of ore, grade or recovery rates, or presence of deleterious elements that would affect the process plant or waste disposal.
- Unexpected geotechnical and hydrogeological conditions that differ from what was assumed in the mine designs, including water management during construction, mine operations, and post mine closure.
- Differences in the timing and amount of estimated future uranium production, costs of future uranium production, sustaining capital requirements, future operating costs, currency exchange rate, requirements for additional capital, unexpected failure of processing plant, and equipment or processes not operating as anticipated.
- Changes in government regulation regarding mining operations, the environment, and taxes.
- Unexpected social risks, higher closure costs, unanticipated closure requirements, mineral title disputes, or delays to obtaining surface access to the property.

The production schedules and financial analysis annualized cash flow (in Canadian Dollars) are presented in Table 22-1, Table 22-2, and Table 22-3 with conceptual years shown. The annual cash flow, the cumulative cash flow and the project economics are shown in Table 22-4.

Table 22-1: Annualized Cashflow Forecast – Project Period

	Project Period				
	Period Total	**Y-4**	**Y-3**	**Y-2**	**Y-1**
Production					
Tonnes Milled (×1000)	0	0.0	0.0	0.0	0.0
Uranium Production (lb U_3O_8 x 1000)	0	0.0	0.0	0.0	0.0
Revenue ($ million)					
Uranium	0	0	0	0	0
NSR	***0***	***0***	***0***	***0***	***0***
Gross Revenue Royalties	0	0	0	0	0
Net Revenue	***14,444***	***0***	***0***	***0***	***0***
Mine	0	0	0	0	0
Processing	0	0	0	0	0
Tailings Facility & Paste Plant	0	0	0	0	0
G&A	0	0	0	0	0
Total Onsite Operating Cost	***1,770***	***0***	***0***	***0***	***0***
Operating Cashflow (EBITDA)	0	0	0	0	0
Project Capital	1,142	119	262	488	272
Working Capital	(39)	(28)	(17)	(19)	25
Capital Expenditures	***1,103***	***91***	***245***	***469***	***298***
Pre-Tax Cashflow	(1,103)	(91)	(245)	(469)	(298)
Taxes ($ million)					
Depreciation (CCA, CDE, CEE)	480	26	50	103	301
Earnings Before Income Tax (EBIT)	(480)	(26)	(50)	(103)	(301)
Federal Income Tax	0	0	0	0	0
Investment Tax Credits	0	0	0	0	0
SK Provincial Income Tax	0	0	0	0	0
SK Profit Tax (Profit Royalty)	0	0	0	0	0
Income after Taxes	(204)	(26)	(26)	(50)	(103)
After-Tax Cash Flow		(91)	(245)	(469)	(298)

Table 22-2: Annualized Cashflow Forecast – Operating Period

	Operating Period											
	Period Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Production												
Tonnes Milled (×1,000)	4,575	278	432	455	455	455	455	455	455	455	417	263
Uranium Production (lb U_3O_8 × 1,000)	233,597	26,103	29,059	29,713	29,664	29,669	15,534	16,785	18,109	15,398	15,750	7,813
Revenue ($ million)												
Uranium	15,573	1,740	1,937	1,981	1,978	1,978	1,036	1,119	1,207	1,027	1,050	521
NSR	***15,573***	***1,740***	***1,937***	***1,981***	***1,978***	***1,978***	***1,036***	***1,119***	***1,207***	***1,027***	***1,050***	***521***
Gross Revenue Royalties	1,129	126	140	144	143	143	75	81	88	74	76	38
Net Revenue	***14,444***	***1,614***	***1,797***	***1,837***	***1,834***	***1,835***	***960***	***1,038***	***1,120***	***952***	***974***	***483***
Mine	691	52	74	77	76	66	66	64	61	58	56	40
Processing	647	50	63	66	67	67	58	59	60	58	56	45
Tailings Facility & Paste Plant	144	9	14	14	14	15	15	14	14	14	11	8
G&A	288	28	28	28	25	27	27	27	26	26	26	19
Total onsite operating cost	***1,770***	***140***	***179***	***185***	***182***	***175***	***165***	***164***	***161***	***157***	***150***	***112***
EBITDA	12,674	1,474	1,617	1,653	1,652	1,660	795	874	959	796	824	371
Working Capital	50	142	(54)	41	(22)	5	(39)	3	5	(7)	2	(24)
Sustaining Capital	362	72	75	66	57	39	9	11	5	5	0	23
Capital Expenditures	***413***	***214***	***21***	***107***	***34***	***44***	***-31***	***14***	***10***	***-2***	***3***	***-2***
Pre-Tax Cashflow	12,261	1,261	1,597	1,546	1,618	1,616	826	860	948	798	821	373
Taxes ($ million)												
Depreciation (CCA, CDE, CEE)	1,260	388	185	155	130	109	84	64	49	38	29	28
EBIT	11,415	1,086	1,432	1,498	1,522	1,551	711	810	909	757	795	343
Federal Income Tax	1,336	53	179	188	192	195	89	102	114	89	95	39

	Operating Period											
	Period Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Investment Tax Credits	0	(0)	0	0	0	0	0	0	0	0	0	0
SK Provincial Income Tax	1,069	43	143	151	153	156	71	81	92	71	76	31
SK Profit Tax (Profit Royalty)	1,684	11	236	243	245	248	121	132	146	121	126	54
Income after Taxes	6,806	(301)	980	873	916	933	951	431	495	557	476	497
After-Tax Cash Flow		1,155	1,037	964	1,028	1,016	546	544	596	516	523	249

Table 22-3: Annualized Cashflow Forecast – Post Operations

	Post Operating Period						
	Period Total	**Y12**	**Y13**	**Y14**	**Y15**	**Y16**	**Y17**
Revenue ($ million)							
Uranium	0	0	0	0	0	0	0
NSR	***0***	***0***	***0***	***0***	***0***	***0***	***0***
Gross Revenue Royalties	0	0	0	0	0	0	0
Net Revenue	***0***	***0***	***0***	***0***	***0***	***0***	***0***
Mine	0	0	0	0	0	0	0
Processing	0	0	0	0	0	0	0
Tailings Facility & Paste Plant	0	0	0	0	0	0	0
G&A	0	0	0	0	0	0	0
Total Onsite Operating Cost	***0***	***0***	***0***	***0***	***0***	***0***	***0***
EBITDA	0	0	0	0	0	0	0
Working Capital	(12)	(14)	(1)	0	2	0	1
Sustaining Capital	79	24	20	2	8	8	0
Salvage	(9)	(9)	0	0	0	0	0
Capital Expenditures	***58***	***0***	***19***	***20***	***10***	***8***	***1***
Pre-Tax Cashflow	(58)	(0)	(19)	(20)	(10)	(8)	(1)
Taxes ($ million)							
Depreciation (CCA, CDE, CEE)	57	19	13	9	7	5	4
EBIT	(57)	(19)	(13)	(9)	(7)	(5)	(4)
Federal Income Tax	0	0	0	0	0	0	0
Investment Tax Credits	0	0	0	0	0	0	0
SK Provincial Income Tax	0	0	0	0	0	0	0
SK Profit Tax (Profit Royalty)	0	0	0	0	0	0	0
Income after Taxes	166	219	(19)	(13)	(9)	(7)	(5)
After-Tax Cash Flow	(58)	(0)	(19)	(20)	(10)	(8)	(1)

Table 22-4: Annual Cash Flow and Cumulative Cash Flow – Life of Mine

		Project Period				Operating Period											Post Operating Period					
	LOM	**Y-4**	**Y-3**	**Y-2**	**Y-1**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**	**Y13**	**Y14**	**Y15**	**Y16**	**Y17**
Pre-Tax																						
Cash Flow ($ million)	11,100	(91)	(245)	(469)	(298)	1,261	1,597	1,546	1,618	1,616	826	860	948	798	821	373	(0)	(19)	(20)	(10)	(8)	(1)
Cumulative cash flow ($ million)	–	(91)	(336)	(805)	(1,103)	157	1,754	3,300	4,918	6,533	7,359	8,219	9,167	9,964	10,786	11,158	11,158	11,139	11,119	11,110	11,102	11,100
NPV 8% ($ million)	5,577	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Payback period (years)	0.8		–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
IRR before tax	64.9%		–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
After Tax																						
Cash Flow ($ million)	7,012	(91)	(245)	(469)	(298)	1,155	1,037	964	1,028	1,016	546	544	596	516	523	249	(0)	(19)	(20)	(10)	(8)	(1)
Cumulative cash flow ($ million)	–	(91)	(336)	(805)	(1,103)	51	1,089	2,052	3,081	4,096	4,642	5,186	5,782	6,299	6,822	7,070	7,070	7,051	7,031	7,022	7,014	7,012
NPV 8% ($ million)	3,465	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–		
Payback period (years)	0.9	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–		
IRR before tax	52.4%	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–		

Note: Payback period is calculated from the start of production

Years shown in Table 22-1, Table 22-2, Table 22-3, and Table 22-4 are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project's assumptions, and may result in changes to the calendar timelines presented, and the information and statements in this report.

The NexGen Board has not yet given development approval. Statutory permits, including environmental permits, will also be required prior to beginning work on the Project.

22.2 Methodology Used

The Project has been evaluated using discounted cash flow analysis. Cash inflows consist of annual revenue projections. Cash outflows consist of project capital expenditures, sustaining capital costs, operating costs, taxes, royalties, and commitments to other stakeholders. These are subtracted from revenues to determine the annual cash flow projections. Cash flows are assumed to occur at the mid-point of each period.

To reflect the time value of money, annual cash flow projections are discounted back to the project valuation date using the yearly discount rate. The discount rate appropriate to a specific project may depend on many factors, including the type of commodity, capital costs, and the level of project risks (e.g., market risk, environmental risk, technical risk and political risk) in comparison to the expected return from the equity and money markets.

The base case discount rate used in the 2021 FS is 8%. The annual cash flows are discounted to the present value using a standard formula that assumes mid-year cash flows. In addition to the NPV, the IRR and the payback period are also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. The payback period is calculated as the time required to achieve positive cumulative cash flow for the project from the start of production.

22.3 Financial Model Parameters

22.3.1 Mineral Resource, Mineral Reserve, and Mine Life

The Mineral Resource discussed in Section 14.0 was converted to the Mineral Reserve outlined in Section 15.0. The estimated Mineral Reserve will support an eleven-year production life, using the mine plan discussed in Section 16.0.

22.3.2 Metallurgical Recoveries

The basis for the process recoveries is included in Section 13.0, and the process design is outlined in Section 17.0.

22.3.3 Commodity Prices

The commodity price basis is discussed in Section 19.0.

22.3.4 Capital and Operating Costs

The capital and operating cost estimates are detailed in Section 21.0.

The economic analysis is based on the timing of an FID, and it does not include the pre-commitment early works capital costs, which are costs NexGen intends on expending prior to the FID. The pre-commitment early works scope includes preparing the site, completing initial freeze hole drilling, and building the supporting infrastructure (i.e., concrete batch plant, Phase I camp accommodations, and bulk fuel storage) required for the Project. Costs for the pre-commitment early works will total an estimated $157.9 million.

22.3.5 Key Assumptions

Economic criteria that were used for the purposes of the cash flow model include the following.

- Long-term price of uranium of US$50/lb U_3O_8, inclusive of YC shipping costs provided by UxC.
- All uranium sold at a long-term price of US$50/lb U_3O_8, inclusive of haulage cost to refinery
- The recovery and sale of by-products is excluded from the cash flow model.
- Exchange rate of C$1.00 = US$0.75.
- LOM processing of 4,575 kt grading 2.37% U_3O_8.
- Nominal 455 kt of material processed per year during steady state operations.
- Mine life of eleven years.
- Overall process recovery of 97.5%, including a ramp-up of recovery in Year 1.
- Total recovered YC of 233.6 Mlb.
- Royalties calculated in accordance with Guideline: Uranium Royalty System (Government of Saskatchewan, February 2017).
- Unit operating costs of $387/t of processed material, or $7.58/lb U_3O_8.
- Pre-production capital costs of $1,142.0 million, spread over four years.
- Sustaining capital costs (including reclamation) of $431.9 million, spread over the mine life.
- Costs incurred prior to mid-year Year -4 (prior to FID) totalling $157.9 million are not included in the financial model in this report.
- Payback period is calculated as the number of years to recover the initial investment from the start of production.

22.4 Taxes

Taxes and depreciation for the Project were modelled based on input from NexGen, as well as a review Guideline: Uranium Royalty System (Government of Saskatchewan, February 2017).

NexGen has opening balances of Canadian exploration expenses (CEEs) and operating losses that were applied in the tax model. According to current (2021) Canadian tax codes, pre-production mine development costs are counted toward CEE; however, this practice is being phased out. Consequently, all pre-production capital was allocated to either Canadian development expenses (CDEs) or capital cost allowance (CCA). Normally, up to 30% of the CDE balance can be applied in any given year. However, as part of the Budget Implementation Act, 2019, No.1, an accelerated investment incentive is available for expenses incurred after 2018 and before 2028.

The allowance for expenses incurred between 2018 and 2023 will be 150% of the normal 30% allowance, and the allowance for expenses incurred after 2023 and before 2028 will be 125% of the normal 30% allowance. All mining equipment and structures that are considered depreciable will fall under Class 41 of Canadian tax codes, and will depreciate at a rate of 25% annually. However, as part of the Budget Implementation Act, 2019, No.1, capital expenditures available for use after 2018 and before 2024 will be eligible for a 50% increase in the CCA deduction, and expenditures available for use after 2018 and before 2028 will not be subject to the half-year rule. These provisions were applied to income tax calculations.

In Saskatchewan, multiple government royalties apply to uranium projects. Royalties generally fall into two categories: revenue royalties and profit royalties. Royalties are described as follows.

- Resource surcharge of 3% of net revenue (where net revenue is defined as gross revenue less transportation costs directly related to transporting uranium to the first point of sale).
- Basic royalty of 5% of net revenue, less a Saskatchewan resource credit of 0.75% of net revenue, for an effective royalty rate of 4.25%.
- Tiered profit royalty, with a 10% royalty rate for the first $24.22 profit per kilogram of YC, followed by a 15% royalty on profits exceeding $24.22 per kilogram. The statutory rate of $22.00 is adjusted for inflation to $24.22 in accordance with Saskatchewan regulations.

For the tiered profit royalty, the basic royalty and resource surcharge are not deductible for calculating profit royalties. For the purposes of the royalty calculation, profits are calculated by subtracting the full value of operating costs, capital costs, and exploration expenditures from the net revenue. Revenue royalties were included in the pre-tax cash flow results; profit royalties are considered a tax, and they were therefore included in the post-tax results.

The royalties and carried interest applicable to certain mineral concessions have not been applied to the tax estimate because the Arrow Deposit is not situated within those concessions.

Federal and provincial taxes were applied at a rate of 15% and 12%, respectively.

Table 22-5 summarizes the taxes and royalties that are anticipated to be paid to the provincial and federal governments.

Table 22-5: Taxation and Royalty Considerations

Payment Type	Description	Units	Value
Provincial payments	Saskatchewan resource surcharge	$ million	467.2
	Basic revenue royalty	$ million	661.9
	Profit royalty < $24.22/kg	$ million	231.3
	Profit royalty >$24.22/kg	$ million	1,452.2
	Provincial taxes	$ million	1,068.8
	Total provincial payments	*$ million*	*3,881.4*
Federal payments	Federal taxes	$ million	1,335,7
Total government royalties and taxes		***$ million***	***5,217.1***

22.4.1 Closure Costs and Salvage Value

Reclamation costs and salvage value were included in the capital cost estimate.

22.4.2 Financing

The base case economic analysis assumes 100% equity financing, and it is reported on a 100% project ownership basis.

22.4.3 Inflation

The base case economic analysis assumes constant prices with no inflationary adjustments.

22.5 Economic Analysis

On a pre-tax basis, the NPV at 8% is $5,577 million, the IRR is 64.9%, and the assumed payback period is eleven months. On a post-tax basis, the NPV at 8% is $3,465.0 million, the IRR is 52.4%, and the assumed payback period is one year.

A summary of the LOM cashflow is provided in Table 22-6 and Figure 22-1. Table 22-7 summarizes the economic results of the 2021 FS, with the NPV at 8% base case highlighted. Refer to Table 22-1 through Table 22-4 to review cashflow on an annualized basis.

Table 22-6: LOM Cashflow Forecast Summary Table

Description	Units	Value
Gross revenue / NSR	$ million	15,573.2
Less: provincial revenue royalties	$ million	(1,129.1)
Net revenue	$ million	14,444.1
Less: total operating costs	$ million	(1,769.8)
Operating cash flow	$ million	12,674.3
Less: capital costs	$ million	(1,573.9)
Pre-tax cash flow	$ million	11,100.4
Less: provincial profit royalties	$ million	(1,683.5)
Less: taxes	$ million	(2,404.5)
Post-tax cash flow	***$ million***	**7,012.4**

Figure 22-1: Undiscounted After-Tax Cash Flow

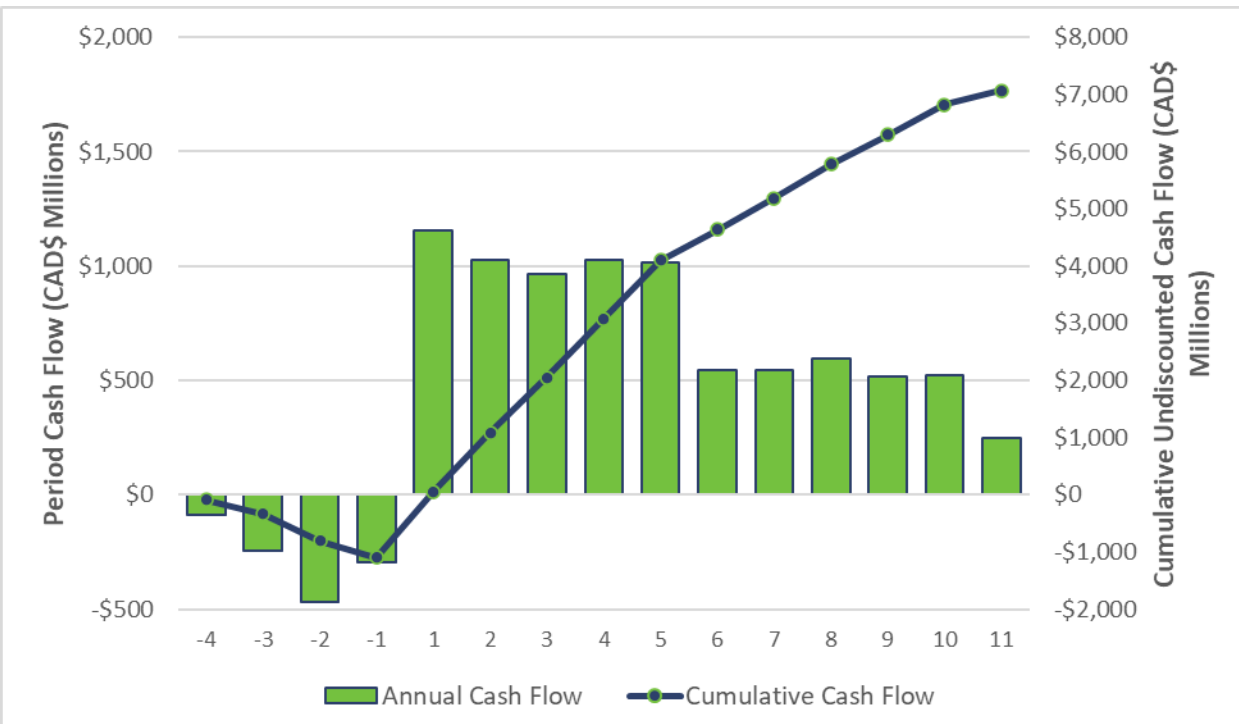


Table 22-7: 2021 Feasibility Study Forecast Economic Results

Description	Units	Value
Pre-Tax		
NPV at 8%	$ million	5,577
NPV at 10%	$ million	4,745
NPV at 12%	$ million	4,051
Internal rate of return	%	64.9%
Payback period	Years	0.8
After-Tax		
NPV at 8%	$ million	3,465
NPV at 10%	$ million	2,930
NPV at 12%	$ million	2,484
Internal rate of return	%	52.4%
Payback period	Years	0.9

Note: Payback period is calculated from the start of production

22.6 Sensitivity Analysis

The cash flow model was tested for sensitivity to variances regarding the following.

- Head grade
- Process recovery
- Uranium price
- Overall operating costs
- Overall capital costs
- Labour costs
- Reagent costs
- CAD to USD exchange rate

Figure 22-2 illustrates the results of the sensitivity analysis. The anticipated project cash flow is most sensitive to fluctuations in the price of uranium, head grade, and process recovery. YC is primarily traded in US dollars, whereas capital and operating costs for the Project are primarily priced in Canadian dollars. Therefore, the CAD to USD exchange rate may significantly influence project economics.

An extended sensitivity analysis was undertaken to evaluate the Project's sensitivity to fluctuations in the uranium price. The results of the extended sensitivity analysis are presented in Figure 22-2. The graph in Figure 22-3 shows that the Project will be resilient when faced with price changes, with positive economics even when the commodity price is at US$30/lb U_3O_8.

Figure 22-2: Sensitivity Analysis

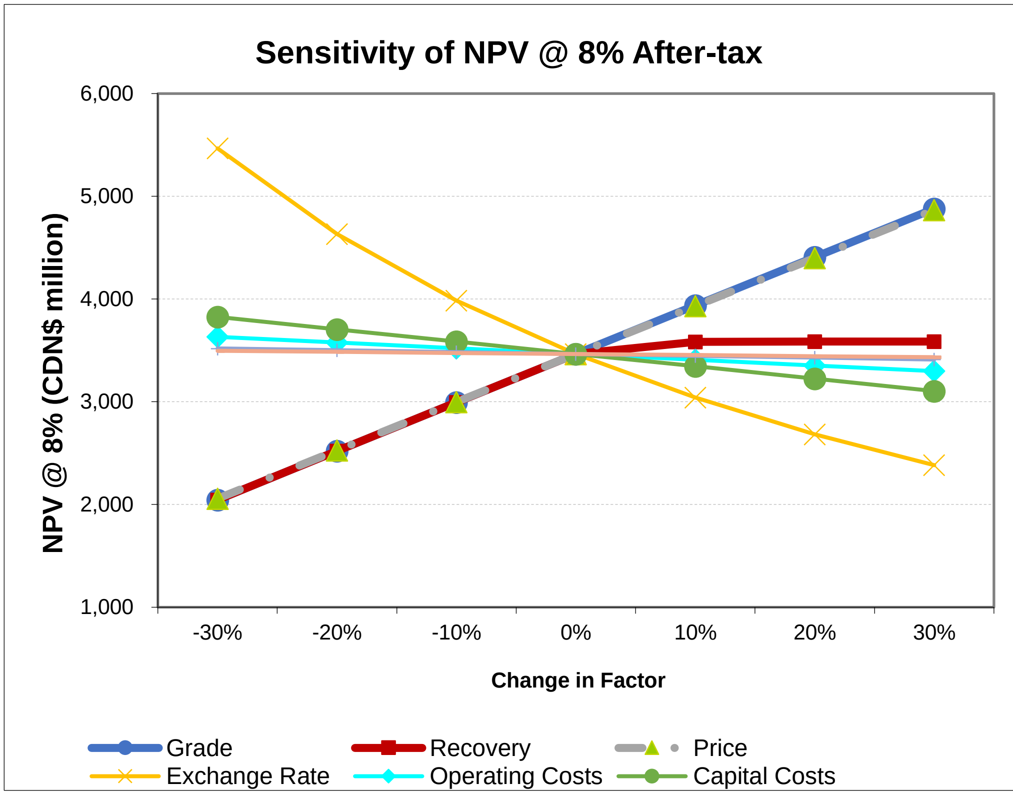
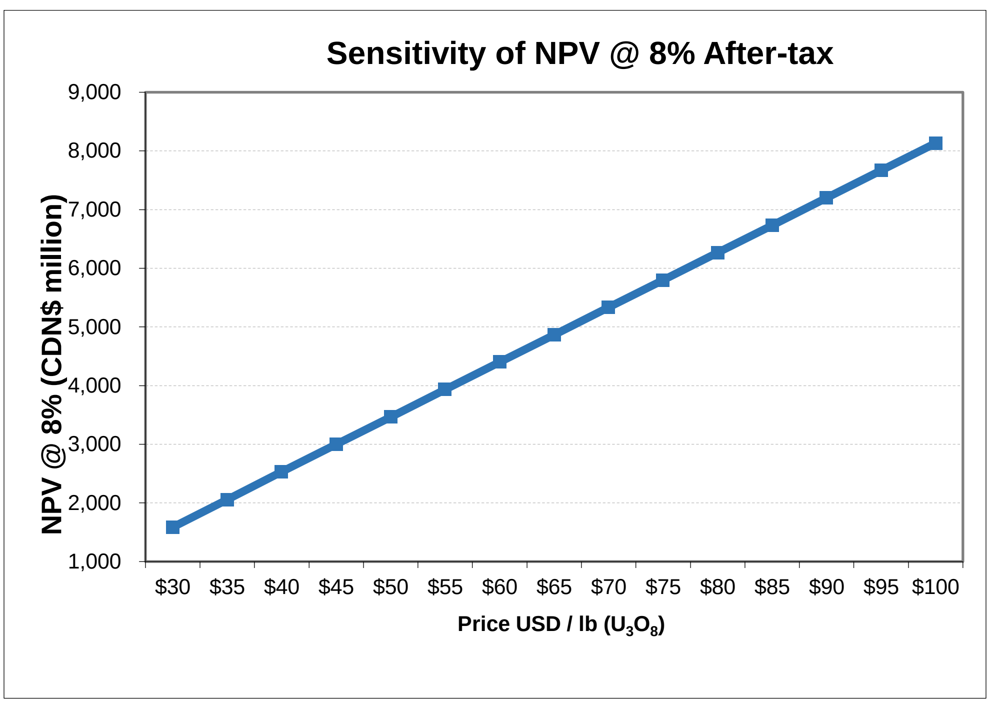


Figure 22-3: Project Uranium Price Sensitivity



22.7 Comments on Section 22

Based on the assumptions listed in Section 22.3.5, the Project shows positive economics.

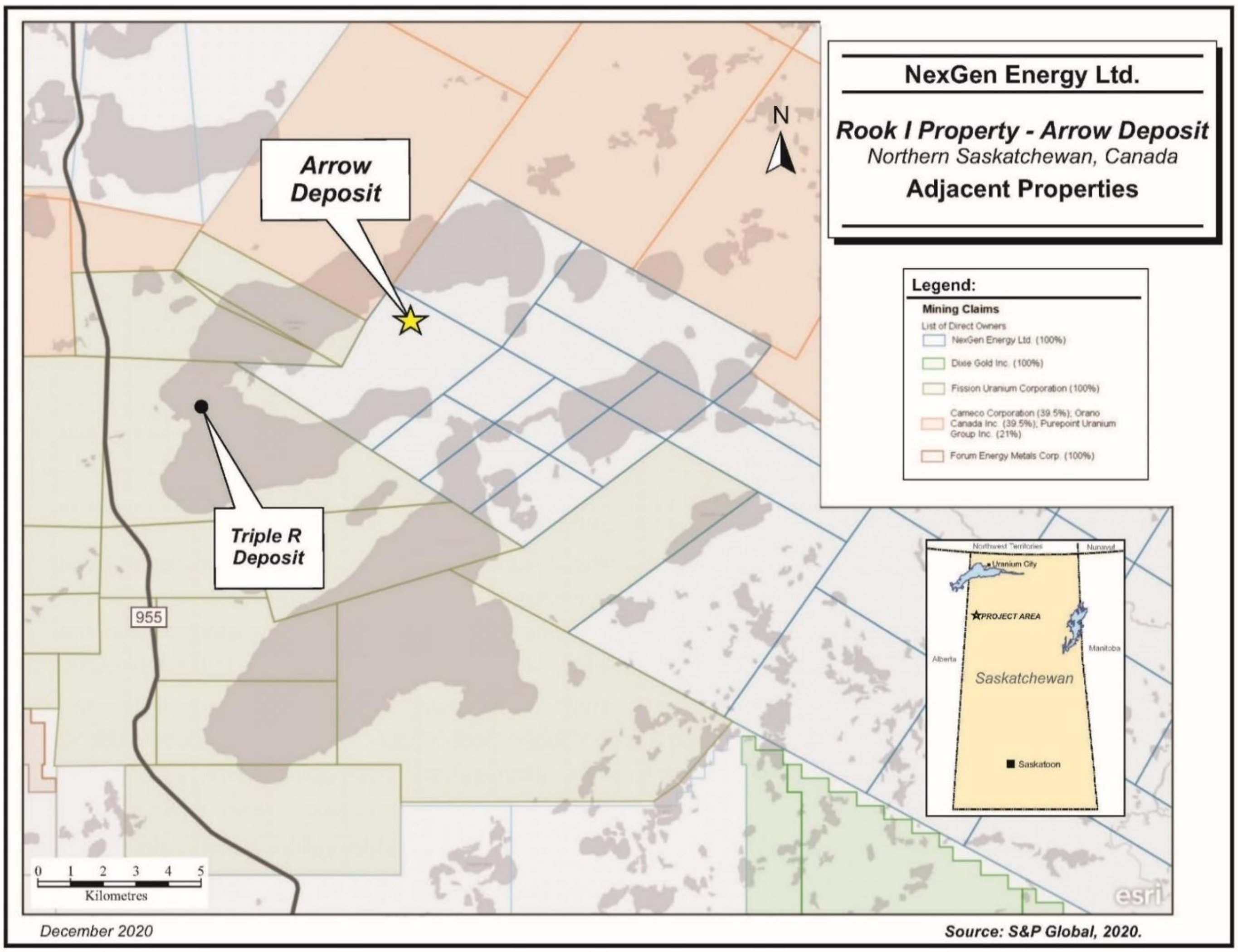
The anticipated project cash flow will be most sensitive to the price of uranium, head grade, and process recovery. The CAD to USD exchange rate may also significantly influence project economics.

23.0 ADJACENT PROPERTIES

The Project property shares borders with properties claimed by various companies and individuals. As of the effective date of this report, the property is adjacent to properties with claims registered to Fission Uranium Corp (100%) to the southwest, and a consortium consisting of Orano Resources Canada (39.5%), Cameco (39.5%), and Purepoint (21%) to the north and northwest (see Figure 23-1).

RPA has not referenced information regarding the adjacent properties for the purposes of this report.

Figure 23-1: Adjacent Properties



24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Risks

NexGen and its lead consultants have assessed critical areas of the Project and identified risks associated with the technical and cost assumptions used. The risks have been classified as low/moderate/high and commented on risk mitigation in the development plan. In all cases, the level of risk refers to the subjective assessment as to how the identified risk could affect the achievement of the Project objectives.

24.2 Risk Analysis Definitions

The following definitions have been employed in assigning risk factors to the various aspects and components of the Project:

- **Low Risk** - Risks that could or may have a relatively insignificant impact on the character or nature of the deposit and/or its economics. Generally can be mitigated by normal management processes combined with minor cost adjustments or schedule allowances.
- **Moderate Risk** - Risks that are considered to be average or typical for a deposit of this nature. These risks are generally recognizable and, through good planning and technical practices, can be minimized so that the impact on the deposit or its economics is manageable.
- **High Risks** - Risks that are largely uncontrollable, unpredictable, unusual, or are considered not to be typical for a deposit of a particular type. Good technical practices and quality planning are no guarantee of successful exploitation. These risks can have a major impact on the economics of the deposit including significant disruption of schedule, significant cost increases, and degradation of physical performance. Included in this category may be environmental/social non-compliance, particularly in regard to Equator Principles and IFC Performance Standards.

In addition to assigning risk factors, the probability of the risk occurring during the Project has been considered. The following definitions have been employed in assigning probability of the risk occurring:

- **Low –** The risk is unlikely to occur during the Project.
- **Moderate –** There is an increased probability that the risk will occur during the Project.
- **High** – The risk is likely to occur during the Project.

A summary of the Project related risks identified is shown in Table 24-1.

Table 24-1: Project Risk Evaluation

Project Element	Risk Description	Risk Level Classification	Risk Probability Classification	Existing / Proposed Strategies / Actions for Mitigation
Mineral Resources	There is a risk that resource tonnes and grade estimates are less than expected.	Low	Low	The core area of the deposit is drilled to the Measured Mineral Resource category, the highest definition.
Mineral Resources	Mineralized material could be present in areas that are currently assumed to be barren. This could cause adverse conditions for drift development, and ventilation	Moderate	Moderate	It is recommended that additional condemnation drilling be conducted in the key areas of the 500L, 590L, and 620L mine infrastructure areas, and areas around the first UGTMF chambers
Construction / Operations	Ability to attract and retain competent and experienced professionals.	Low	Moderate	Compensation packages and site conditions favourable to attract and retain quality construction and operations personnel.
Mining – Shaft Sinking	Ground freezing conditions depend on a number of factors, and currently there is limited data to estimate the time it will take for the ground to temporarily freeze prior to the shaft pre-sink commencing.	Low	Low	Consider commencing ground freezing as part of the pre-commitment early works package, to remove this item from the critical path construction timeline.
Mining	There is a risk of poor ground conditions within the deposit which could affect key design inputs such as stope dimensions, UGTMF dimensions, and development rate assumptions	Low	Moderate	Continue with geotechnical characterization, including refinement of a localized faults and structures model.
Mining	There is a risk that certain elements of the material handling system, including waste and ore passes, underground bins, skips, and surface bins, have not been sized to provide steady state and peak operations, including allowances for downtime.	Moderate	Low	It is recommended that material handling simulations be undertaken to identify critical pinch points.

Project Element	Risk Description	Risk Level Classification	Risk Probability Classification	Existing / Proposed Strategies / Actions for Mitigation
Mining	There is a risk that underground rock breakers (as opposed to crushing stations), are not able to keep up with throughput requirements, especially for material from the UGTMF chambers.	Moderate	Low	Review capabilities of the underground rock breakers in relation to estimated blast fragmentation distribution from the UGTMF chambers and stopes.
Mining	Some areas of the waste rock are NPAG, while others are PAG, with each waste rock type reporting to different waste pads on surface. There is a risk that NPAG and PAG are blended together through the use of shared waste passes, so that they are no longer able to be separated.	Moderate	Moderate	Consider design alternatives to the two separate waste pads on surface, including the construction of a larger pad designed to store PAG waste that could store both PAG and NPAG.
Processing	There is a risk that uranium recovery of the process plant is less than expected.	Low	Low	Test work supports recovery assumption. Additional test work completed to allow optimization of flowsheet.
Processing - Tailings	The concept of the UGTMF is relatively unique. Issues could arise with the design, construction, and commissioning of the UGTMF and paste plant system	Moderate	Moderate	Continue developing testing plans, commissioning plans, and material movement modeling
Regulatory	There is a risk of radiation over-exposure to personnel during operations, with the potential to for additional regulatory scrutiny and/or license review	Low	Moderate	Continue radiological studies and design for radiation safety in detailed design.
Regulatory	There is a risk that the project will encounter environmental permitting delays	Moderate	Moderate	Continue to hold regular stakeholder and regulatory meetings.

Project Element	Risk Description	Risk Level Classification	Risk Probability Classification	Existing / Proposed Strategies / Actions for Mitigation
Regulatory	There is risk that the UGTMF may not be viewed favourably by regulatory bodies, as it is a new concept for uranium mines. Failure to achieve regulatory approvals for the UGTMF would result in alternative tailings management strategies needing to be developed, potentially causing project delays.	Moderate	Moderate	Awareness and education campaigns are critical to ensure that the UGTMF concept is clearly articulated to, and understood by, the necessary regulatory agencies and other stakeholders.
Cost Estimation	There is a risk that the cost of key materials and supplies will increase	Moderate	Moderate	Continue to refine cost estimate, and identify key vendors
Procurement	Lead times for certain equipment are highly variable. Some long-lead items could cause overall project delays if not procured in a reasonable time frame.	Moderate	Low	Ongoing lead-time awareness of certain key pieces equipment should be prioritized, with the refinement of a long-lead register, and other procurement initiatives.
Marketing Ramp Up	There is a risk that the saleable product produced at Arrow will have a slower than expected ramp-up.	Moderate	Moderate	Consider advancing the marketing plan in greater detail, including identifying potential customers, locations, and quantities required.
UGTMF	Operational challenges related to the UGTMF include sequencing the mining activities, minimizing overbreak, and ensuring that the UGTMF chambers remain stable throughout the excavation and filling cycle.	Low	Moderate	Detailed geotechnical work and operational simulation work is required to effectively plan the excavation and filling of these chambers.

24.3 Opportunities

NexGen and its lead consultants performed a review of the opportunities that should be explored both as the project is advanced through the next level of study, and into eventual construction and operations. Some of the key opportunities include the following.

- Expansion of overall Mineral Resources by targeting areas of the Arrow Deposit that remain open along strike, dip, and plunge.
- Follow-up of regional occurrences and discoveries that could potentially be developed into mineral deposits.
- Extension of the mine life beyond 11 years by converting Inferred Resources to the Indicated category through diamond drilling.
- Evaluate methods in which the mining extraction factor (mining recovery) could be improved, given the high-grade nature of the deposit.
- Investigate the use of the most technologically advanced remote-controlled and autonomous mobile mining equipment, to help lower radiological exposures to personnel, and potentially reduce the number of personnel on-site.
- Review whether water consumption of the mine and process plant can be reduced by enhanced process water recycling.
- Finalize the site water management philosophy and optimize the required infrastructure.
- Although the metallurgical recovery is already high, evaluate whether this could be further improved upon.
- Evaluate the use of waste heat from the proposed acid plant and power plant, to determine whether it can be used for building heat or other purposes.
- Investigate the use of renewable energy such as wind power as an addition to baseload power generation, to reduce greenhouse gas emissions, and drive down overall net power costs.
- Assess the viability of connecting the project to a provincial power grid, to reduce or minimize the reliance on an on-site power plant.
- Continue to identify critical early works activities that could be completed ahead of major construction.
- Continue to develop the project execution plan to reduce overall project risk, and identify areas of the initial capital cost outlay, such as the power plant, acid plant, camp, that could be deferred through the use of specialized service providers.
- Develop the supply chain network, including the establishment of key vendors, warehouse locations and function, and enterprise resource planning systems, to streamline eventual project construction.

25.0 INTERPRETATION AND CONCLUSIONS

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this report.

25.1 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

The Rook I property consists of 32 contiguous mineral claims with a total area of 35,065 ha. All claims are 100% owned by NexGen. Six of the 32 claims are subject to a 2% NSR royalty payable to ARC, and a 10% production carried interest with Terra. The NSR may be reduced to 1% upon payment of $1.0 million to ARC.

The Arrow Deposit is located outside of the six claims.

As of 06 December 2012, mineral dispositions are defined as electronic mineral claims parcels within the MARS using a GIS. MARS is a web-based electronic tenure system for issuing and administrating mineral permits, claims, and leases. Mineral claims are acquired by electronic map staking, and administration of the dispositions is also web-based.

As of the effective date of this report, all 32 mineral claims comprising the Rook I property are in good standing, and are registered in the name of NexGen.

Surface rights are a distinct and separate right from subsurface or mineral rights. The Project is located on provincial Crown lands and, as the owner, the province of Saskatchewan can grant surface rights under the authority of the Forest Resources Management Act and the Provincial Lands Act. Granting surface rights for the purpose of accessing the land to extract minerals is done through issuing a mineral surface lease that is subject to the Crown Resource Land Regulations. Mineral surface leases have a 33-year maximum term which may be extended, as necessary.

NexGen does not currently hold surface rights in the project area. Surface rights are obtained after the ministerial review and approval of the EIS and the successful negotiation of a mineral surface lease agreement with the province of Saskatchewan.

RPA is not aware of any environmental liabilities to which the Rook I property is subject. 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 Rook I property.

25.2 Geology and Mineralization

The Rook I property is located along the southwestern rim of the Athabasca Basin, a large Paleoproterozoic-aged, flat-lying, intracontinental, fluvial, redbed sedimentary basin that covers much of northern Saskatchewan and part of northern Alberta. The

Athabasca Basin is oval-shaped at surface, with approximate dimensions of 450 km × 200 km. It reaches a maximum thickness of approximately 1,500 m near its centre.

The southwest portion of the Athabasca Basin is overlain by the flat-lying Phanerozoic stratigraphy of the Western Canada Sedimentary Basin, including the carbonate-rich rocks of the Lower to Middle Devonian Elk Point Group, Lower Cretaceous Manville Group sandstones and mudstones, moderately lithified diamictites, and Quaternary unconsolidated sediments.

South of the Athabasca Basin, where Athabasca sandstone cover becomes thin, paleo-valley fill and debris flow sandstones of the Devonian La Loche / Contact Rapids formation (Alberta) or Meadow Lake (Saskatchewan) formation unconformably overlie the basement rocks.

The Paleoproterozoic basement rocks of the Taltson Domain unconformably underlies the Athabasca Basin and the Phanerozoic stratigraphy within the extents of the Rook I property. The crystalline basement rocks comprise a spectrum of variably altered mafic to ultramafic, intermediate, and local alkaline rock types. The most abundant basement lithologies consist of gneissic, metasomatized-feldspar-rich granitoid rocks, and dioritic to quartz dioritic and quartz monzodioritic gneiss, with lesser granodioritic and tonalitic gneiss.

Mineralization occurs at the following seven locations on the property, and is exclusively hosted in basement lithologies below the unconformity that is overlain by the Athabasca Group.

- Arrow Deposit
- South Arrow Discovery
- Harpoon occurrence
- Bow occurrence
- Cannon occurrence
- Camp East occurrence
- Area A occurrence

Of the seven mineralized locations, the Arrow Deposit has undergone the most investigation.

The Arrow Deposit is currently interpreted as being hosted chiefly in variably altered porphyroblastic quartz-flooded quartz-feldspar-garnet-biotite (± graphite) gneiss. Mineralization at the Arrow Deposit is defined by an area comprised of several steeply-dipping shears that have been labelled as the A0, A1, A2, A3, A4, and A5 shears. The A0 through A5 shears locally host HG uranium mineralization.

The Arrow Deposit is considered to be an example of a basement-hosted, vein type uranium deposit.

25.3 Exploration, Drilling, and Analytical Data Collection in Support of Mineral Resource Estimation

Since acquiring the Project property in December 2012, NexGen has carried out exploration activities, such as the following.

- Ground gravity surveys.
- Ground resistivity and DCIP surveys.
- Airborne magnetic-radiometric-VLF survey.
- Airborne VTEM survey.
- Airborne ZTEM survey.
- Airborne gravity survey.
- Radon-in-water geochemical survey.
- Ground radiometric and boulder prospecting program.

Diamond drilling programs have also helped to test several targets on the property, which is what resulted in the discovery of the Arrow Deposit in drill hole AR-14-01 (formerly known as RK-14-21) in February 2014.

As of the effective date of this report, NexGen and previous owners of the Rook I property have completed 754 holes totalling 380,051 m. From 2013 to the effective date of this report, NexGen has completed 716 holes, totalling 374,917 m of drilling.

Three types of drill core samples are collected at site for geochemical analysis and uranium assay.

- One-metre and 0.5-metre samples taken over intervals of elevated radioactivity, and one metre or two metres beyond radioactivity.
- Point samples taken at nominal spacings of five metres or 50 m for infill holes, which is meant to be representative of the interval or of a particular rock unit.
- Composite samples in the Devonian and Athabasca sandstone units where one-centimetre long pieces are taken and spaced throughout sample intervals ranging from one metre to 10 m long.

All samples are analyzed at SRC Geoanalytical Laboratories by inductively coupled plasma optical emission spectroscopy (ICP-OES) or inductively coupled plasma mass spectroscopy (ICP-MS) for 64 elements, including uranium. Samples with low radioactivity are analyzed using ICP-MS. Samples with anomalous radioactivity are analyzed using ICP-OES.

NexGen personnel perform full core bulk density measurements on-site using standard laboratory techniques. In mineralized zones, average bulk density is measured from samples at 2.5 m intervals, where possible (i.e., approximately 20% of all mineralized samples). In order for density to be correlated with uranium grades across the data set, each density sample directly correlates with a sample sent to SRC for assay.

Samples are also collected for clay mineral identification using infrared spectroscopy in areas of clay alteration. Samples are typically collected at five-metre intervals. and consist of centimetre-long pieces of core selected by a geologist.

Based on the data validation and the results of the standard, blank, and duplicate analyses, RPA believes that the assay and bulk density databases are of sufficient quality for Mineral Resource estimation at the Arrow Deposit.

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

In RPA's opinion, the drilling, core handling, logging, and sampling procedures meet or exceed industry standards, and are adequate for the purpose of Mineral Resource estimation.

25.4 Metallurgical Test Work

The metallurgical test program included a bench test program (March 2018), a pilot plant program (July 2018), and paste backfill testing (July 2018).

Additional testing was conducted to optimize the process design. This testing included a gypsum purification study (July 2019), belt filter dewatering and washing tests (July 2019), and centrifuge dewatering tests (August 2019).

An advanced phase of paste backfill testing (2019) was conducted using samples generated from the 2018 pilot plant program.

The average recovery estimate used in the FS was determined from the pilot testing.

The results of the trade-off study / test work of dewatering and washing technologies using belt filters determined that belt filters will not be the preferred technology for the UNLR. These results and follow-up analysis lead to CCD thickening being selected for this application over belt filters and centrifuges. Belt filters were selected for gypsum, YC, NLR, and the tailings mixture.

The trade-off study / test work of dewatering and washing technologies using centrifuges produced acceptable results. However, belt filters were selected over centrifuges; as a result, there are no longer any centrifuges in the design.

The results from the 2019 paste backfill testing were used to optimize the paste plant design.

No major deleterious elements or elemental concentrations have been identified to date.

25.5 Mineral Resource Estimates

The Mineral Resource estimate for the Project was based on results from 521 diamond drill holes (excluding 45 drill holes drilled at the South Arrow Discovery). It was reported using a \$50/lb U_3O_8 price, at a cut-off grade of 0.25% U_3O_8.

Measured Mineral Resources total 2.18 Mt at an average grade of 4.35% U_3O_8, for a total of 209.6 Mlb of U_3O_8.

Indicated Mineral Resources total 1.57 Mt at an average grade of 1.36% U_3O_8, for a total of 47.1 Mlb U_3O_8.

Inferred Mineral Resources total 4.40 Mt at an average grade of 0.83% U_3O_8, for a total of 80.7 Mlb U_3O_8.

The effective date of the Mineral Resource estimate is 19 July 2019. Estimated block model grades are based on chemical assays only. The Mineral Resources were estimated by NexGen and audited by RPA. Mineral Resources are inclusive of Mineral Reserves. RPA has noted that the deposit is open in many directions.

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 other than what has been described in this report.

25.6 Mineral Reserve Estimates

The Mineral Reserve estimate is reported using the 2014 CIM Definition Standards. The effective date of the Mineral Reserve estimate is 21 January 2021.

The Mineral Resource for M&I material extends from approximately 290 m below surface to 720 m below surface. The resource dips steeply, averaging between 80° to 85° from horizontal.

The M&I resources are contained within two shear zones, the A2 and A3. The A2 shear zone is located FW to the A3 with a natural pillar between the two shear zones.

The M&I resources are contained within two shear zones: the A2 and A3 shears. The A2 shear zone is located FW to the A3 with a natural pillar between the two shear zones.

Mineral Reserves are reported using a cutoff grade of 0.30% U_3O_8, assumptions of transverse stope and longitudinal retreat stope mining methods, an average mining rate of 1,207 t/d, a total combined dilution of 33.8%, and an overall mining recovery of 95.5% for longhole mining and ore development.

Factors that may affect the Mineral Reserve estimate include the following.

- Commodity price assumptions.
- Changes in local interpretations of mineralization geometry and continuity of mineralization zones.
- Changes to geotechnical, hydrogeological, and metallurgical recovery assumptions.
- Input factors used to assess stope dilution.
- Assumptions that facilities such as the UGTMF can be permitted.
- Assumptions regarding social, permitting, and environmental conditions.
- Additional infill or step out drilling.

25.7 Mine Plan

Access to the UG Arrow Deposit will be via two shafts, an 8.0 m diameter Production Shaft (intake air) and a 5.5 m diameter Exhaust Shaft (second egress). Access to the working will be from the Production Shaft with stations on 500 and 590 Levels. Levels will be spaced 30 m apart UG and will be connected via an internal ramp.

The estimated mill capacity is targeted at 1,300 t/d of ore. To realize this target, the mine plan will include longhole production on four separate mining blocks, with multiple stopes available per block. The estimated production rates of the stopes range from 250 t/d to 300 t/d. This will require approximately five stopes to be active to achieve 1,300 t/d. Additionally, the mill will be capped at a head grade of 5.0% U_3O_8. The grade varies within each stope, allowing for better ability to control the head grade from the mine.

The 2021 FS proposes that a UGTMF be used to store tailings and gypsum. A portion of the tailings will be used to generate paste backfill for use in the UG mining operations.

The Rook I mine will be developed using a high degree of equipment mechanization, and equipment will be captive in the mine. Each of the main pieces of equipment will have remote operating capability, and in some cases will be autonomous to reduce radiation exposures.

The mine plan will use conventional mining techniques and conventional equipment. The proposed implementation of an UGTMF is innovative for uranium mines as it facilitates permanent storage of process tailings UG and active reclamation of the mine.

25.8 Recovery Plan

The planned process plant will use conventional technology for the uranium industry, consisting of the following.

- Grinding
- Leaching
- Liquid–solid separation via counter current decantation
- SX

- YC precipitation
- YC calcining
- YC packaging
- Paste tailings backfill plant

25.9 Infrastructure

The infrastructure planned for the Project is common in the uranium industry and conventional UG mining, and is designed for operations in a cold climate.

An on-site power plant will be required due to the high capital costs associated with running a power line to the site.

25.10 Environmental, Permitting and Social Considerations

25.10.1 Environmental Considerations

Environmental and social baseline studies have been completed and/or are underway. The results of the baseline studies to date have not identified any known environmental or social issues that could materially impact NexGen's ability to extract the Mineral Resources or Mineral Reserves.

25.10.2 Ore and Special Waste Stockpiles

Stockpile design is applicable for LOM operations requirements. The ore and special waste stockpiles will be double-lined with HDPE.

25.10.3 Waste Rock Storage Areas

WRSA design is applicable for LOM storage requirements. Both lined and unlined storage areas are envisaged, for storage of PAG material and NPAG respectively.

25.10.4 Water Management

The water management infrastructure has been designed to maximize the diversion of non-contact surface runoff water away from site developed features. Precipitation events and snow melt runoff that come in contact with disturbed infrastructure areas, or potential contact zones, will be captured, collected, and directed to respective impound areas identified as site runoff ponds or collection areas.

Water management structures include two storm water runoff ponds, dikes, and six process ponds.

25.10.5 Closure Planning

A conceptual decommissioning plan has been developed in accordance with Provincial and Federal regulations and guidelines. The detailed decommissioning plan will be prepared during the permitting and approvals process. Once finalized, the plan and an application for approval to decommission will be submitted to Provincial and Federal authorities.

25.10.6 Permitting Considerations

There are several federal and provincial regulatory approvals required for a new uranium mine and mill development. The CNSC and the SMOE are currently coordinating their respective requirements for the EA. Following the EA, several other federal and provincial permits and licences will be required.

No provincial drilling permits were required for 2020. Standard operational-related permits (such as the Industrial Lease and Water Use permits) will be kept current and/or renewed for 2021.

25.10.7 Social Considerations

NexGen has developed project-specific plans for regulatory, Indigenous, and public engagement. In 2019, NexGen entered into study agreements with several Indigenous communities in the region. NexGen is continuing to work on the development of IBAs with several Indigenous communities.

25.11 Markets and Contracts

Marketing studies and commodity price assumptions are based on research and forecasts by UxC.

NexGen is considering selling production from the Project through all avenues of selling uranium including long-term contracts that would be entered into with buyers. It is expected that any such contracts would be within industry norms for such uranium contracts. No contracts have currently been entered into for the Project.

The financial analysis assumes that 100% of uranium produced from the planned Rook I Project can be sold at long-term price of US$50/lb U_3O_8, using an exchange rate of C$1.00 = US$0.75.

25.12 Capital Cost Estimates

The capital cost estimate meets the criteria to be classified as a Class 3 estimate, as defined by AACE International. It has an approximate accuracy of ±15%. All costs included in the estimate are reported in Q4 2020 Canadian dollars.

The capital cost estimate reflects a detailed bottom-up approach based on key engineering deliverables that define the project scope. This scope was described and quantified within material take-offs (MTOs) in a series of line items.

NexGen intends to make its FID midway through Y-4. Capital costs incurred prior to the FID are categorized as pre-development capital costs and are excluded from the economic analysis as sunk costs.

The capital costs are defined as all costs incurred from Year -4, up to Year 1, while sustaining capital costs are costs incurred from Year 1 through to the end of mine life.

Capital costs—including project, sustaining, and reclamation costs, and excluding pre-development capital costs—total $1,573.9 million.

25.13 Operating Cost Estimates

Operating cost estimates were developed to show annual costs for production. Unit costs are expressed as $/tonne processed and $/lb U_3O_8.

LOM operating costs are estimated to be $1,769.8 million, $386.80 per tonne processed, and $7.58 per lb U_3O_8.

25.14 Economic Analysis

The study is based on an assumption of processing of 4,575 kt grading 2.37% U_3O_8 over an eleven-year mine life to produce 233.6 Mlb of recovered YC, using a metallurgical recovery forecast of 97.5%.

On a pre-tax basis, the NPV8% is $5,577.0 million, the IRR is 64.9%, and the assumed payback period is 0.8 years. On a post-tax basis, the NPV8% is $3,465.0 million, the IRR is 52.4% and the assumed payback period is 0.9 years.

The anticipated project cash flow is most sensitive to the price of uranium, head grade, and process recovery. The Canadian to United States dollar exchange rate significantly influences project economics.

25.15 Risks and Opportunities

NexGen and its lead consultants have assessed critical areas of the Project and identified risks associated with the technical and cost assumptions used. The main risks identified in the Project include: assumptions around the prevalence of mineralized material in areas designated for mine infrastructure; assumptions around ground freezing and overall shaft development; adverse ground conditions as they relate to planned mining excavations, material handling systems unable to meet planned and peak production, commissioning of the UGTMF being slower than anticipated resulting in delays to first production, regulatory risks around permitting, and stakeholder engagement, and risks around cost escalation and project execution.

NexGen and its lead consultants performed an opportunities analysis. Opportunities that were recognized included: a potential expansion of Mineral Resources, and corresponding extension of the mine operating life, improvements to the mine extraction factor, reduction in mining operating costs and improved safety by considering remote or autonomous mining equipment, reductions in mining and process water usage through recycling, consider heat recovery opportunities from the acid plant and power plant, evaluate alternative energy options including renewables and connecting to a provincial grid, and advancing critical early works construction packages to streamline overall project execution.

25.16 Conclusions

The proposed mining method is standard throughout the industry. The geometry of the resource, with four separate mining front, would support opportunities to create either a more concentrated or an increase production profile, if required. Deposition of the all the tailings as backfill underground, although not common in the mining industry, will utilize common industry methods.

The results of the FS indicate that NexGen's proposed Rook I Project is technically feasible and shows positive economics under the assumptions presented in this Report. The anticipated Project cash flow is most sensitive to the price of uranium, head grade, and process recovery. The Canadian dollar to United States dollar exchange rate significantly influences Project economics.

The FS supported the conversion of mineral resources to mineral reserves and is considered sufficiently reliable to guide NexGen in a decision to advance to the next phase of the Project development through basic engineering and including de-risking.

26.0 RECOMMENDATIONS

Due to the positive, robust economics, it is recommended to advance the Rook I project to the next phase of engineering. The recommended development path is to continue to advance the environmental assessment and licensing efforts while concurrently advancing key activities that will provide further project definition and reduce project execution timeline risks. Associated project risks are manageable, and identified opportunities can provided enhanced economic value.

Engineering and field investigations should be advanced in support increased certainty of costs and project timelines in preparation for permit approvals and a FID.

This following subsections list the programs that are recommended for the next stage of engineering work for the Rook I Project.

26.1 Basic Engineering

Its is recommended that NexGen proceeds to Basic Engineering. Basic engineering design forms the basis for later successful completion of the detailed engineering, procurement, construction, and commissioning work, and further provides NexGen valuable information to finalize internal discussion and evaluation of the feasibility of the project.

The target for basic engineering to create a Class 2 Estimate along with the related Level 4 Schedule.

The total estimated cost for basic engineering is $30 - 35 million.

26.2 Site Investigation

It is recommended that NexGen proceeds with site investigations to support Basic Engineering, including, but not limited to the following.

- Detailed materials characteristics and quantification assessment to confirm borrow source locations and available volumes of aggregates.
- Drill hole investigations of nuisance mineralization observed in the footwall of Arrow proximal to LOM infrastructure, the quartz vein observed in GAR-18-013 (Exhaust Shaft pilot hole), and the northern extents of the UGTMF.
- Hydrogeological studies to increase NexGen's understanding of the impact of groundwater on the UG mine and mine dewatering requirements.
- Investigate near surface and subsurface conditions in the area of proposed surface infrastructure, focusing on the Mine Terrace and Waste Rock Storage Facility.

The total estimated project cost for the geotechnical, geomechanical, hydrological and surface material assessment is $8-9 million.

26.3 Process Plant Optimizations

The following studies are proposed.

- Loaded strip acid recovery
- Gypsum belt filter optimization
- YC particle size enhancement
- YC belt filter optimization
- Clarifier optimization
- Paste plant optimization
- Geo-metallurgical characterization
- Mine water pre-treatment technology

The total estimated cost for this program is $1.0-1.5 million.

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