Arrow Deposit, Rook I Project

Saskatchewan

NI 43-101 Technical Report on Pre-feasibility Study

Prepared for:
NexGen Energy Ltd.

Prepared by:
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Effective Date:
5 November, 2018

Project Number:
199053

wood.

CERTIFICATE OF QUALIFIED PERSON

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I, Paul O'Hara, P.Eng., am employed as a Manager, Process with Wood Canada Limited, 301-121 Research Drive, Saskatoon, SK S7N 1KS, Canada

This certificate applies to the technical report titled "Arrow Deposit, Rook I Project, Saskatchewan, NI 43-101 Technical Report on Pre-feasibility Study", dated 5 November 2018, (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 32 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.

I am responsible for Sections 1.1, 1.2, 1.3, 1.4, 1.5, 1.10, 1.16 to 1.18, 1.20 to 1.26; Sections 2.1 to 2.6; Section 3; Sections 4.1, 4.2, 4.5 to 4.11; Section 5; Section 13; Section 17; Section 18; Section 20; Sections 21.1, 21.2.1, 21.2.3 to 21.2.8, 21.3.1, 21.3.3 to 21.3.5, 21.4; Section 22; Section 23; Sections 24.1, 24.3; Sections 25.1, 25.2, 25.5, 25.9 to 25.11, 25.13 to 25.17; Sections 26.1, 26.3.3, 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 pre-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: 20 December, 2018

"Signed and sealed"

Paul O'Hara, P.Eng.

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CERTIFICATE OF QUALIFIED PERSON

Suite 501, 55 University Ave
Toronto, ON, M5J 2H7
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I, Jason Cox, P.Eng., am employed as a Principal Mining Engineer and Executive Vice President, Mine Engineering, with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7.

This certificate applies to the technical report titled "Arrow Deposit, Rook I Project, Saskatchewan, NI 43-101 Technical Report on Pre-feasibility Study", effective date 5 November, 2018 (the "technical report").

I am a Professional Engineer in the Province of Ontario (Reg. #90487158). I graduated from Queen's University, Kingston, Ontario, Canada, in 1996 with a Bachelor of Science degree in Mining Engineering.

I have worked as a Mining Engineer for a total of 23 years since my graduation. My relevant experience for the purpose of the technical report is:

- Review and report as a consultant on many mining operations and projects around the world for due diligence and regulatory requirements
- Engineering study (PEA, PFS, and FS) project work on many mining projects around the world, including North America
- Operational experience as Planning Engineer and Senior Mine Engineer at three North American mines
- Contract co-ordinator for underground construction at an American mine

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 May 18, 2017.

I am responsible for Sections 1.1, 1.3, 1.13, 1.14, 1.19, 1.24, 1.26; Sections 2.1 to 2.6; Section 3; Section 15; Section 19, Sections 25.1, 25.7, 25.12, 25.16; Sections 26.1, 26.2, 26.3.1, 26.3.2, 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 previously co-authored a technical report on the Rook I Project:

- 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

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: 20 December, 2018

"Signed and sealed"

Jason Cox, P.Eng.

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CERTIFICATE OF QUALIFIED PERSON

Suite 501, 55 University Ave
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I, David Robson, P.Eng, am employed as a Senior Mining Engineer with Roscoe Postle Associates Inc. located at Suite 501, 55 University Ave, Toronto, ON, M5J 2H7.

This certificate applies to the technical report titled "Arrow Deposit, Rook I Project, Saskatchewan, NI 43-101 Technical Report on Pre-feasibility Study", effective date 5 November, 2018 (the "technical report").

I am a Professional Engineer in the Province of Saskatchewan (Reg. #13601). I graduated from Queen's University, Kingston, Ontario, Canada, in 2005 with a Bachelor of Science degree in Mining Engineering and Schulich School of Business, York University, in 2014 with an MBA degree.

I have practiced my profession for a total of 13 years since graduation. My relevant experience for the purpose of the Technical Report is:

- Review and report as a consultant on mining operations and projects for due diligence and regulatory requirements.
- Engineering study (scoping study, PEA, PFS) project work on mining projects around the world, including North America.
- Operational experience as a Mine Engineer at a Canadian uranium mine.

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 May 18, 2017 and May 16, 2018.

I am responsible for Sections 1.1, 1.2, 1.3, 1.13, 1.14, 1.15, 1.20,1.21, 1.22, 1.23, 1.24, 1.25, 1.26; Sections 2.1 to 2.6; Section 3; Section 15; Section 16; Sections 21.1, 21.2.1, 21.2.2, 21.2.5 to 21.2.8; 21.3.1, 21.3.2, 21.3.5, 21.4; Section 22, Sections 24.2, 24.3, Sections 25.1, 25.7, 25.8, 25.13, 25.14, 25.15, 25.16, 25.17, Sections 26.1, 26.2, 26.3.1, 26.3.2, and Section 27 of the technical report.

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I have previously co-authored a technical report on the Rook I Project:

- 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

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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: 20 December, 2018

"Signed and sealed"

David Robson, P.Eng.

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CERTIFICATE OF QUALIFIED PERSON

Suite 505, 143 Union Boulevard
Lakewood, Co., 80228
USA

I, Mark B. Mathisen, CPG, am a Principal Geologist with RPA (USA) Ltd., located at Suite 505, 143 Union Boulevard, Lakewood, Co., USA 80228

This certificate applies to the technical report titled "Arrow Deposit, Rook I Project, Saskatchewan, NI 43-101 Technical Report on Pre-feasibility Study", effective date 5 November, 2018 (the "technical report").

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

I have worked as a geologist for a total of 23 years since my graduation. My relevant experience for the purpose of the Technical Report is:

- Mineral Resource estimation and preparation of NI 43-101 Technical Reports.
- Director, Project Resources, with Denison Mines Corp., responsible for resource evaluation and reporting for uranium projects in the USA, Canada, Africa, and Mongolia.
- Project Geologist with Energy Fuels Nuclear, Inc., responsible for planning and direction of field activities and project development for an in-situ leach uranium project in the USA. Cost analysis software development.
- Design and direction of geophysical programs for US and international base metal and gold exploration joint venture programs.

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 January 19 to 20, 2016 and January 22 to 25, 2017.

I am responsible for Sections 1.1, 1.3, 1.5, 1.6, 1.7, 1.8, 1.9, 1.11, 1.12, 1.24, 1.26; Section 2; Sections 3.1, 3.2, Sections 4.4, 4.11; Section 6; Section 7; Section 8; Section 9; Section 10; Section 11; Section 12; Section 14; Sections 25.1, 25.2, 25.3, 25.4, 25.6, 25.16; Sections 26.1, 26.2; 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 previously co-authored the following technical reports on the Rook I Project:

- 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

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: 20 December, 2018

"Signed and sealed"

Mark Mathisen, CPG

IMPORTANT NOTICE

This report was prepared as National Instrument 43-101 Technical Report for NexGen Energy Ltd. (NexGen) by Wood Canada Limited, formerly known as Amec Foster Wheeler Americas Limited, and Roscoe Postle Associates, collectively the "Report Authors". The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by NexGen subject to terms and conditions of its respective contracts with the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

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1.0 SUMMARY

1.1 Introduction

Wood Canada Limited (Wood) and Roscoe Postle Associates Inc. (RPA) were retained by NexGen Energy Ltd. (NexGen) to complete a technical report (the Report) on the results of a prefeasibility study (the 2018 Pre-feasibility Study) for the Arrow uranium deposit within the Rook I Project (the Project) in Saskatchewan, Canada.

1.2 Principal Outcomes

The 2018 Pre-feasibility Study is based on an assumption of processing of 3,433 kt grading 3.09% U_3O_8 over a nine-year mine life to produce 228.4 Mlb of recovered yellowcake, using a metallurgical recovery forecast of 97.6%.

Initial capital costs are estimated at C$1,246.9 million. Total capital costs, inclusive of sustaining (C$213.9 million) and closure/reclamation (C$48 million) costs, are estimated at C$1,508.9 million.

Total life-of-mine (LOM) operating costs are estimated to be C$1,328.3 million.

On a pre-tax basis, the NPV8% is C$6,055.2 million, the IRR is 73.6%, and the assumed payback period is 1 year.

On a post-tax basis, the NPV8% is C$3,661.1 million, the IRR is 56.8% and the assumed payback period is 1.2 years.

1.3 Terms of Reference

The Report was prepared to support disclosures in the NexGen news release dated 5 November, 2018, entitled "NexGen Announces 64% Increase In Average Annual After-Tax Cash Flow In Pre-Feasibility Study, After Tax NPV Of $3.7bn, 43% Increase In Indicated Resources, And Initiates The Largest Drill Campaign In Company's History To Expedite Arrow To Feasibility".

The Report uses Canadian English and metric units unless otherwise indicated. Monetary units are in Canadian dollars (C$), except uranium pricing, which is provided in United States dollars (US$).

1.4 Project Setting

The Rook I 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 has 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 with drumlins and lakes/wetlands dominating the northwest and southeast parts of the Project respectively, and lowland lakes, rivers, and muskegs dominating the central part of the Project. The northwest part of the Project 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 masl on drumlins to 480 masl in lowland lakes. The elevation of Patterson Lake is 499 m.

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.

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

The Rook I Project consists of 32 contiguous mineral claims with a total area of 35,065 ha, registered in the name of NexGen Energy Ltd. All claims are in good standing until at least 2035, and the claim that hosts the Arrow deposit, S-113927, is in good standing until 2039.

Six of the 32 claims that make up the Project are subject to a 2% NSR and a 10% production-carried interest. The Mineral Resources and Mineral Reserves do not occur within claims covered by the 2% NSR or 10% production carried interest and therefore the Arrow deposit is free of royalties.

Surface rights are a distinct and separate right from subsurface or mineral rights. To obtain surface rights in support of mining operations, negotiation with the landowner may be required in the case of private property. In the case of Crown lands, a surface permit must be obtained. NexGen currently holds no surface rights in the Project area.

1.6 Geology and Mineralization

The Arrow Deposit is basement hosted and considered to be an example of a vein-style uranium deposit where mineralization occurs in open-space filling as well as chemical replacement. The deposit is hosted predominantly within crystalline basement unit currently interpreted as a semi-pelitic gneiss.

The Rook I Project is situated on the southwestern rim of the Athabasca Basin, a large Paleoproterozoic-aged, flat-lying, intracontinental, fluvial, red-bed sedimentary basin which covers much of northern Saskatchewan and part of northern Alberta. 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 the Proterozoic Taltson Magmatic Zone (TMZ) to the west. The Athabasca Group basal unconformity is spatially related to all significant uranium occurrences in the region, and the Project area straddles a portion of the Athabasca Group basal unconformity.

The Arrow deposit, as defined in the Mineral Resource estimate, consists of several stacked lenses within a 308 m wide zone with an overall strike length of 970 m starting at 110 m from surface and extending to 980 m and remains open in most directions and at depth. Uranium mineralization at Arrow is closely associated with narrow, strongly graphitic, pelitic, and graphitic semipelitic gneiss lithologies thought to represent discrete shear zones. High-grade uranium zones often occur immediately adjacent to sheared and strongly graphitic zones. Of the five recognized main parallel structural shear panels (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 and A3 shears is known as the higher-grade A2 sub-zone (A2-HG) and A3 sub-zone (A3-HG). Mineralization can take the form of open space fillings, or chemical replacement. The principal uranium mineral is uraninite.

Exploration has defined six additional uranium occurrences at Harpoon, Bow, Cannon, Camp East, Area A, and South Arrow that warrant additional work.

1.7 History

Exploration activities at the Rook I properties prior to NexGen Energy's ownership were conducted by the following companies: Bow Valley Company Ltd, Wainoco Oil and Chemicals Ltd, Canada Southern Petroleum and Gas Ltd., Uranerz Exploration and

Mining Ltd., Canadian Occidental Petroleum Ltd., Houston Oil and Gas Ltd., Hudson Bay Exploration and Development Company Ltd., Kerr Addison Mines Ltd., Saskatchewan Mining and Development Corp., Titan Uranium Inc., and Mega Uranium Ltd. These activities consisted of geological mapping, prospecting, lake sediment sampling, soil and rock geochemical sampling, airborne geophysical surveys (magnetic and radiometric, helicopter borne radiometric, fixed-wing transient electromagnetic or MEGATEM, and versatile time-domain electromagnetic or VTEM), ground geophysical surveys (MaxMin II, gravity), and core drilling.

Since 2013, NexGen has completed ground radiometric and geophysical surveys (gravity, DCIP), boulder prospecting program, airborne geophysical surveys (magnetic-radiometric-very low frequency (VLF), VTEM, Z-Axis Tipper EM (ZTEM), gravity) and core drilling. NexGen has also completed Mineral Resource and Mineral Reserve estimates on the Arrow deposit, a preliminary economic assessment in 2017 and the pre-feasibility study that is the subject of this Report in 2018. Drill programs conducted on the Rook I properties have discovered uranium mineralization in the following areas: Area A occurrence in 2013, Arrow deposit in 2014, Camp East, Harpoon and Cannon occurrences in 2016, and South Arrow in 2017.

1.8 Drilling and Sampling

A total of 593 core holes (302,021 m) has been completed on the Project, of which 555 drill holes (296,681 m) was conducted by NexGen. All NexGen drilling has used core methods. Two contractors have been used, Guardian Drilling Corp (2013) and Aggressive Drilling Ltd. (2014–2017).

Each core box was initially surveyed with a Radiation Solutions RS-120 scintillometer to determine if any boxes contained mineralization. Before the core was split for sampling, depth markers were checked, core was carefully reconstructed, washed, geotechnically and geologically logged for lithologies, alteration, structures, and mineralization, rock mass rating (RMR'), resurveyed in detail with a scintillometer, photographed (wet), and marked for sampling. Sampling of the holes for assay was guided by the observed geology and readings from a hand-held scintillometer. Logging and sampling information was entered into a Microsoft Access database template on a computer which was integrated into the Project master digital database on a daily basis.

Core recovery at Arrow has allowed for representative samples to be taken and accurate analyses to be performed. The mineralization in the Arrow deposit is sub-vertical and

true width is estimated to be from 30% to 50% of reported core lengths based on currently-available information.

All NexGen drill hole collars have been surveyed using a differential base station global positioning system (GPS) instrument. The trajectory of all drill holes was determined during drilling with a Reflex instrument in single point mode, which measured the dip and azimuth at 30 m intervals. Both immediately below casing and after completion, all holes at the Arrow deposit were surveyed via Stockholm Precision Tools north-seeking gyro, which measured the dip and azimuth continuously down hole.

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.

Three types of samples are collected for geochemical analysis: point samples, composite samples, and samples in areas of elevated radioactivity. Samples were sent to Saskatchewan Research Council Geoanalytical Laboratories (SRC) in Saskatoon, Saskatchewan. SRC operates in accordance with ISO/IEC 17025:2005 (CAN-P-4E), General Requirements for the Competence of Mineral Testing and Calibration laboratories and is independent of NexGen.

SRC crushed each sample to 60% passing -10 mesh and then riffle split the sample to 200 g with the remainder retained as coarse reject. The 200 g sample was milled to 90% passing -140 mesh. All samples were analyzed at SRC 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 were analyzed using ICP-MS. Samples with anomalous radioactivity were analyzed using ICP-OES. Selected samples were also analyzed for gold, platinum, and palladium using traditional fire assay methods.

The quality assurance and quality control (QA/QC) program used at Arrow included the insertion of standard reference materials (SRMs), blanks, and duplicates into the sample stream. Results from the QA/QC samples were continually tracked by NexGen as certificates for each sample batch were received. If QA/QC samples of a sample batch passed within acceptable limits, the results of the sample batch were imported into the master database.

Results of the QA/QC program were well documented by NexGen. RPA has relied on documentation provided by NexGen in addition to audit of the QA/QC data. RPA considers the QA/QC protocols in place for 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.

Only the contractor and NexGen geological staff were authorized to be at drill sites and in the core processing facility. After logging, sampling and shipment preparation, samples were transported directly from the Project site to SRC by NexGen staff.

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 and inspected 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 found minimal errors and all are most likely due to rounding. No limitations were placed on RPA's data verification process.

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

1.10 Metallurgical Testwork

The metallurgical test program included a bench test program, a pilot plant and paste backfill testing. Testwork samples comprised three composite samples, consisting of low-, medium- and high-grade material, and ten samples of localized deposit areas.

Completed testwork included quantitative evaluation of materials by scanning electron microscopy (QEMSCAN), potential acid generation (PAG), SAGDesin and Bond ball mill index, batch leach, optimization leaching, confirmation and variability, settling, solvent extraction, separating funnel shakeout, stripping, gypsum precipitation, yellowcake

precipitation, preliminary sulfide flotation, and diagnostic gravity separation. Two pilot leaching tests were performed using two different feed samples.

The average recovery estimate used in the 2018 Pre-feasibility Study was determined from pilot plant testwork. Pilot leach testing had uranium extractions of 99.3%. The washing efficiency in the counter current decantation was very high at >99.6%. All other unit operations in the pilot testing had uranium recoveries of >99.6%. Uranium recovery was estimated by evaluating the recovery of the individual circuits and combining these into an overall recovery. Total net uranium recovery is forecast to be 97.6%.

Metallurgical variability testwork consists of 11 leaching tests on samples that had uranium grades ranging from 0.51% U_3O_8 to 8.53% U_3O_8. Of the samples tested, only one is from material that returned lower recoveries that is included in the 2018 Pre-feasibility Study mine plan.

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.

1.11 Mineral Resource Estimation

RPA was supplied with an individual drill hole database for the Arrow deposit in MS Access format, with a close-out date of May 25, 2018. The database included 406 drill holes, of which 14 were not used in estimation as they did not reach target depth.

Geological interpretations supporting the estimate were generated by NexGen and reviewed by RPA. Topographical surfaces, solids and mineralized wireframes were modelled. In total, 162 wireframes were constructed within the Arrow A1, A2, A3, A4 and A5 shear zones. Due to a limited number of drill holes, it was not possible to fully differentiate between the A4 and A5 shears; thus mineralized intercepts in the A5 shear zone were grouped into the A4 shear for the mineral resource estimate presented herein. High-grade domain models were created using a grade intercepts limit ≥1 m with a minimum grade of 5% U_3O_8, although lower grades were incorporated in places to maintain continuity and a minimum 1 m thickness. Low-high-grade domain models were created using a lower-grade intercept limit ≥1 m, with a minimum grade-thickness product of 0.1%m, or 2 m at 0.05%.

Samples were composited to 1 m lengths. High-grade outlier assay values were capped. For samples with no density measurements, the density variable was imputed using a uranium grade polynomial regression. Variograms were constructed, and were used to

support search ellipsoid anisotropy, linear trends observed in the data, and Mineral Resource classification decisions.

Interpolation methods included ordinary kriging (OK) and inverse distance weighting to the second power (ID2). Estimates used a minimum of 2–3 to a maximum of 50 composites per block estimate with a maximum of two composites per drill hole. Hard boundaries were used to limit the use of composites between domains. When the first search was not sufficient to estimate all the blocks, the minimum number of composites required for estimate was reduced by one. All blocks in the domains were populated by pass two. 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.

RPA validated the block model using swath plots, volumetric comparison of blocks versus wireframes, visual inspection, a parallel secondary estimation using inverse distance weighting to the third power (ID3) and statistical comparison of block grades with assay and composite grades. RPA found grade continuity to be reasonable and confirmed that the block grades were reasonably consistent with local drill hole composite grades.

Indicated and Inferred categories were assigned based on the following parameters:

- Indicated Mineral Resources: defined by 25 m by 25 m drill hole spacing and a NN distance of ≤30 m with strong geological continuity between drill hole intercepts
- Inferred Mineral Resources: defined by a drill hole spacing that is >25 m by 25 m and a NN distance of ≤100 m with reasonable continuity assumed between holes.

Reasonable prospects of eventual economic extraction were assessed by RPA using a potential underground mining cut-off grade estimated using assumptions based on historical and known operating costs for mines operating in the Athabasca Basin. The breakeven cut-off grade was estimated using a price of US$50/lb U_3O_8 and based on assumptions for process plant recovery, total operating cost, and incremental component of operating cost. The estimated cut-off grade of 0.25% U_3O_8 is in line with comparable operations with basement hosted mineralization, and similar geologically to Arrow. A minimum mining width of 1 m was assumed using underground mining methods.

1.12 Mineral Resource Statement

The Mineral Resource estimate is reported using the 2014 CIM Definition Standards. The effective date of the Mineral Resource estimate is May 25, 2018. Estimated block model grades are based on chemical assays only. The Mineral Resources were estimated by NexGen and audited by RPA. The Qualified Person for the estimate is Mr. Mark Mathisen, CPG, an RPA employee. Table 1-1 summarizes Mineral Resources based on a $50/lb uranium price at a cut-off grade of 0.25% U_3O_8.

In RPA's opinion, the estimation methodology is consistent with standard industry practice and the Indicated and Inferred Mineral Resource estimates for Arrow are considered to be reasonable and acceptable.

Factors that may affect the resource estimate include: 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 reasonable prospects for eventual economic extraction; assumptions as to social, permitting and environmental conditions; and additional infill or step out drilling or results obtained from the planned 2019 drill program.

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 that are not discussed in this Report.

1.13 Mineral Reserve Estimation

The vertical extent of the Mineral Reserves extends from approximately 355 m below surface to 655 m below surface.

Based on the initial cut-off grade assessment, RPA selected material with a cut-off grade of 0.25% as the basis for which mine planning would occur. RPA used 0.25% U_3O_8 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. RPA notes that a nominal amount of material between 0.03% U_3O_8 (the regulatory limit between benign waste and mineralized material) and 0.07% U_3O_8, which is uneconomic to process, has been included in the mine plan. Although the deposit is relatively insensitive to cut-off grade,

Table 1-1: Mineral Resource Estimate

Classification	Zone	Tonnage (Tonnes)	Grade (U_3O_8 %)	Contained Metal (U_3O_8 lb)
Indicated	A2-LG	1,244,000	0.79	21,700,000
	A2-HG	459,000	17.88	181,000,000
	A3-LG	1,005,000	0.70	15,500,000
	A3-HG	175,000	9.94	38,400,000
Indicated Total		***2,883,000***	***4.04***	***256,600,000***
Inferred	A1	1,510,000	0.72	23,900,000
	A2-LG	1,293,000	0.70	19,900,000
	A2-HG	5,000	12.78	1,400,000
	A3-LG	1,233,000	1.10	30,000,000
	A3-HG	1,000	8.11	200,000
	A4	801,000	0.92	16,300,000
Inferred Total		***4,844,000***	***0.86***	***91,700,000***

Notes to accompany Mineral Resource table:

1. The estimate effective date is May 25, 2018. The Qualified Person for the estimate is Mr. Mark Mathisen, CPG, an RPA employee.
2. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Mineral Resources are reported at a cut-off grade of 0.25% U3O8. The cut-off grade includes considerations of metal price, process plant recovery, and mining, processing and general and administrative costs.
4. Mineral Resources are estimated using a long-term uranium price of US$50 per lb U_3O_8 and a minimum mining width of 1 m. Tonnes are based on bulk density weighting
5. Totals may not sum due to rounding.

RPA recommends that a detailed cut-off grade and grade-control program be evaluated in future studies.

It was assumed that both transverse stope and longitudinal retreat stope mining methods would be used. The assumed mining rate is 1,100 t/d. Total rock dilution of approximately 31% is projected for the longhole stopes. Fill dilution, applied to secondary stopes only, represents an additional 13% applied to secondary stope tonnes. Extraction (mining recovery), used in the 2018 Pre-feasibility Study is 95% for longhole mining and ore development.

Post-blast ore sorting is not considered in the 2018 Pre-feasibility Study, and represents an upside potential area for further head grade control and improvement.

1.14 Mineral Reserve Statement

The Mineral Reserve estimate is reported using the 2014 CIM Definition Standards. The effective date of the Mineral Reserve estimate is May 25, 2018. The Qualified Persons for the estimate are Mr. Jason Cox, P.Eng., and Mr. David Robson, P.Eng., employees of RPA. Table 1-2 summarizes Mineral Reserves based on a $45/lb uranium price at a cut-off grade of 0.25% U_3O_8.

Factors that may affect the Mineral Reserve estimate include: 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 can be permitted; assumptions as to social, permitting and environmental conditions; and additional infill or step out drilling or results obtained from the planned 2019 drill program.

1.15 Mining Methods

BGC Engineering Inc. (BGC) identified eight geotechnical domains in the Arrow deposit area. Domain 1 is overburden, Domains 2 to 4 represent sedimentary geological units and would be encountered during shaft sinking only. Domains 5 to 8 are basement lithologies that have undergone variable intensity of weathering and/or hydrothermal alteration. Within Domain 8 is a distinct hanging wall domain that constrains the hanging wall of the A3 to A5 mineralized zones.

Initial shaft sinking will be through domains D1 through D4, which at the proposed shaft locations consist of 40 m of overburden, 60 m of sedimentary rock, and 25 m of paleo-weathered basement rock with a combined thickness of 125 m. These domains consist of poor to very poor-quality rock masses; however, once these have been temporarily artificially frozen for shaft construction, they are not anticipated to be problematic. Based on the information used in the 2018 Pre-feasibility Study, 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.

Table 1-2: Mineral Reserve Estimate

Category	Tonnage (kt)	Grade (% U_3O_8)	Contained Metal (Mlbs U_3O_8)
Proven	—	—	—
Probable	3,433.0	3.09%	234.1
Total Proven and Probable	***3,433.0***	***3.09%***	***234.1***

Notes to accompany Mineral Reserves table:

1. The estimate effective date is May 25, 2018. The Qualified Persons for the estimate are Mr. Jason Cox, P.Eng., and Mr. David Robson, M.B.A., P.Eng., employees of RPA.
2. Mineral Reserves include transverse and longitudinal stopes, development, and incremental ore.
3. Stopes and ore development were estimated at a cut-off grade of 0.25% U_3O_8. Incremental ore is material between 0.03% U_3O_8 and 0.25% U_3O_8 that must be extracted to access mining areas. A grade of 0.03% U_3O_8 is the regulatory limit for what is considered benign waste and material that must be treated and stockpiled in an engineered facility.
4. Mineral Reserves are estimated using a long-term metal price of US$45/lb U_3O_8, and a 0.75 US$/C$ exchange rate. A minimum mining width of 3.0 m was applied for all longhole stopes. The density varies according to the U_3O_8 grade in the block model. Waste density is 2.464 t/m_3.
5. Totals may not sum due to rounding.

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 of the deposit, in an area that is interpreted to have relatively minimal alteration or fault or shear structures. The 2018 Pre-feasibility Study proposes that the underground tailings management facility (UGTMF) will consist of 88 diamond-shaped excavations each approximately 25 m wide by 25 m long by 60 m high. The excavations will be arranged in a regular pattern with 45 m rib pillars separating primary excavations, and 10 m between primary and secondary openings. 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, 0.84 m^3 of paste tailings will be created, along with 0.31 m^3 of combined waste precipitates. Based on a steady-state production rate, total fill demand is nominally 328,000 m^3 per year for paste tailings, and 121,000 m^3 per year for combined precipitates. Tailings not used for paste backfill will be stored in the UGTMF. A by-

product of the processing will be other waste products such as gypsum. The 2018 Pre-feasibility Study proposes to store the gypsum in secondary chambers within the UGTMF, comprising approximately 50% of the total chamber volume.

The proposed options for dewatering the mine are to use a "dirty water" system to the 500 m level, operating both submersible and centrifugal pump systems. The dewatering system will be capable of collecting and removing all strata and operational process water from the mine infrastructure, ongoing developments, operational stopes, and shaft inflow, and pastefill seepage and collecting it at the main dewatering station sumps located at 500 m level.

Transverse stope mining will be used in areas of higher grade (generally greater than 4% U_3O_8) and wider stopes (generally greater than 10 m), while longitudinal retreat stope mining will be used in areas of lower grade and thinner stope widths. Transverse longhole mining will be done using primary and secondary stoping methods to avoid leaving pillars. The order in which stopes are extracted will be largely driven by the head grade, with the overarching goal of achieving annual production of 30 Mlbs U_3O_8. Primary stopes will be recovered first, followed by primary stopes on two vertical levels above, and 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 vein, for a total of four separate production areas.

The Arrow deposit is planned to be accessed using two shafts. Both shafts will be located in the footwall of the deposit. The first shaft will be used as a production shaft, and for transportation of personnel and materials into the mine. Shaft #1 will be sunk to a depth of 658 m below surface. Shaft #1 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. Shaft #1 will have a permanent headframe and hoisting house. The second shaft will be used as an exhaust ventilation shaft. Shaft #2 will be sunk to a depth of 533 m below surface. An emergency escapeway system will also be installed in Shaft #2.

Eleven level spacings at Rook I are planned at 30 m intervals, sill to sill. Lateral development will be concentrated in three years (Year -1, Year 1, and Year 2). In addition to the lateral development, there will be an internal ramp system that will connect all mining levels.

The ore handling system will begin with scoops loading muck in transverse and longitudinal retreat stopes. The scoops will tram muck to centrally-located ore and waste passes. The bottom of the ore pass will be located on 620 mL where a control system directs ore into the underground ore crushing system. Ore will be directed on to a grizzly equipped with a remotely-operated rock breaker. The underground crusher will reduce ore to <120 mm in size. The crusher has been sized for a throughput of 1,500 t/d, although it will be capable of substantially more than this, if required. Crushed ore will be loaded onto a conveyor and hauled to the shaft for skip loading.

The main waste pass will bypass the underground crusher, as this material will be exclusively development material. Broken and screened material will be loaded onto a conveyor and hauled to the shaft for skip loading. A second waste pass and underground crusher will be located at the UGTMF. At the UGTMF waste pass loadout, a separate grizzly, rock breaker, and crusher will reduce UGTMF waste to <120 mm for skip loading. UGTMF waste will be conveyed to the shaft where it will join the mine development waste.

The ventilation system is designed as a predominately negative or "pull" system. Fresh air will be distributed throughout the mine by three shaft stations from Shaft #1 and an 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 exposures. A raisebore machine will be used for development of ore and waste passes, and internal ventilation raises.

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

1.16 Recovery Methods

The process 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.

Ore will be crushed underground and will be suitably sized to feed the grinding mill. The hoisted ore will be loaded into an ore truck at the headframe. The truck will drive

through a radiometric scanner to confirm ore grade and the delivery location of the ore on the ore pad. Different ore grades and types can be stored in different piles.

A loader operator will deliver ore to the ore feed hopper. A variable speed feeder belt will feed ore from the hopper into the semi-autogenous grind (SAG) mill at a prescribed rate that will be close to the ground ore feed rate to be fed to mill leaching. SAG mill discharge will report by gravity to feed the ball mill. The ball mill will also be fed recycled oversize particles from a classification cyclone that will be situated above the ball mill. Ball mill discharge will report to a pump box that will pump the ore slurry to the classification cyclone/cyclones. The grinding circuit will have tonnage capacity over that required for leach feed. This will allow the grinding circuit to fill the ore storage tank.

The leaching circuit will consist of six mechanically-agitated tanks that will be connected in series. The flow between each tank will be by gravity. The discharge of each tank will be from a baffled upcomer to ensure minimal solids short circuiting in each tank. The tanks in total will provide the target 10-hour residence time to oxidize and dissolve the uranium. It is expected that 99.3% of uranium in ore will dissolve in the leaching circuit.

A counter-current decantation (CCD) circuit will consist of six units. The solutions will pass from one thickener to the next in the opposite direction as the solids slurry, i.e. from CCD 6 to feed CCD 5 etc. Circuit performance will be based on a combination of the amount of uranium in the feed solution, the amount of wash water to be added to CCD 6 feed and 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. The overflow from CCD 1 will feed a pin bed clarifier, and overflow from the clarifier will gravity-flow to the SX feed tank.

There will be four extraction mixer–settler units. In this counter current flow, pregnant aqueous will be fed into mixer 1 and will discharge as barren raffinate from settler 4. Conversely, barren organic will be fed into mixer 4 and will discharge from settler 1 as loaded organic (high uranium content organic). Loaded organic at this point is expected to contain 99.9% of the uranium that has been fed in the pregnant aqueous. Barren raffinate from settler 4 will be pumped to the raffinate tank. As much of the raffinate as possible will be recycled to capture the acid that is contained in the raffinate. Recycling of raffinate will, however, increase the circulating load of contaminant elements. This elevation of contaminant levels will result in the need to bleed some of the raffinate to the effluent treatment circuit.

There will be six strip mixer–settlers. In stripping, barren aqueous strip solution will be used to strip uranium off the organic. The loaded strip will be very concentrated in uranium at 150 g U3O8/L. Lime will be added to increase the pH of the loaded strip solution to remove the acid in the strip solution in preparation to precipitating uranium. Loaded strip solution will be pumped from the loaded strip tank into the first reactor tank of a train of seven tanks. The flow will report from one tank to the other by gravity.

Purified loaded strip from the gypsum clarifier overflow will report to two yellowcake (YC) precipitation tanks. Slurry will feed wash/thickener units, and will then be centrifuged. The yellowcake will discharge the centrifuge at 70% solids and will report to a conveyor that will feed the calciner. 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 calcine bin will feed a packaging system that will load the calcine into 200 L steel drums. Typically, there will be about 100 drums packaged per mill operating day.

CCD thickener 6 underflow slurry will be pumped to the paste plant and into the tails centrifuge feed tank. Discharge cake from the tails centrifuge will be 77–85% solids, and will be fed into a paste mixer with binder and calcium hydroxide. Water will also be added into the paste mixer to ensure that the mixed paste has the prescribed moisture content (67% solids in one metallurgical test).

Feed water will report to two first-stage water treatment reactor tanks and subsequently to water treatment clarifiers. The treated water tank will be a source for the treated water distribution pump that supplies recycled treated water to mine and mill process uses.

Mine sump pumps will discharge to a surface feed settling pond. Water that runs off from potentially-contaminated site areas such as the ore storage pads and from potentially-contaminating uses such as dry and laundry and maintenance shops, will also discharge to the feed settling pond. Water will be pumped from the feed settling pond surface to feed either the grinding circuit or the first-stage water treatment circuit. Effluent monitoring ponds will allow storage of treated effluent until water parameters are assayed and approved. If all of the assays are in the required ranges, approval will be given for the pond to be discharged into the environment.

1.17 Project Infrastructure

The key infrastructure contemplated for the Project includes:

- Underground mine with two shafts

- Underground infrastructure including: material handling systems, maintenance facilities, fuel bay, explosives magazine, ventilation, paste backfill, electrical and communications facilities, underground water supply, dewatering facilities
- Underground tailings management facility
- Surface support infrastructure for the mine, including: headframes and hoist facilities, liquid natural gas (LNG) facilities, and ventilation fans
- Process plant and associated analytical laboratory
- Surface waste rock storage facility, special waste pad and ore pad
- Permanent and construction accommodation camps
- Mine support buildings, including accommodations, maintenance, warehouse and security buildings
- Water management facilities, including storm water runoff pond and six process ponds
- Airstrip.

Due to the high capital costs associated with running a power line to site, the 2018 Pre-feasibility Study design includes an on-site power plant. The power requirement for the site is estimated to be 14 MW. In order to meet the site power requirement with a N+2 design, the Project requires nine generators. The plant will be fuelled by LNG which will be trucked to site.

1.18 Environmental, Permitting and Social Considerations

1.18.1 Environmental Considerations

Environmental baseline studies have been undertaken or are underway to gather information on the current conditions for the biophysical, cultural and socioeconomic environment in the area of and in relation to the Project. These include air quality, acoustic and light, geology and soils, hydrogeology, hydrology, surface water quality, aquatic resources, vegetation, wildlife, cultural studies, and socio-economic factors.

1.18.2 Waste Rock Management Facility

About 5.1 Mm^3 of clean waste rock will be generated over the course of the LOM. This material is considered non-mineralized and non-acid generating and therefore the

storage area will not need to be lined. It is estimated that about 1% of the waste rock brought to surface will be mineralized but will not meet economic criteria. This material is labelled special waste rock and must be stored in a contained area. This area will be dual-lined with high-density polyethylene (HDPE) and will have a leak detection system installed between the liners.

1.18.3 Water Management

For the 2018 Pre-feasibility Study design, water management infrastructure has been designed to maximize the diversion of surface runoff water away from the general site footprint and disturbed area. Water that interacts with the general site footprint and the clean waste rock pile will be captured and controlled. The storm water runoff pond (SWRP) is designed to hold runoff from the immediate plant-site and mine island footprint. All ponds and pads containing mineralized or radiologically contaminated material have been designed to accommodate a 24-hour probable maximum precipitation (PMP) event. The runoff volumes from the general site footprint and disturbed area anticipated from a 24-hour probable maximum precipitation (PMP) storm event will be captured by a constructed dike between the site and Patterson Lake to the north. All other water conveyance and containment structures have been designed to accommodate a 24-hour 100-year storm event as well as the anticipated volumes of water generated under routine and non-routine operating conditions.

Six water storage ponds are planned, and will include four monitoring ponds for treated effluent, one contingency pond and one minewater feed settling pond. Each monitoring pond and the contingency pond is sized for 5,000 m^3 of storage capacity, and will maintain 1 m of freeboard to contain a PMP storm event.

1.18.4 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 practical, surface and underground 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.

1.18.5 Permitting Considerations

There are several Federal and Provincial regulatory approvals required for a new uranium mine and mill development. The Project falls under both Federal and Provincial jurisdiction for Environmental Assessments (EA) which means the Canadian Nuclear Safety Commission (CNSC) and Saskatchewan Ministry of Environment (MOE) – Environmental Assessment Branch (EA Branch) will each require an EA prior to project approval. The CNSC and MOE will coordinate the EA under the guidance of the Canada-Saskatchewan Agreement on Environmental Assessment Cooperation (2005).

The CNSC's licensing process for uranium mines and mills typically follows the lifecycle of a project and includes the four general licensing phases:

- License to Prepare Site and Construct
- License to Operate
- License to Decommission
- License to Abandon.

The Project will also have to consider regulation compliance requirements under approximately 12 separate Federal acts, and about 27 Provincial acts.

NexGen requires the following permits in support of exploration drill programs: aquatic habitat permit; forest protection permit; industrial permit; water use permit. These permits were in place for the 2018 drill programs. NexGen has applied for permit renewals for the four permits in support of the planned 2019 drill program.

1.18.6 Social Considerations

NexGen advised Wood that NexGen is developing a Public and Indigenous Engagement Plan that is specific to the Project. The plan is intended to foster long-term positive

relationships with the public and Indigenous Communities. NexGen will maintain a transparent process whereby issues are understood, considered, addressed where possible and practical, and a defensible scientific analysis is completed whereby concerns can be referred during the EA, allowing the regulatory process to make informed decisions with respect to the Project.

1.19 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. No contracts have currently been entered into for the Project.

The financial analysis assumes that 100% of uranium produced from the planned Arrow mine 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.

1.20 Capital Cost Estimates

The estimate meets the classification standard for a Class 4 estimate as defined by AACE International, and has an intended accuracy of ±30%. The estimate is reported in Q3 2018 Canadian dollars. Table 1-3 outlines the estimated capital cost for designing, supplying, constructing and pre-commissioning the Project.

Mining capital costs primarily comprise the following areas: shaft sinking, lateral mine development, and stationary mine infrastructure. Mine mobile equipment, typically another capital cost area, is assumed to be purchased on a “lease to own” basis, with the costs incurred in the lease payments counted toward operating costs. Process plant costs were divided between process plant directs and process plant construction. Infrastructure costs include provision for the liquid natural gas (LNG) power plant, as well as allocations for 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, engineering, procurement and construction management (EPCM) costs, construction equipment, freight, Owner’s costs and contingency.

Table 1-3: Total Capital Cost Estimate

Description	Units	Cost
Underground mining	C$ millions	303.4
Processing	C$ millions	238.7
Infrastructure	C$ millions	131.8
Subtotal pre-production direct costs	*C$ millions*	*673.9*
Pre-production indirect costs	C$ millions	365.2
Subtotal direct and indirect	*C$ millions*	*1,039.1*
Contingency	C$ millions	207.8
Initial capital cost	***C$ millions***	***1,246.9***
Sustaining	C$ millions	213.9
Closure	C$ millions	48.0
Total	***C$ millions***	***1,508.9***

Note: totals may not sum due to rounding.

Sustaining capital incorporates all capital expenditures after the pre-production period of Year -3, Year -2, and Year -1. Reclamation costs on $48 million have been included in Year 9.

1.21 Operating Cost Estimates

Operating cost estimates were developed to show annual costs for production. Unit costs are expressed as C$/tonne processed and C$/lb U_3O_8. Operating costs were allocated to one of mining, process or general and administration (G&A). LOM operating costs are estimated to be C$1,328.3 million. LOM operating costs are summarized in Table 1-4.

Underground mining takes place during Year -2 to Year 9 (note in Years -2 and -1 underground mining costs are capitalized). Underground mining begins with capital development in Year -2 and the capitalized development continues through Year 8. Process operating costs are primarily composed of labour, power consumption, water and consumables. Consumables consist of reagents, grinding media, mill liners and LNG. An allowance was made for annual maintenance.

Table 1-4: Operating Cost Estimate Summary

Description	LOM Cost (C$ millions)	Average Annual (C$ millions)	Unit Cost (C$/t processed)	Unit Cost (C$/lb U_3O_8)
Mining	537.1	59.7	157.31	2.35
Processing	562.1	62.5	164.65	2.46
General and administration	229.1	25.5	67.11	1.00
Total	***1,328.3***	***147.6***	***389.07***	***5.81***

Note: totals may not sum due to rounding.

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 made for reimbursable fees paid to the CNSC.

1.22 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, the estimation of Mineral Resources and Mineral Reserves, the estimated mine production and uranium recovered, the estimated capital and operating costs, and the estimated cash flows generated from the planned mine production. Actual results may be affected by:

- Differences in estimated initial capital costs and development time from what has been assumed in the 2018 Pre-feasibility Study
- 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 amount 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, 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.

The production schedules and financial analysis annualized cash flow table are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project assumptions as discussed in this Report and may result in changes to the calendar timelines presented and the information and statements contained in this Report.

No development approval has been forthcoming from 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 initial 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 flow projections. Cash flows are taken to occur at the end 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 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 2018 Pre-feasibility Study 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 NPV8% is C$6,055.2 million, the IRR is 73.6%, and the assumed payback period is one year. On a post-tax basis, the NPV8% is C$3,661.1 million, the IRR is 56.8% and the assumed payback period is 1.2 years.

A summary of the LOM cashflow is provided in Table 1-5. Table 1-6 summarizes the economic results of the 2018 Pre-feasibility Study, with the NPV8% base case highlighted.

1.23 Sensitivity Analysis

The cash flow model was tested for sensitivity to variances in head grade, process recovery, uranium price, overall operating costs, overall capital costs and to Canadian to United States dollar exchange rate.

Figure 1-1 illustrates the results of the sensitivity analysis. The anticipated Project cash flow is most sensitive to the price of uranium, head grade and process recovery. Yellowcake is primarily traded in United States dollars, whereas capital and operating costs for the Project are primarily priced in Canadian dollars. Therefore, the Canadian to United States dollar exchange rate significantly influences Project economics.

An extended sensitivity analysis was undertaken to evaluate the Project sensitivity to fluctuations in the uranium price. The results of the extended sensitivity analysis are displayed in Figure 1-2. This graph shows that the Project is robust to price changes, with negative economics only seen when the commodity price reaches US$10/lb U_3O_8.

1.24 Risks and Opportunities

1.24.1 Opportunities

NexGen performed an opportunity analysis. Opportunities that were recognized included: water useage reduction and recycling, reduction in the number of generator sets needed in the power plant through use of renewable energy or waste heat from the acid plant; using leading-edge remote and autonomous equipment to improve project economics, lower radiological exposures, optimize VOD and heating needs; use of ore-sorting technology to reduce waste sent to surface; and slurry thickening to reduce the number of required storage cavities and the amount of clean waste needed to be stored on surface.

Table 1-5: LOM Cashflow Forecast Summary Table

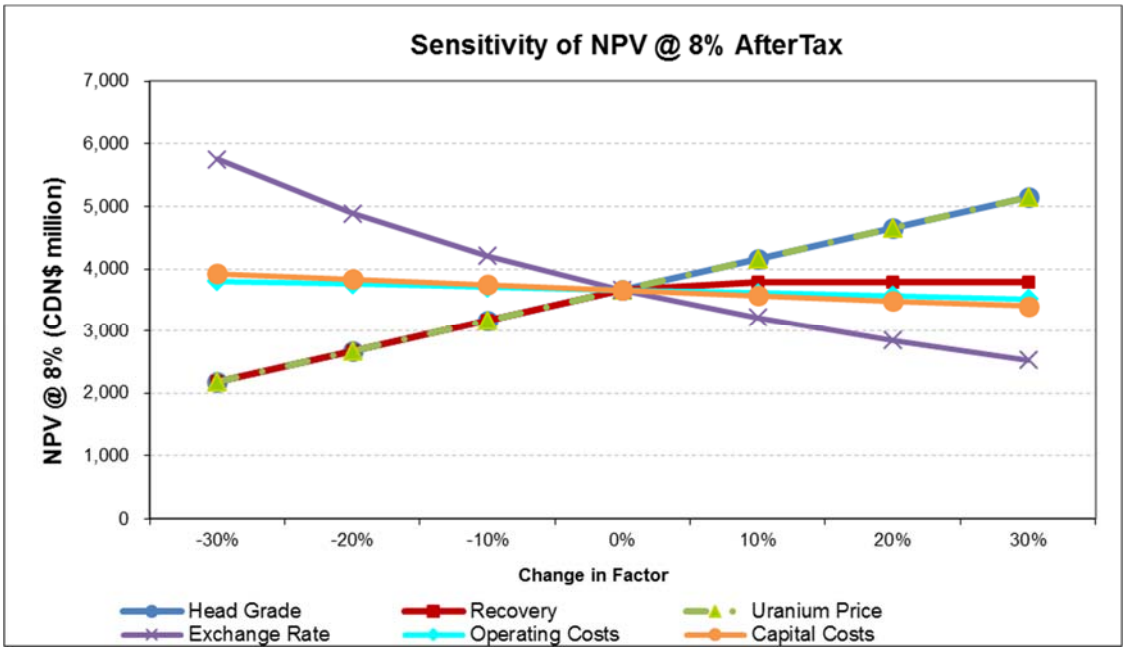
Description	Units	Value
Gross revenue	C$ millions	15,228.7
Less: transportation	C$ millions	(76.7)
Net smelter return	C$ millions	15,152.0
Less: provincial revenue royalties	C$ millions	(1,098.5)
Net revenue	C$ millions	14,053.5
Less: total operating costs	C$ millions	(1,328.3)
Operating cash flow	C$ millions	12,725.2
Less: capital costs	C$ millions	(1,508.9)
Pre-tax cash flow	C$ millions	11,216.3
Less: provincial profit royalties	C$ millions	(1,727.4)
Less: taxes	C$ millions	(2,554.2)
Post tax cash flow	***C$ millions***	**6,934.8**

Table 1-6: 2018 Pre-feasibility Study Forecast Economic Results

Description	Units	Value
Pre-Tax		
Net present value at 8%	***C$ millions***	***6,055.2***
Net present value at 10%	C$ millions	5,232.7
Net present value at 12%	C$ millions	4,534.2
Internal rate of return	***%***	***73.6%***
Payback period	***years***	***1.0***
After-Tax		
Net present value at 8%	***C$ millions***	***3,661.1***
Net present value at 10%	C$ millions	3,140.3
Net present value at 12%	C$ millions	2,698.5
Internal rate of return	***%***	***56.8%***
Payback period	***years***	***1.2***

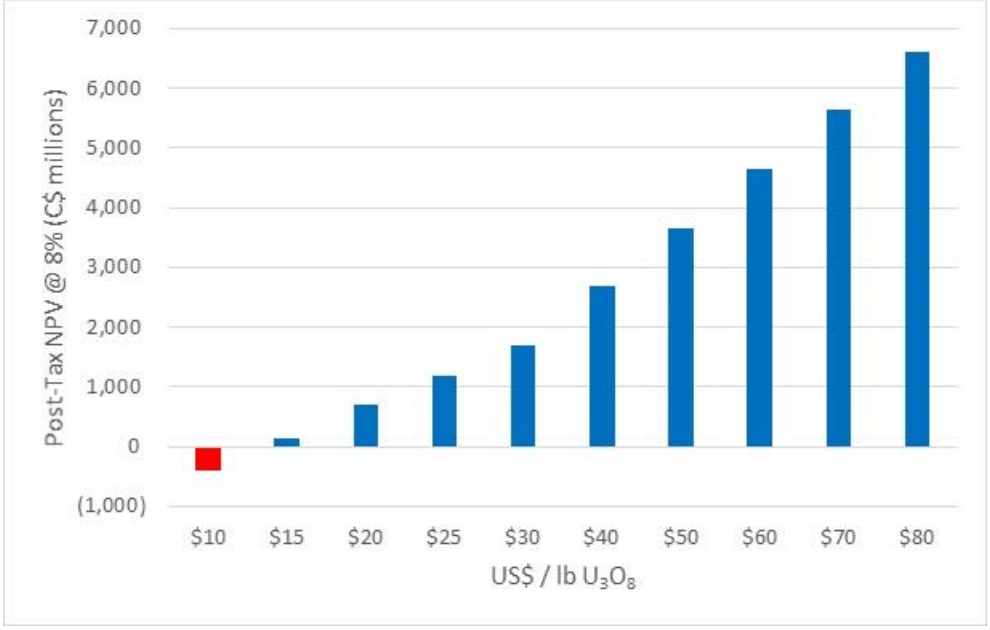
Note: Base case is bolded.

Figure 1-1: Sensitivity Analysis



Note: Figure prepared by Wood, 2018.

Figure 1-2: Project U Price Sensitivity



Note: Figure prepared by Wood, 2018.

1.24.2 Risks

Two separate risk evaluations were undertaken, one by NexGen/Wood, and one by RPA.

- The main NexGen/Wood risks related to: assumptions as to volumes needed in the UGTMF; environmental permitting delays; worker radiation exposure; shaft sinking uncertainties primarily related to temporary artificial freezing down the first 150 m; and increases in costs to key materials and supplies when compared to the 2018 Pre-feasibility Study estimates
- The main RPA risks related to: if shafts or lateral development encounter unknown mineralization of varying grades, this could cause delays, grouting may be required to address shaft water inflows, and the UGTMF concept has not previously been implemented for a uranium mine.

1.25 Interpretation and Conclusions

Under the assumptions presented in this Report, the Project shows positive economics. 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.

1.26 Recommendations

The recommendations proposed are presented as a two-phase work program. Portions of the second recommended work phase are dependent on information generated in the first phase. The Phase 1 recommendations are projected to require a budget of $45 million to complete. The Phase 2 recommendations are estimated at $6.5 million.

Phase 1 recommendations consist of a recommended 125,000 m drill program that will provide support to potentially upgrade Indicated Mineral Resources to Measured, Inferred Mineral Resources to Indicated, and provide further characterization of the geotechnical and hydrological/hydrogeological setting in the planned underground operations and UGTMF areas.

The Phase 2 recommendations include:

- Geotechnical: Confirm ground support requirements for the underground mine and optimum shaft locations, and placement of the UGTMF, and confirm the suitability and design requirements for the selected sites for the WRF and ore stockpile, processing facilities, and infrastructure and conduct a site-specific hazard

assessment for these locations. Some of this work will require information to be available from the Phase 1 drill program

- Hydrological/hydrogeological: Complete evaluations of the impact of groundwater on the underground mine, mine dewatering requirements, ground support requirements and to understand the long-term influence of backfilling underground on the environment; undertake hydrological studies in order to fully understand the water flow rates and to assess the water management and water treatment requirements for the site. Some of this work will require information to be available from the Phase 1 drill program
- Environmental: complete environmental baseline studies including terrestrial and aquatic flora and fauna surveys, soils, sediment, and hydrogeology and testing such as geochemical and water analyses in order to support, with a high degree of confidence, that the activities associated with the mine development can be carried out in a manner that will not degrade the environment.

2.0 INTRODUCTION

2.1 Introduction

Wood Canada Limited (Wood) and Roscoe Postle Associates Inc. (RPA) were retained by NexGen Energy Ltd. (NexGen) to complete a technical report (the Report) on the results of a prefeasibility study (the 2018 Pre-feasibility Study) for the Arrow uranium deposit within the Rook I Project (the Project) in Saskatchewan, Canada (Figure 2-1).

2.2 Terms of Reference

The Report was prepared to support disclosures in the NexGen news release dated 5 November, 2018, entitled "NexGen Announces 64% Increase In Average Annual After-Tax Cash Flow In Pre-Feasibility Study, After Tax NPV Of $3.7bn, 43% Increase In Indicated Resources, And Initiates The Largest Drill Campaign In Company's History To Expedite Arrow To Feasibility".

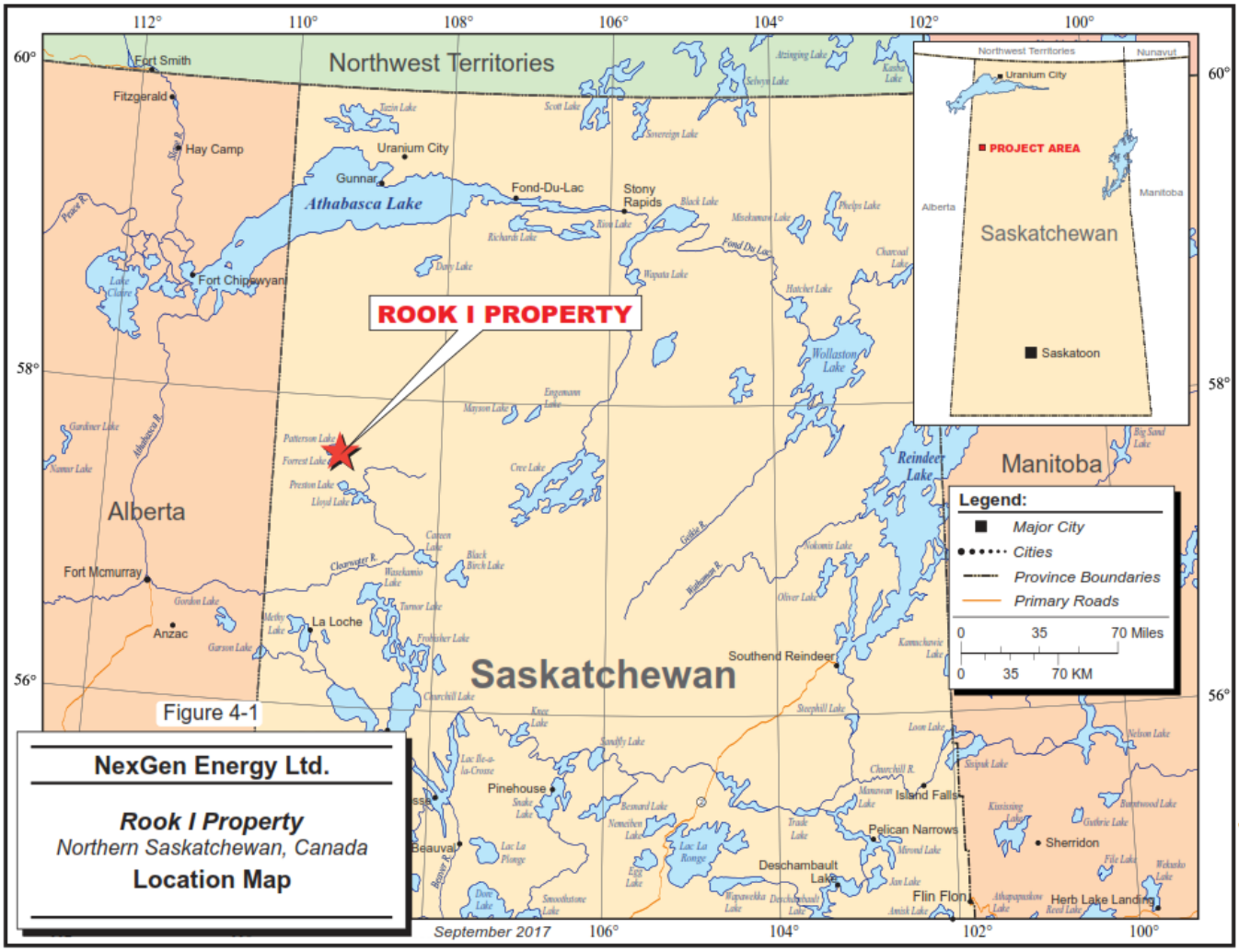
The Report uses Canadian English and metric units unless otherwise indicated. Monetary units are in Canadian dollars (C$), except uranium pricing, which is provided in United States dollars (US$).

2.3 Qualified Persons

The following persons serve as Qualified Persons (QPs) as defined in NI 43-101:

- Mr. Paul O'Hara, P.Eng., Manager Process, Wood
- Mr. Jason Cox, P.Eng., Executive Vice President of Mine Engineering, Principal Mining Engineer, RPA
- Mr. David M. Robson, P.Eng., Senior Mine Engineer, RPA
- Mr. Mark Mathisen, C.P.G., Principal Geologist, RPA.

Figure 2-1: Location Plan



Note: Figure courtesy NexGen, 2018.

2.4 Site Visits and Scope of Personal Inspection

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

Mr. Robson visited the Project on 18 May 2017 and again on 16 May 2018. During the visits, Mr. Robson inspected the camp, visited the core storage area, reviewed the core handling procedures and reviewed project core. Mr. Robson also toured the project site and viewed the area proposed for the plant site and shaft collars. In addition to the site

tour, Mr. Robson had an aerial tour of the Project site to view potential infrastructure areas.

Mr. Cox visited the Project on 18 May 2017. Mr. Cox had an aerial and ground tour of the Project site to view potential infrastructure areas. Mr. Cox also viewed Project core.

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

2.5 Effective Dates

The Report has the following effective dates:

- Date of database close-out for Mineral Resource estimation: 25 May, 2018
- Date of Mineral Resource estimate: 25 May, 2018
- Date of Mineral Reserve estimate: 31 August, 2018
- Date of supply of latest information on mineral tenure: 29 November, 2018
- Date of supply of latest information on drill programs: 22 November, 2018
- Date of financial analysis: 5 November, 2018.

The overall effective date of the Report is the date of the financial analyses and is 5 November, 2018.

2.6 Information Sources and References

The key information source for the technical report is the 2018 Pre-feasibility Study document:

- Wood and RPA, 2018: NexGen Energy Rook I Project, Pre-feasibility Study: report prepared for NexGen Energy, 05 November, 2018, 454 p.

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

Additional information was sought from NexGen personnel where required.

2.7 Previous Technical Reports

NexGen has previously filed the following technical reports on 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.

3.0 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The QPs have relied upon the following expert reports.

3.2 Mineral Tenure, Surface Rights, Royalties and Encumbrances

The QPs have not independently reviewed ownership of the Project area and any underlying mineral tenure, surface rights, encumbrances or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from NexGen and legal experts retained by NexGen for this information through the following documents:

- Boisjoli, T., 2018a RE: Information on Mineral Tenure, Surface Rights, Royalties and Encumbrances: letter from NexGen to Paul O'Hara, Project Manager, Wood Canada Limited, 10 December, 2018, 7 p.

This information is used in Section 4 of the Report. The information is also used in support of the Mineral Resource estimate in Section 14, the Mineral Reserves in Section 15 and the financial analysis in Section 22.

3.3 Taxation

The QPs have relied upon, and disclaim responsibility for, experts retained by NexGen for the taxation information as applied in the financial model, which was sourced from the following document:

- Boisjoli, T., 2018b: Financial Model Review Report – Tax Considerations: letter from NexGen to Paul O'Hara, Project Manager, Wood Canada Limited, 14 December, 2018, 1 p.

This information is used in Section 22 of the Report. The information is also used in support of the Mineral Reserve estimate in Section 15.

3.4 Markets

The QPs have relied upon, and disclaim responsibility for, experts retained by RPA for the marketing and uranium price forecast information, which was sourced from the following document:

- UxC, 2018: Uranium Market Outlook, Third Quarter, 2018: report prepared by UxC, 2018.

This information is used in Section 19 of the Report. The information is also used in support of the Mineral Reserve estimate in Section 15 and the financial analysis in Section 22.

The QPs consider that it is reasonable 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 a vast array of custom services for clients in all areas of the nuclear energy markets. In addition, UxC also prepares special reports on various topics, as well as provides data services, such as nuclear fuel price indicator reporting, including support for the CME/NYMEX UX uranium futures contract. UxC publishes the Ux Weekly and Market Outlook reports on uranium, enrichment, conversion, and fabrication, and nuclear power, as well as publishing the industry-standard Ux Prices, referenced in many fuel contracts.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Rook I 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 lies within portions of NTS map sheets 74F/7, 74F/10, and 74F/11 and is approximately centred at Universal Transverse Mercator (UTM) coordinates of 620,000 mE and 6,385,000 mN (NAD 83, Zone 12N). The Arrow deposit is located at approximate UTM coordinates of 604,350 mE and 6,393,600 mN.

4.2 Property and Title in Saskatchewan

4.2.1 Mineral Title

In Canada, natural resources fall under Provincial jurisdiction. Saskatchewan has four types of subsurface mineral rights (Table 4-1).

All mineral resource rights in the Province of Saskatchewan are governed by the *Crown Minerals Act* and the *Mineral Tenure Registry Regulations, 2012*, that are administered by the Saskatchewan Ministry of the Economy. Mineral rights are owned by the Crown and are distinct from surface rights.

As of December 6, 2012, dispositions are defined as electronic mineral claims parcels within the Mineral Administration Registry System (MARS) using a Geographical Information System (GIS). MARS is an electronic tenure system for issuing and administrating mineral permits, claims, and leases that is web-based. Mineral claims are now acquired by electronic map staking and administration of the dispositions is also web-based.

In order to maintain mineral claims in good standing in the Province of 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.

Mineral claims in good standing may be converted to mineral lease(s) upon application. Mineral leases allow for mineral extraction, have 10-year terms, and are renewable.

Table 4-1: Mineral Rights in Saskatchewan

Mineral Title Type	Notes
Crown mineral rights	Administered by the Saskatchewan Ministry of Energy and Mines, owned by the Saskatchewan government and in some cases by the federal government (where the property is federally owned). Most mineral rights in the province consist of Crown mineral rights.
Freehold mineral rights	Privately-held rights (by individuals or companies) that permit the holder to exploit, mine, or produce any or all minerals lying below the surface of the property or to convey those rights to a third party.
Split rights	Mineral titles where the ownership is split between two separate parties which could include private owners or the Crown
First Nations mineral rights	Property granted to the Saskatchewan First Nations via treaties signed during the 1870s. First Nations title includes ownership of surface and subsurface rights

4.2.2 Surface Rights

Surface rights are a distinct and separate right from subsurface or mineral rights. To obtain surface rights in support of mining operations, negotiation with the landowner may be required in the case of private property. In the case of Crown lands, a surface permit must be obtained.

Surface facilities constructed in support of mineral extraction require a surface lease. Surface leases have 33-year maximum terms and are also renewable.

4.2.3 Royalties

Each owner or joint venture participant in a uranium mine is a royalty payer. Individual interests of a royalty payer are consolidated on a corporate basis for the calculation of royalties applied to the royalty payer's sales of uranium. The royalty system has three components:

- Basic royalty: 5% of gross revenue
- Profit royalty: rates increase from 10% to 15% as net profit increases
- Saskatchewan Resource Credit: a credit of 0.75% gross revenue

4.2.4 Fraser Institute Survey

Wood used the Investment Attractiveness Index from the 2017 Fraser Institute Annual Survey of Mining Companies report (the Fraser Institute survey) as a credible source for the assessment of the overall political risk facing an exploration or mining project in Saskatchewan.

Wood has relied on the Fraser Institute survey because it is globally regarded as an independent report-card style assessment to governments on how attractive their policies are from the point of view of an exploration manager or mining company, and forms a proxy for the assessment by industry of political risk in Ecuador from the mining perspective.

The Fraser Institute annual survey is an attempt to assess how mineral endowments and public policy factors such as taxation and regulatory uncertainty affect exploration investment.

Overall, Saskatchewan ranked second out of the 91 jurisdictions in the survey in 2017 for investment attractiveness, third for policy perception, and second for best practices mineral potential.

4.3 Project Ownership

The Project is wholly-owned by NexGen.

4.4 Mineral Tenure

The Rook I Project consists of 32 contiguous mineral claims with a total area of 35,065 ha. As of November 29, 2018, all 32 mineral claims were in good standing and registered in the name of NexGen Energy Ltd.

The Project formerly consisted of nine larger dispositions which were acquired by NexGen in 2012; however, in 2015, NexGen divided eight of those dispositions into 31 smaller dispositions to accommodate a more efficient spreading of mineral assessment credits over the Project.

All claims are in good standing until at least 2035, and the claim that hosts the Arrow deposit, S-113927, is in good standing until 2039.

Dispositions S-113928 through S-113933 were recorded in 2005 and are subject to minimum work requirements of $25/ha per year. Dispositions S-113903 through

S-113916 were recorded in 2007 and are subject to minimum work credits of $25/ha per year beginning in 2017. All other dispositions comprising the Rook I Project are subject to minimum work requirements of $15/ha per year. 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, with a maximum claim grouping size of 18,000 ha.

Table 4-2 is a claims list, and Figure 4-1 shows the claim (disposition) locations.

4.5 Surface Rights

The Project is located on Provincial Crown land; surface rights are obtained after successful ministerial decision, after an environmental decision and following successful negotiation of a mineral surface lease agreement.

NexGen currently holds no surface rights in the Project area.

Land access for the planned 2019 exploration sites is from a NexGen construction road of about 15 km in length that connects to Highway 955. No land access agreements are required to be negotiated; however, NexGen does pay a small yearly fee for water and forest permits. A new exploration permit is required annually (see also discussion in Section 20).

4.6 Royalties and Encumbrances

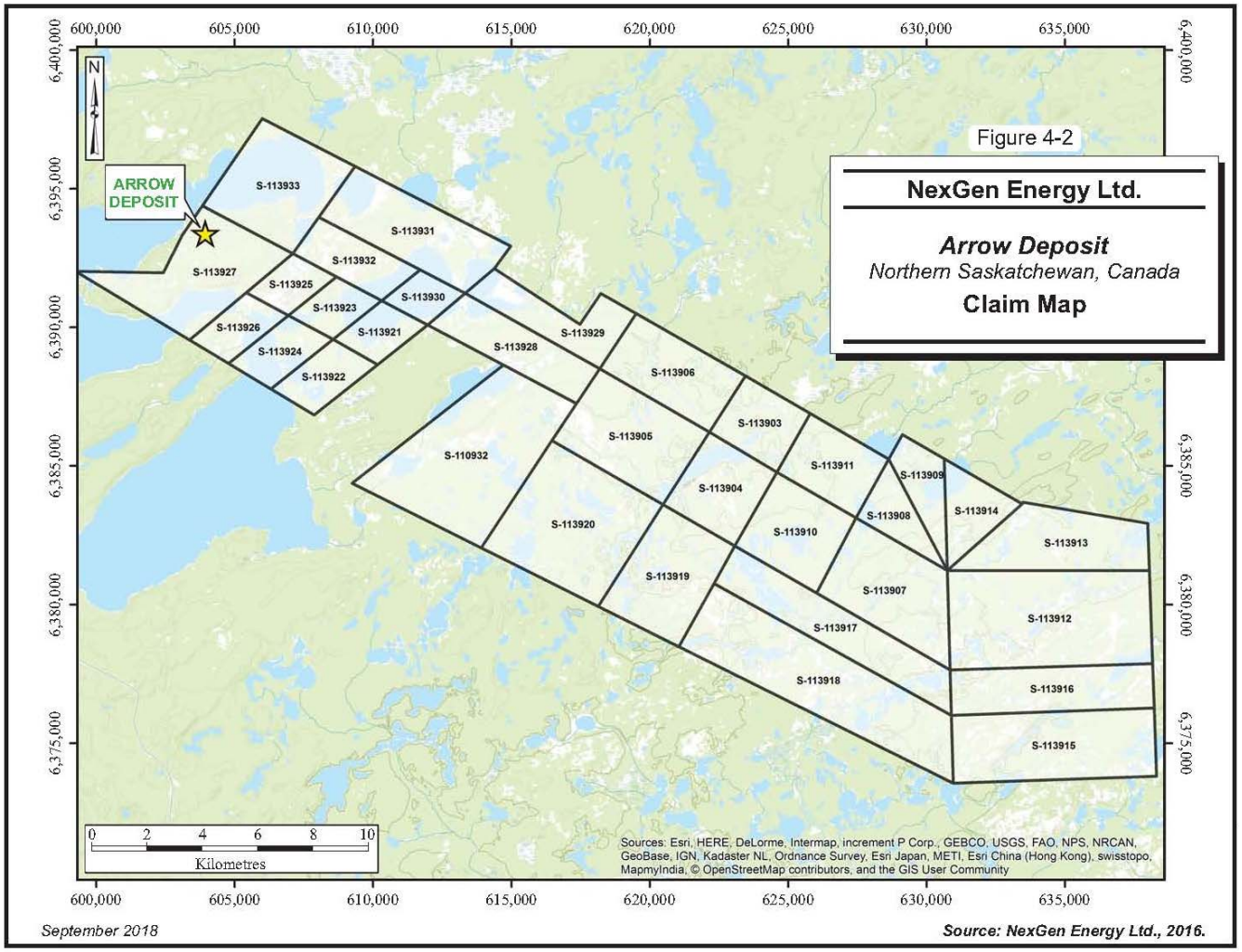
The Arrow deposit is free of royalties although there are six of the 32 claims that make up the Project are subject to a 2% NSR and a 10% production-carried interest. These claims are S-113928, S-113929, S-113930, S-113931, S-113932, and S-113933. The NSR may be reduced to 1% for C$1 million. The 10% production carried interest provides for the owner with a right to 10% of potential future production, provided the owner repays NexGen (from 75% of the holder's share of production) their 10% pro rata portion of the collective expenditure from June 20, 2005. The Mineral Resources and Mineral Reserves do not occur within claims covered by the 2% NSR or 10% production carried interest and therefore the Arrow deposit is free of royalties.

Table 4-2: Claims List

Disposition Number	Previous Disposition Number	NTS	Record Date	Anniversary Date	In Good Standing To	Area (ha)	Annual Expenditure ($)	Subject to Royalty
S-110932	S-110932	74F/11	17-Mar-08	17-Mar-19	13-Jun-39	2,558	63,950	
S-113903	S-110575	74F/10	13-Feb-07	13-Feb-19	13-May-39	673	16,825	
S-113904	S-110575	74F/10	13-Feb-07	13-Feb-19	13-May-39	900	22,500	
S-113905	S-110575	74F/10, 74F/11	13-Feb-07	13-Feb-19	13-May-39	1,432	35,800	
S-113906	S-110575	74F/10, 74F/11	13-Feb-07	13-Feb-19	13-May-39	1,092	27,300	
S-113907	S-110574	74F/10	13-Feb-07	13-Feb-19	13-May-36	1,436	35,900	
S-113908	S-110574	74F/10	13-Feb-07	13-Feb-19	13-May-39	462	11,550	
S-113909	S-110574	74F/10	13-Feb-07	13-Feb-19	13-May-39	492	12,300	
S-113910	S-110574	74F/10	13-Feb-07	13-Feb-19	13-May-39	1,029	25,725	
S-113911	S-110574	74F/10	13-Feb-07	13-Feb-19	13-May-39	800	20,000	
S-113912	S-110573	74F/10	13-Feb-07	13-Feb-19	13-May-39	2,539	63,475	
S-113913	S-110573	74F/10	13-Feb-07	13-Feb-19	13-May-39	1,280	32,000	
S-113914	S-110573	74F/10	13-Feb-07	13-Feb-19	13-May-39	560	14,000	
S-113915	S-110572	74F/10, 74F/7	13-Feb-07	13-Feb-19	13-May-39	1,806	45,150	
S-113916	S-110572	74F/10	13-Feb-07	13-Feb-19	13-May-39	1,187	29,675	
S-113917	S-110934	74F/10	17-Mar-08	17-Mar-19	13-Jun-38	1,385	34,625	
S-113918	S-110934	74F/10, 74F/7	17-Mar-08	17-Mar-19	13-Jun-38	2,481	62,025	
S-113919	S-110933	74F/10, 74F/11	17-Mar-08	17-Mar-19	13-Jun-39	1,328	33,200	
S-113920	S-110933	74F/10, 74F/11	17-Mar-08	17-Mar-19	13-Jun-39	2,098	52,450	
S-113921	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-35	392	9,800	
S-113922	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-35	498	12,450	
S-113923	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-39	378	9,450	

Disposition Number	Previous Disposition Number	NTS	Record Date	Anniversary Date	In Good Standing To	Area (ha)	Annual Expenditure ($)	Subject to Royalty
S-113924	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-37	475	11,875	
S-113925	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-39	360	9,000	
S-113926	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-35	429	10,725	
S-113927	S-110931	74F/11	17-Mar-08	17-Mar-19	13-Jun-39	1,514	37,850	
S-113928	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-39	920	23,000	Yes
S-113929	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-39	811	20,275	Yes
S-113930	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-39	303	7,575	Yes
S-113931	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-36	1,395	34,875	Yes
S-113932	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-39	627	15,675	Yes
S-113933	S-108095	74F/11	17-Mar-05	17-Mar-19	13-Jun-36	1,425	35,625	Yes
					Total	***35,065***	***876,625***	

Figure 4-1: Claims Map



Note: Figure courtesy NexGen, 2016.

Other than as noted in the previous paragraph, NexGen advised Wood that the Project is not subject to any royalties, back-in rights, payments or other agreements and encumbrances.

4.7 Property Agreements

The Project is not subject to any property agreements.

4.8 Permitting Considerations

Permitting is discussed in Section 20.

4.9 Environmental Considerations

Environmental considerations are discussed in Section 20.

4.10 Social License Considerations

Social licence considerations are discussed in Section 20.

4.11 Comments on Section 4

The QP notes:

- NexGen holds a 100% Project interest
- Mineral tenures held are valid and sufficient to support declaration of Mineral Resources and Mineral Reserves
- NexGen currently holds no surface rights in the Project area
- The Mineral Resources and Mineral Reserves do not occur within claims covered by the 2% NSR or 10% production carried interest and therefore the Arrow deposit is free of royalties
- The Project is not subject to any royalties, back-in rights, payments or other agreements and encumbrances.

The QP was advised by NexGen that NexGen is not aware of any significant environmental, social or permitting issues that would prevent future exploitation of the Arrow deposit.

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

5.1 Accessibility

The Project can be accessed via all-weather gravel Highway 955, which travels north-south approximately 8 km west of the Arrow deposit. The highway, which is maintained year-round by the Provincial government, leads from La Loche, the nearest population centre, 150 km to the south of the Project area, to the Cluff Lake mine site (decommissioned), which is 75 km to the north of the Project. La Loche is connected to Prince Albert and Saskatoon by paved provincial highways. Fort McMurray, Alberta, is 180 km southwest of Rook I and can be reached from La Loche via winter road between the months of December and April.

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.

Fixed wing aircraft on floats can land on lakes on and near the Project area. Remote Project areas can be accessed by helicopter.

5.2 Climate

The Project has a sub-Arctic 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. Winters are long and cold, with mean monthly temperatures of below freezing for seven months.

Annual precipitation is approximately 0.45 m with approximately 70% of this occurring as rain during the warmer months and the remainder as snow during the winter.. Freeze-up normally starts in October and break-up occurs in May.

It is expected that mining activities will be conducted on a year-round basis.

5.3 Local Resources and Infrastructure

Fuel, groceries, emergency medical services, and basic construction services are available at La Loche, and 100 km to the south of La Loche at Buffalo Narrows, which also has fixed wing float planes for charter. Approximately 20 km to the north on Highway 955 is the Big Bear Camp outfitter lodge which provides food, accommodation, fuel, other

supplies and basic services. The major centres of Prince Albert and Saskatoon can provide advanced services for mining operations.

Additional information on resources and infrastructure is included in Section 18.

5.4 Physiography

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

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

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

Wildlife species common to the area include moose, deer, black bear, wolf, woodland caribou, and all other mammal species commonly found in boreal forest ecosystems. Common recreational fish species include walleye, lake trout, northern pike, whitefish, burbot and perch.

5.5 Comments on Section 5

Any future underground mining operations are expected to be operated year-round.

There is sufficient suitable land available within the mineral tenure held by NexGen for infrastructure such as waste disposal, process plant and related mine facilities.

A review of the existing power and water sources, labour availability, and transport options indicates that there are reasonable expectations that sufficient labor and infrastructure will be available to support exploration activities and any future mine development.

6.0 HISTORY

6.1 Exploration History

A summary of the exploration history is provided in Table 6-1.

6.2 Production

There has been no production from the Project.

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 Rook I Project area. 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 Rook I 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) Saskatchewan Mining and Development Corp. (SMDC, now Cameco)	Canoxy had claims CBS 4745, 4756, 4747, 4748 covering most of the area of current dispositions S-108095, S-110931 and S110932. Houston Oil and Gas Ltd. had one claim (CBS 5680) covering part of claim S-110575. HBED had two small claims covering S-110933. Kerr had claims covering S-110573 and S-110574. SMDC/Cameco had MPP 1076 (later CBS 8807) which covered part of S-108095. 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 done 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	24 holes on what is now S-110573 and S-110574. No significant results were returned.
1978–1980	Canoxy	Drilled 41 holes, but only 27 of these are on current dispositions comprising the current Project. Drilling did not intersect any uranium mineralization.
1980–1982	SMDC/Cameco	Drilled 13 holes, PAT-01 to PAT-13 on what is now S-108095; identifies the Bow occurrence. The mineralization and alteration were reported to be similar to that seen at unconformity-associated uranium deposits in the Athabasca Basin

Year	Operator	Comment
1982	HBED	Two holes drilled on claims which cover part of what is now S-110933. Intersected graphitic gneisses but no radioactivity.
2005–2008	Titan Uranium Inc. (Titan)	Carried out airborne EM surveys, MegaTEM and VTEM, which detected numerous strong EM anomalies. A ground MaxMin II survey in 2008 confirmed the airborne anomalies
2012	Mega Uranium Ltd. (Mega)	Pursuant to a mineral property acquisition agreement between Mega and Titan dated February 1, 2012, Mega acquired all nine dispositions comprising the Rook I Project. A gravity survey was completed over 60% of S-108095 and S-110931 which defined several regional features and some more local smaller scale features. At the same time Mega undertook sampling of organic rich soils and prospecting in the same area as the gravity survey. No soil geochemical anomalies or radioactive boulders were found.
2012	NexGen	Acquires Mega's interest.
2013–2018	NexGen	Completes ground gravity surveys, ground DCIP surveys, an airborne magnetic-radiometric-very low frequency (VLF) survey, an airborne Versatile Time-Domain EM (VTEM) survey, an airborne Z-Axis Tipper EM (ZTEM) survey, an airborne gravity survey, a radon-in-water geochemical survey, and a ground radiometric and boulder prospecting program, core drilling. Discovers Area A occurrence in 2013, Arrow deposit in 2014, Camp East, Harpoon and Cannon occurrences in 2016, and South Arrow in 2017. After discovery of Arrow deposit, completes additional drilling, Mineral Resource estimates, a preliminary economic assessment in 2017, and a pre-feasibility study in 2018.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

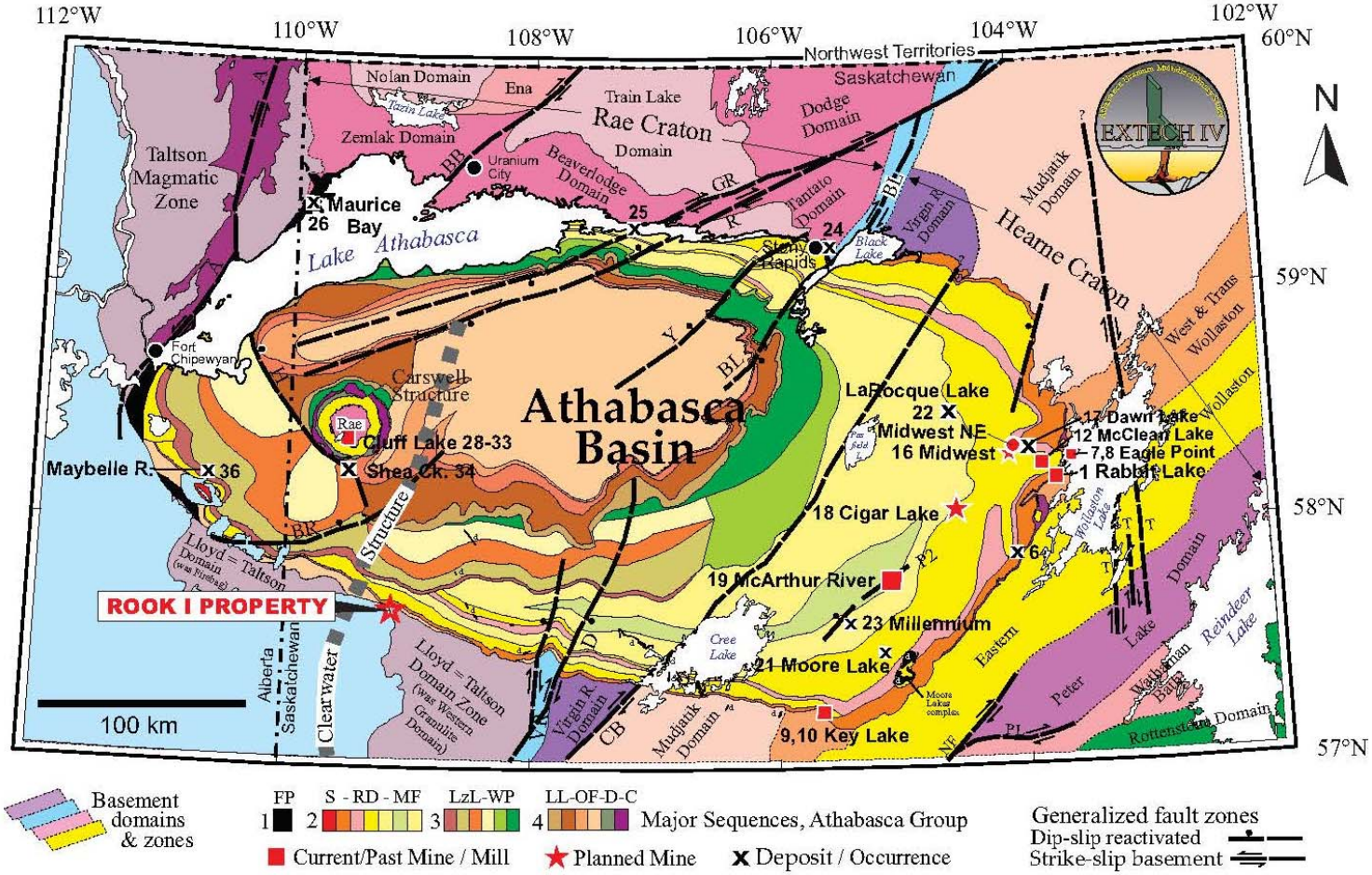
The Rook I Project is situated on the southwestern rim of the Athabasca Basin (Figure 7-1), a large Paleoproterozoic-aged, flat-lying, intracontinental, fluvial, red-bed sedimentary basin which covers much of northern Saskatchewan and part of northern Alberta (Jefferson et al., 2007). The Athabasca Basin is oval shaped at surface with approximate dimensions of 450 km by 200 km and reaches a maximum thickness of approximately 1,500 m near the centre. It consists principally of unmetamorphosed sandstones with local conglomerate beds that are collectively known as the Athabasca Group.

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 the Proterozoic Taltson Magmatic Zone (TMZ) to the west (Card et al., 2007). The Rae Province consists mostly of metasedimentary supracrustal sequences as well as granitoid rocks. In contrast, the Hearne Province consists primarily of granitoid gneisses with interleaved 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 ranging from a few centimetres to up to 220 m thick where fluid migration was aided by fault zones (MacDonald, 1980). Paleoweathered profiles usually 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 of the Western Canada Sedimentary Basin comprising mudstones, siltstones, and sandstones.

Figure 7-1: Regional Geology Plan



Note: Figure prepared by RPA, 2018, after Jefferson and Delaney (2007). Mines, deposits and occurrences other than the Rook 1 Project area are held by third-parties.

7.2 Project Geology

Within the Project area, the TMZ consists chiefly of granitic, granodioritic, tonalitic, dioritic, and locally gabbroic gneisses (Figure 7-2).

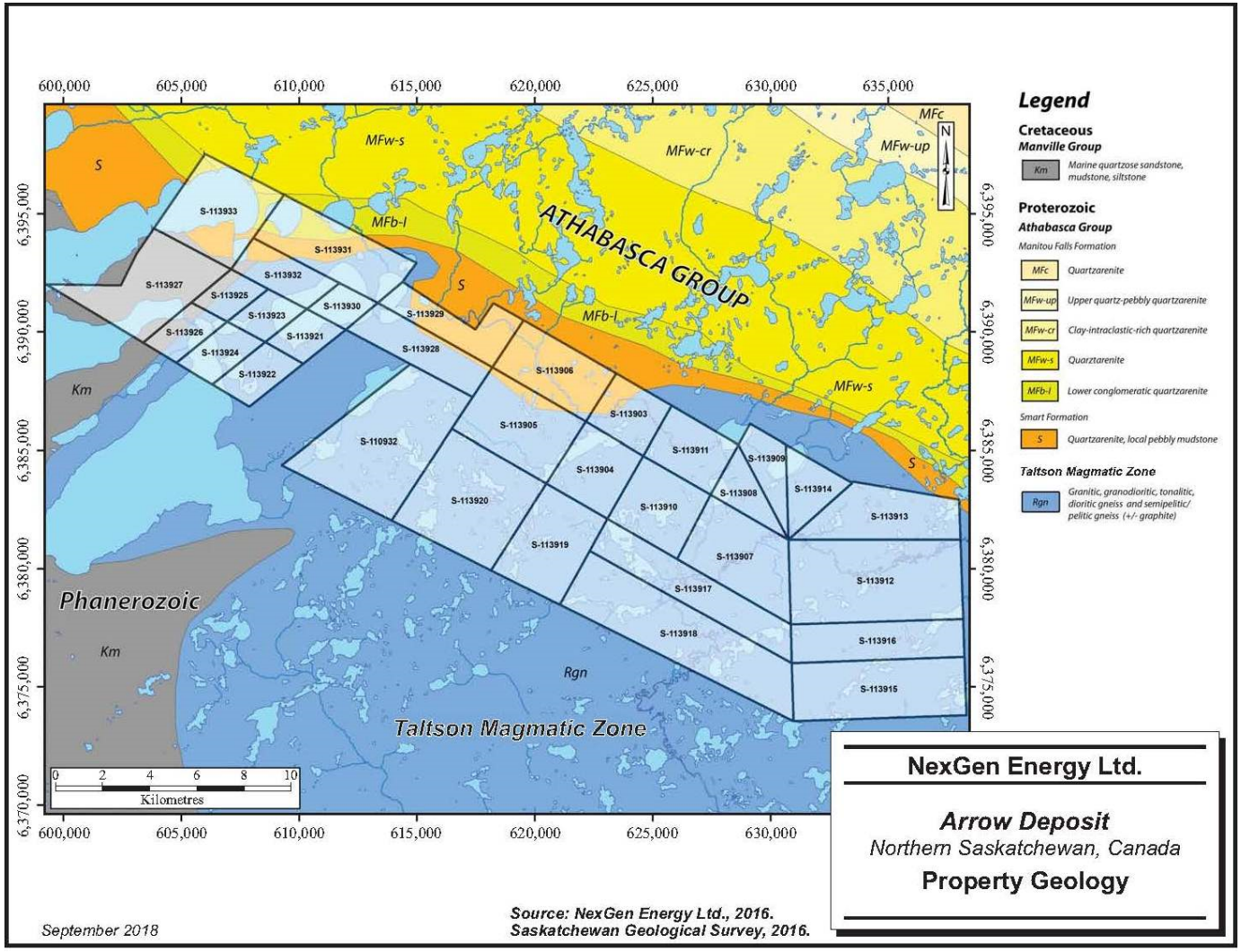
There are also local bodies of graphitic and chloritic semipelitic to pelitic gneisses that typically occur as discontinuous, elongate, north-northeast trending lenses and schlieren ranging from less than one kilometre to greater than 10 km in length (Grover et al., 1997). These paragneiss bodies are the chief 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 Project area straddles the Athabasca Group basal unconformity. Overlying the basement rocks are flat-lying sandstones of the Athabasca Group. Where intersected in 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 Manville Group and Devonian La Loche Formation overlie the Athabasca Group and basement rocks on portions of the western side of the Project, and above the Arrow deposit. The Manville Group is characterized by non-marine and marine shales and sandstones. A coal-bed marker horizon at the base of the Mannville Group is often observed in drill core. The La Loche Formation consists of arenitic to arkosic sandstones and conglomerates.

The Project and surrounding area are covered by Pleistocene glacial deposits composed of sand, Athabasca Group sandstone boulders, and rare basement and Manville Group boulders. Glacial geomorphological topographic features are common and 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. Over the Arrow deposit, glacial overburden is approximately 60 m thick.

Figure 7-2: Project Geology



Note: Figure courtesy NexGen, 2018.

Mineralization is currently known to occur at seven locations within the Project (Figure 7-3), and is exclusively hosted in basement lithologies below the unconformity with the Athabasca Group:

- Arrow deposit
- South Arrow occurrence
- Harpoon occurrence
- Bow occurrence
- Cannon occurrence
- Camp East occurrence
- Area A occurrence.

7.3 Deposit Descriptions

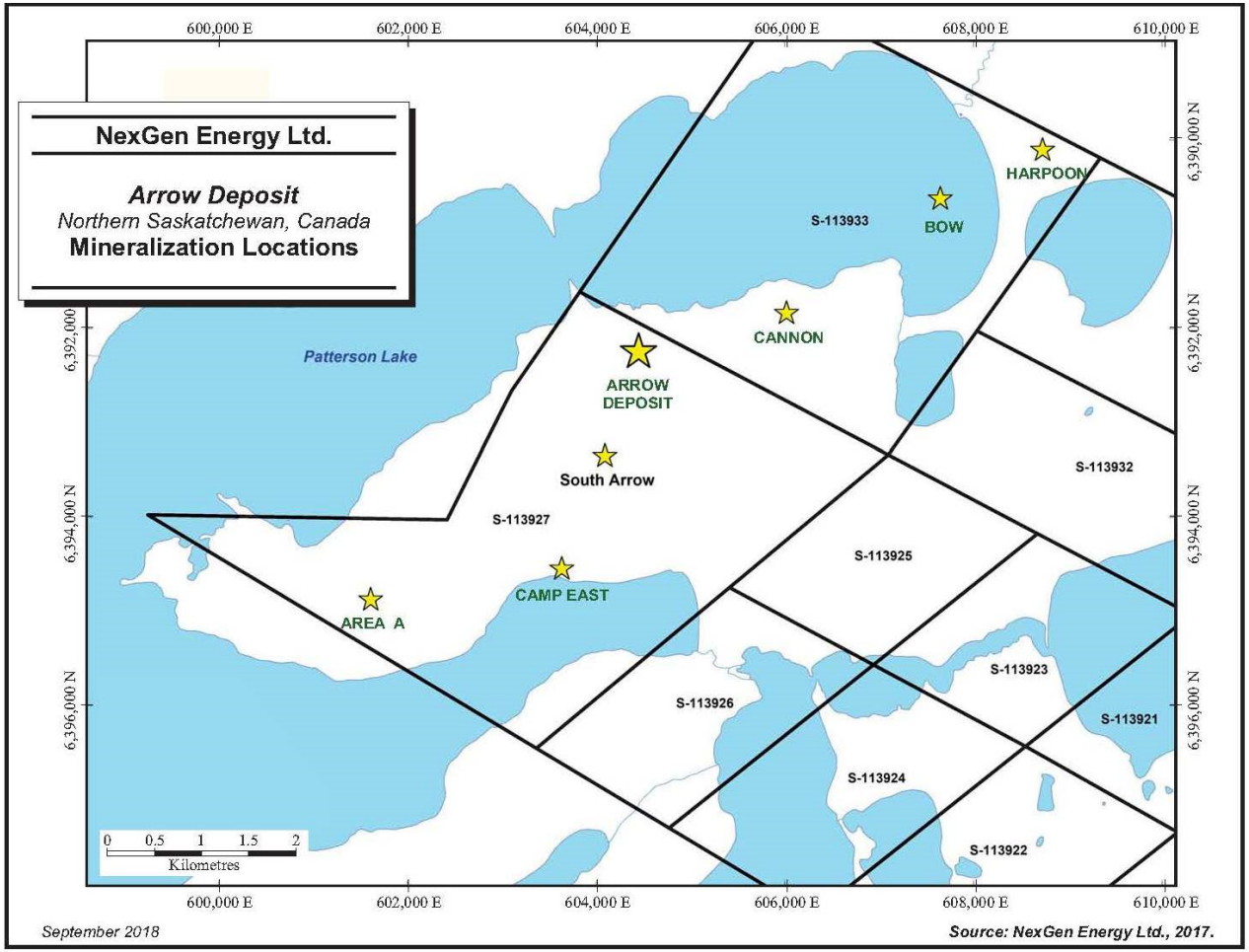
7.3.1 Arrow Deposit

The Arrow deposit, as defined in the Mineral Resource estimate, consists of several stacked lenses within a 308 m wide zone with an overall strike length of 970 m starting at 110 m from surface and extending to 980 m and remains open in most directions and at depth.

Uranium mineralization at Arrow is closely associated with narrow, strongly graphitic, pelitic, and graphitic semipelitic gneiss lithologies thought to represent discrete shear zones. High-grade uranium zones often occur immediately adjacent to heavily-sheared and strongly graphitic zones.

The Arrow deposit is currently interpreted as being hosted chiefly in semipelitic gneiss composed almost solely of quartz and garnet porphyroblast pseudomorphs which are now almost exclusively chlorite, hematite, illite, or sudoite. Other minor mineral phases present include plagioclase, potassium feldspar, biotite, muscovite, and amphibole, in varying concentrations. Local bodies of pelitic gneiss have also been observed. This lithology is distinct from semipelitic gneiss as it is defined by lower concentrations of quartz. The geology of the immediate area of the Arrow deposit is also marked by the presence of a large sill-like intrusive body containing granitic to gabbroic gneisses commonly cross-cut by mineralization. The main foliation present in the Arrow area trends towards the northeast and has vertical to sub-vertical dips.

Figure 7-3: Deposit and Occurrence Locations



Note: Figure courtesy NexGen, 2017.

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

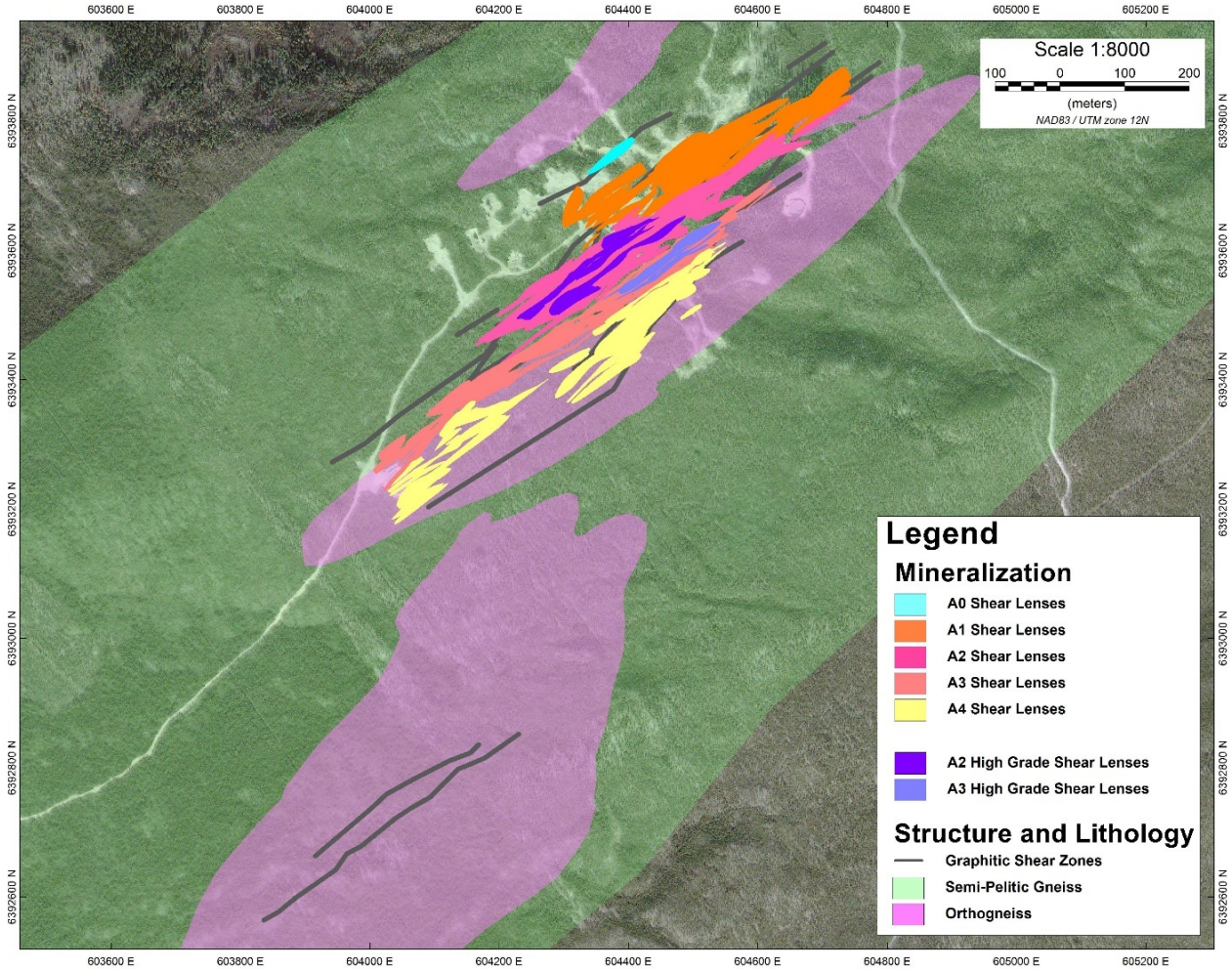
- Quartz–sericite–sudoite–illite alteration: a pervasive alteration assemblage that nearly completely replaces the host rock, although pre-alteration textures are often preserved
- Hematite alteration: pervasive brick red-coloured
- Dravite: occurs in centimetre to decimetre wide breccia vein bodies beginning tens of metres from high grade uranium mineralization and increasing in size and frequency closer to mineralization
- Carbon: Carbon buttons are commonly observed in association with dravite
- Drusy quartz: Centimetre sized veins that occur ubiquitously in the vicinity of the deposit. Where proximal to high grade mineralization, these veins are often pink coloured.

Currently, mineralization occurs within five discrete, parallel shear panels referred to as the A1 though A5 shears (Figure 7-4). Each shear panel is approximately 50 m wide and contains a number of narrow graphitic shear zones that are oriented parallel to foliation striking at approximately 050° to 060° and dipping vertically to sub-vertically. These graphitic shear zones are host to the uranium mineralized lenses and pods which are also oriented parallel and sub-parallel to the regional foliation.

Slickensides observed on fault faces within the graphitic shear zones close to high-grade uranium mineralization show two general orientations, an older dip-slip orientation and a younger overprinting strike-slip/oblique-slip orientation. This suggests at least two distinct plunge directions.

Of the five recognized main parallel structural shear panels (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 and A3 shears is known as the higher-grade A2 sub-zone (A2-HG) and A3 sub-zone (A3-HG).

Figure 7-4: Arrow Deposit Geology



Note: Figure courtesy NexGen, 2018.

Two key but contrasting types of uranium mineralization occur at Arrow:

- Open-space fillings: Open-space fillings include massive uraninite bodies interpreted to be uranium veins, and breccia bodies where the matrix is comprised nearly exclusively of massive uraninite. Uranium veins and breccias typically range in thickness from less than 0.1 m to greater than 1 m and display sharp contacts with the surrounding wall rocks. Individual uranium veins usually occur at parallel to sub-parallel orientations to the regional foliation, however, at least one set of veins cross-cuts the regional foliation. Clasts present in uranium breccias at Arrow are typically fragments of the immediate wall rocks and often contain additional disseminated uraninite mineralization. Uranium breccias occur as both clast-supported and matrix-supported forms, with the latter typically hosting higher grades. Both styles of open-space filling mineralization are categorized by high uranium grades
- Chemical replacement styles: Chemical replacement types of mineralization present at Arrow include disseminated, worm-rock and near complete to complete replacement styles. Disseminated mineralization is typically associated with strong to intense hydrothermal alteration (discussed below) where uraninite occurs as fine- to medium-grained anhedral crystals and crystal agglomerates spread throughout the host in concentrations of typically less than five modal percent. Worm-rock style mineralization is named for the wormy texture it by definition displays, 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 to complete uraninite replacement of the host rock has also been observed at Arrow. These zones range in thickness from <0.1 m to >1.0 m and, in contrast to open-space fillings, show gradual contacts. Near-complete to complete replacement bodies also often contain centimetre sized vugs which may once have been garnet porphyroblasts, pseudomorphs of which are common in the host rocks. The presence of vugs in this style of mineralization and in some zones interpreted to be uraninite veins suggests that in at least some places, the veins may actually be the result of chemical replacement and not open-space filling.

Uranium mineralization 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 often observed.

7.4 Prospects/Exploration Targets

7.4.1 Harpoon Occurrence

The Harpoon occurrence is located 4.7 km northeast of the Arrow deposit and has been traced over a strike length of 340 m. The mineralization is foliation parallel, striking towards the northeast at approximately 035° to 045° and dipping towards the southeast at approximately 60° to 70°. The occurrence has currently been drilled to within 27 m of the northeast boundary of the Rook I Project.

Basement lithologies observed in the area of mineralization include both orthogneiss and paragneiss of varying composition. The occurrence is currently exclusively basement-hosted and occurs within a chloritic and graphitic shear zone that is heavily clay altered.

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

7.4.2 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 paragneisses with compositions ranging from psammitic to pelitic. Quartzite 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.

7.4.3 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 semi-pelitic gneiss, pelitic gneiss, quartzite and orthogneiss, with relatively narrow intervals of chloritic and graphitic mylonite, the latter of which host the low-grade uranium mineralization discovered to date.

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

7.4.4 Camp East Occurrence

The Camp East occurrence is located approximately 2.3 km south–southwest of the Arrow deposit.

Lithologies in the area include semi-pelitic to pelitic gneiss and orthogneiss. Chloritic and locally graphitic shear zones with widths ranging from 1 m 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 area including:

- 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 up to 12 m.

7.4.5 Area A Occurrence

Area A is situated approximately 3.5 km southwest of the Arrow deposit. Visible pitchblende was identified within a strongly hematite altered breccia. The 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. Further drilling is currently being considered.

7.4.6 South Arrow Occurrence

The South Arrow is located 400 m southeast of the Arrow deposit.

The South Arrow, as defined in the Mineral Resource estimate, consists of five narrow fractures varying from one metre to 10 m thick with a strike length of 290 m and vertical extent of 185 m starting at 10 m below the surface. Mineralization strikes towards the northeast at approximately 045˚ and dips towards the southeast between 70˚ and 83˚.

Uranium mineralization at the South Arrow is exclusively basement-hosted, and lithologies observed in the area include both orthogneiss and paragneiss of varying

composition. 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 as fracture coatings.

7.5 Comments on Section 7

The knowledge of the Arrow deposit setting, lithologies, mineralization and alteration controls on uranium grades is sufficient to support Mineral Resource estimation.

8.0 DEPOSIT TYPES

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

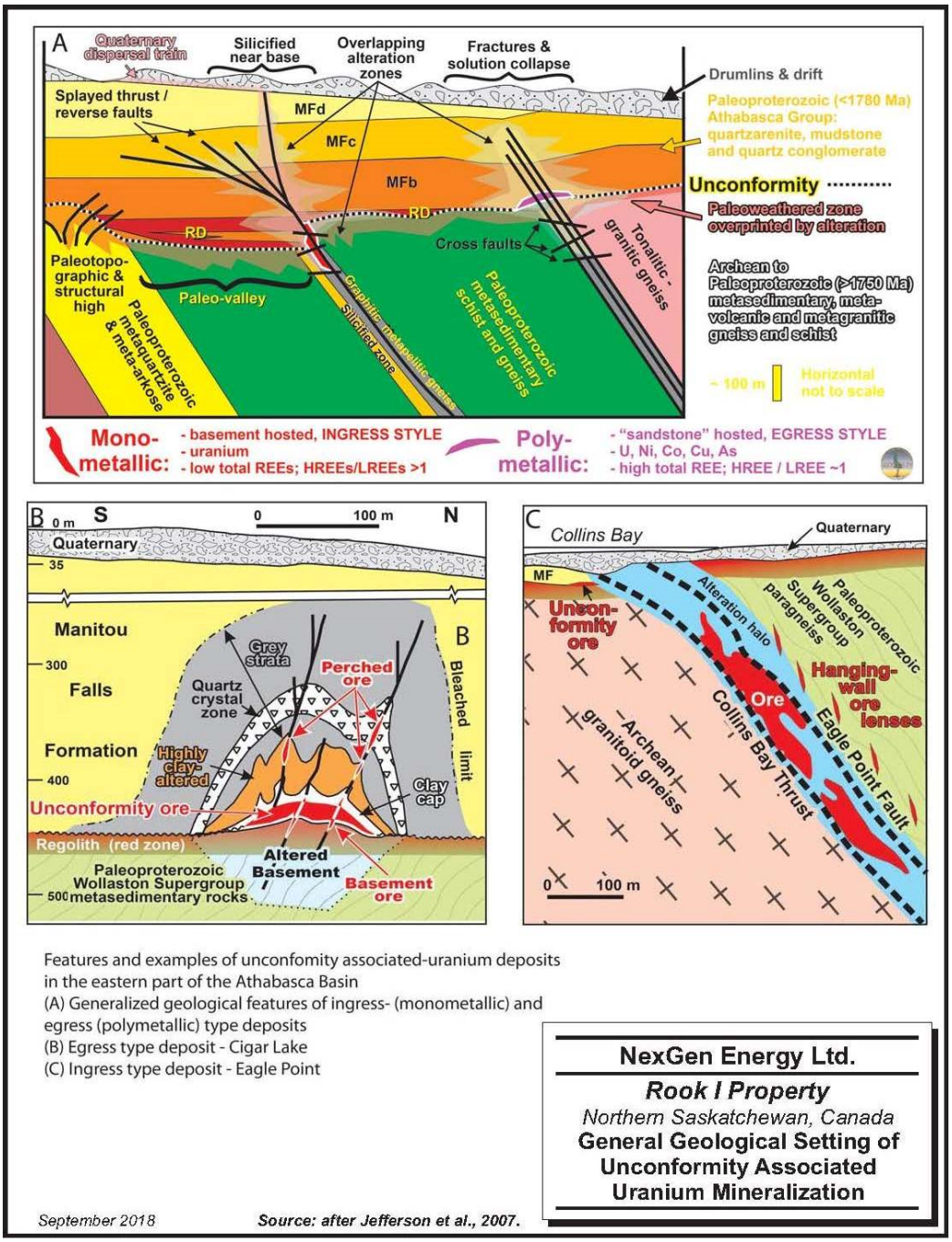
At numerous locations in Saskatchewan, uranium deposits have been discovered at, above, and below the Athabasca Group unconformity. Mineralization can occur hundreds of metres into the basement or can be perched up to 100 m above in the sandstone. At Arrow, no uranium has been identified at or above the unconformity. Massive veins have been discovered in the basement at depths ranging from immediately below the unconformity to 700 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. Such structures are thought to represent both important fluid pathways as well as 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 uranium deposits have been identified in the Athabasca Basin (Figure 8-1 and Figure 8-2), namely egress type and ingress type.

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, cobalt, arsenic, and lead in addition to uranium. Ingress-style mineralization is typically monometallic, containing nearly exclusively uranium.

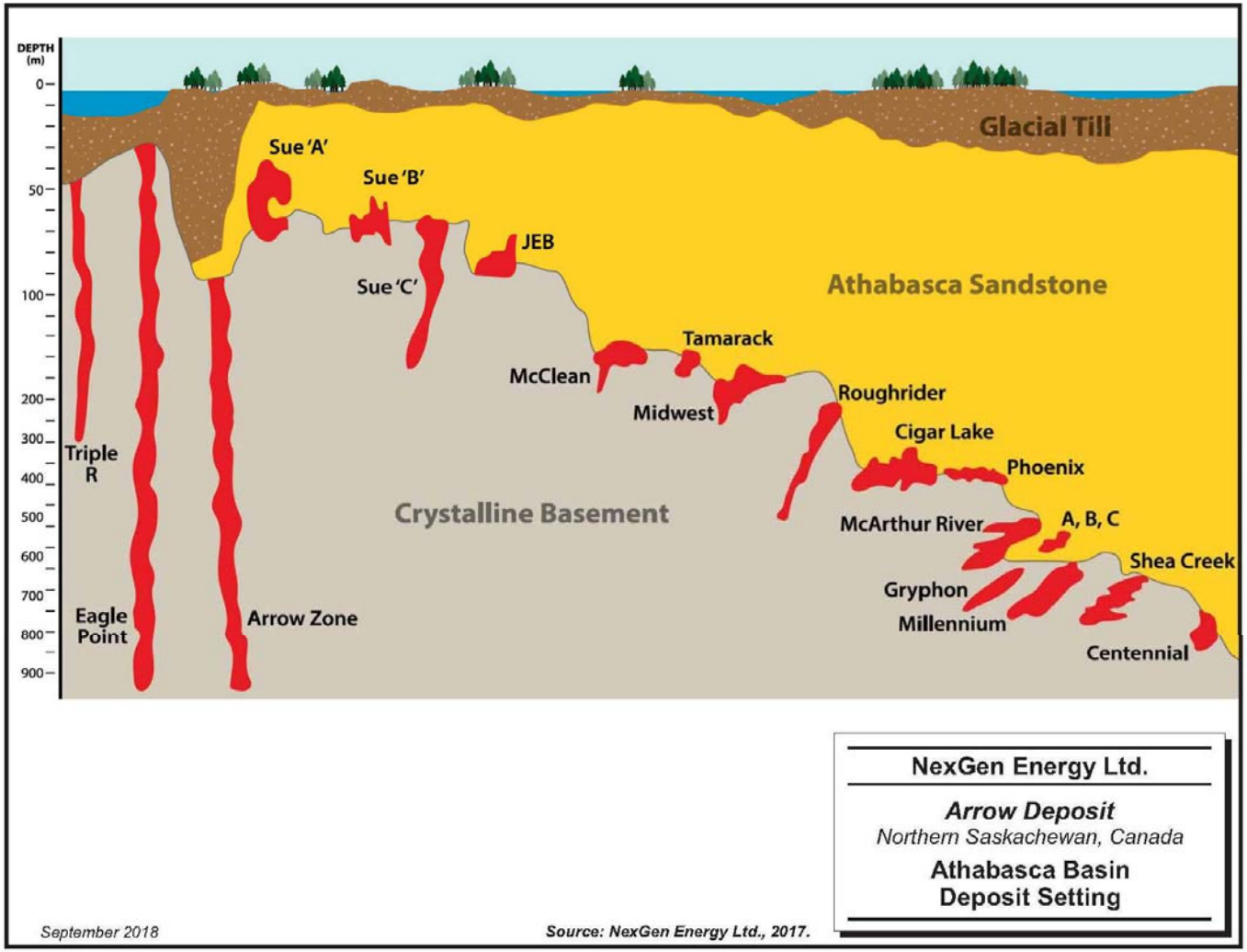
Unconformity-associated uranium deposits of the Athabasca Basin typically display extensive hydrothermal alteration halos, especially in the sandstones above major deposits where relatively higher porosity/permeability allowed for increased fluid flux. Where mineralization is basement-hosted like the Arrow deposit, alteration is typically confined to structures in the basement.

Figure 8-1: General Geological Setting of Basement Hosted Vein-Type Uranium Mineralization – Type C Ingress



Note: Figure prepared by RPA, 2018, after Jefferson et al., (2007).

Figure 8-2: Athabasca Basin Deposit Settings



Note: Figure courtesy NexGen, 2018.

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.

8.1 Comments on Section 8

The basement hosted vein-type Type C uranium deposit model shown in Figure 8-1 is a reasonable basis for the design of additional exploration programs.

9.0 EXPLORATION

9.1 Grids and Surveys

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

9.2 Geological Mapping

There is limited basement outcrop in the Project area; therefore, geological mapping or outcrops is not a primary exploration tool.

In 2014, NexGen carried out a ground radiometric and boulder prospecting program in order to investigate many of the radiometric anomalies identified by the 2013 airborne survey (refer to Section 9.4.2). Radioactivity was measured at 698 stations, mostly on boulders which were chiefly Athabasca Group sandstones. Rare boulders of basement affinity were also measured. 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 radioactive boulders were discovered; however, in each case, spectrometer analyses showed the radioactivity to be sourced from thorium. No samples were assayed.

9.3 Geochemical Sampling

Due to the significant glacial-derived cover, geochemical sampling is not considered to be a primary exploration tool.

In 2015, radon-in-water surveys were conducted by RadonEx Exploration Management Ltd. over parts of Patterson, Beet, and Naomi Lakes (Charlton, 2015). The surveys consisted of the collection of 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 generally, but not always, spaced 200 m apart. The results showed multiple areas with anomalous radon gas concentrations.

9.4 Geophysics

9.4.1 Ground Geophysical Surveys

Gravity

NexGen completed a ground gravity survey over much of the western half of the Project area (Koch, 2015; Koch 2013). The surveys were completed by Discovery Geophysics International Inc. (Discovery) and MWH Geo-Surveys Ltd. (MWH) from the fall of 2013 to the winter of 2015. In total, 12,867 gravity measurements were acquired within the survey areas, including a number of duplicate measurements acquired in areas surveyed by Mega before the Project was acquired by NexGen. Stations were spaced 50 m apart along lines spaced at 200 m and were located by differential global positioning system (GPS) instrument. Features identified from the survey results are interpreted to be larger regional trends upon which smaller, more localized features occur. These smaller features, showing both relatively high and low gravity responses, can 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 1 km diameter. The gravity low present at the Arrow deposit may be the result of clay alteration (illite/dravite/sudoite) of the basement rocks within and adjacent to the deposit.

DC Resistivity

In 2013, NexGen completed a DC resistivity survey over a small area on the westernmost portion of the Project area (Koch, 2013b). This survey was completed by Discovery on 200 m spaced grid lines via pole-dipole array with stations spaced at 50 m along lines. Estimated depth penetration based on the array parameters used was approximately 225 m. The survey successfully identified several prospective basement-hosted electromagnetic (EM) anomalies. It also identified a near-surface, flat-lying conductive horizon interpreted to be carbonaceous Manville Group rocks overlying the basement.

3D DC Resistivity

Dias Geophysical completed two 3D DCIP resistivity surveys on behalf of NexGen. The first survey, in 2016 (Rudd and Lepitzki, 2017) was situated over the Arrow deposit located within claim block S-113927. The initial survey consisted of a 1.44 x .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 to the southwest of the Arrow deposit along a parallel conductor. This second anomaly was tested in early 2017 (Rudd and Thibaud, 2017), and resulted in the discovery of the South Arrow occurrence.

An expanded 3D resistivity survey was subsequently completed over claim S-113927 and added 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, DGPS coordinates were determined for each station. It was determined that the 3D resistivity completed over the Rook I Project 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 Airborne Surveys was contracted by NexGen to fly a high resolution radiometric magnetic gradiometer–VLF EM survey over the entire Rook I Project. The survey included 3,491 line-km flown on lines spaced 200 m apart (Goldak, 2013). VLF data acquired as part of the survey have confirmed the widespread presence of basement structures on the Project. Magnetic data acquired suggest highly variable geology within the Project and a complex geological history. Radiometric data acquired show a number of surficial radiometric anomalies.

VTEM

In 2014, Aeroquest Airborne was contracted by NexGen to fly a VTEM survey over a portion of the Rook I Project (Pendrigh and Witherly, 2015). The survey included 793 line-km on lines spaced 100 m apart. Magnetic data were also collected in tandem with EM data. The results showed a number of northeast-trending EM conductors, most of which remain untested by drilling. 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.

Figure 9-1 is an image of the VTEM survey showing the interpreted conductors.

ZTEM

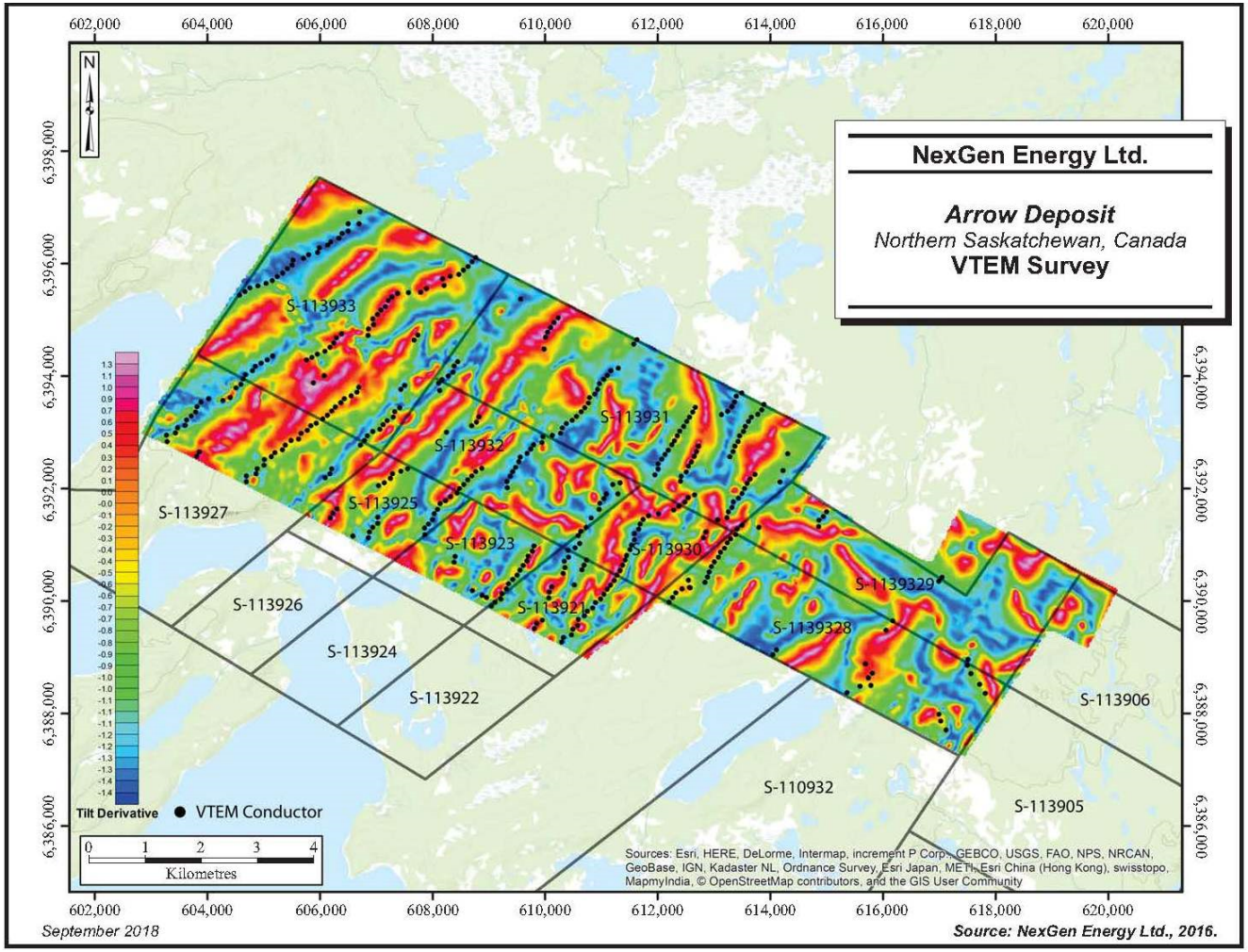
In 2016, Geotech Ltd. was contracted by NexGen to carry out a ZTEM survey over a portion of the Project (Pendrigh and Witherly, 2017). The survey was flown parallel to the Patterson conductive corridor and included 584 line-km on lines spaced 100 m apart. Due to the position of the area of interest along the corridor, a non-standard flight orientation parallel to the primary geological strike was chosen. While this is normally not advised for active-source technologies such as VTEM, with ZTEM, the fact that the two orthogonal components are recorded allows for effective mapping of fields along both survey lines and tie lines. The results showed that a broad corridor of low resistivity traverses the Project from southwest to northeast (Figure 9-2). The Arrow deposit occurs within this corridor. The corridor remains largely undrilled and represents a significant exploration opportunity.

Gravity

In 2016, CGG Canada Services Ltd. was contracted to acquire HeliFalcon gravity data along the Patterson conductive trend (Pendrigh and Witherly, 2017). The survey included 255 line-km on lines spaced 200 m apart and oriented in a northeast–southwest direction. Similar to the ground gravity survey, features identified from the survey results are interpreted to be larger regional trends upon which smaller, more localized features occur (Note: Figure courtesy NexGen, 2017.

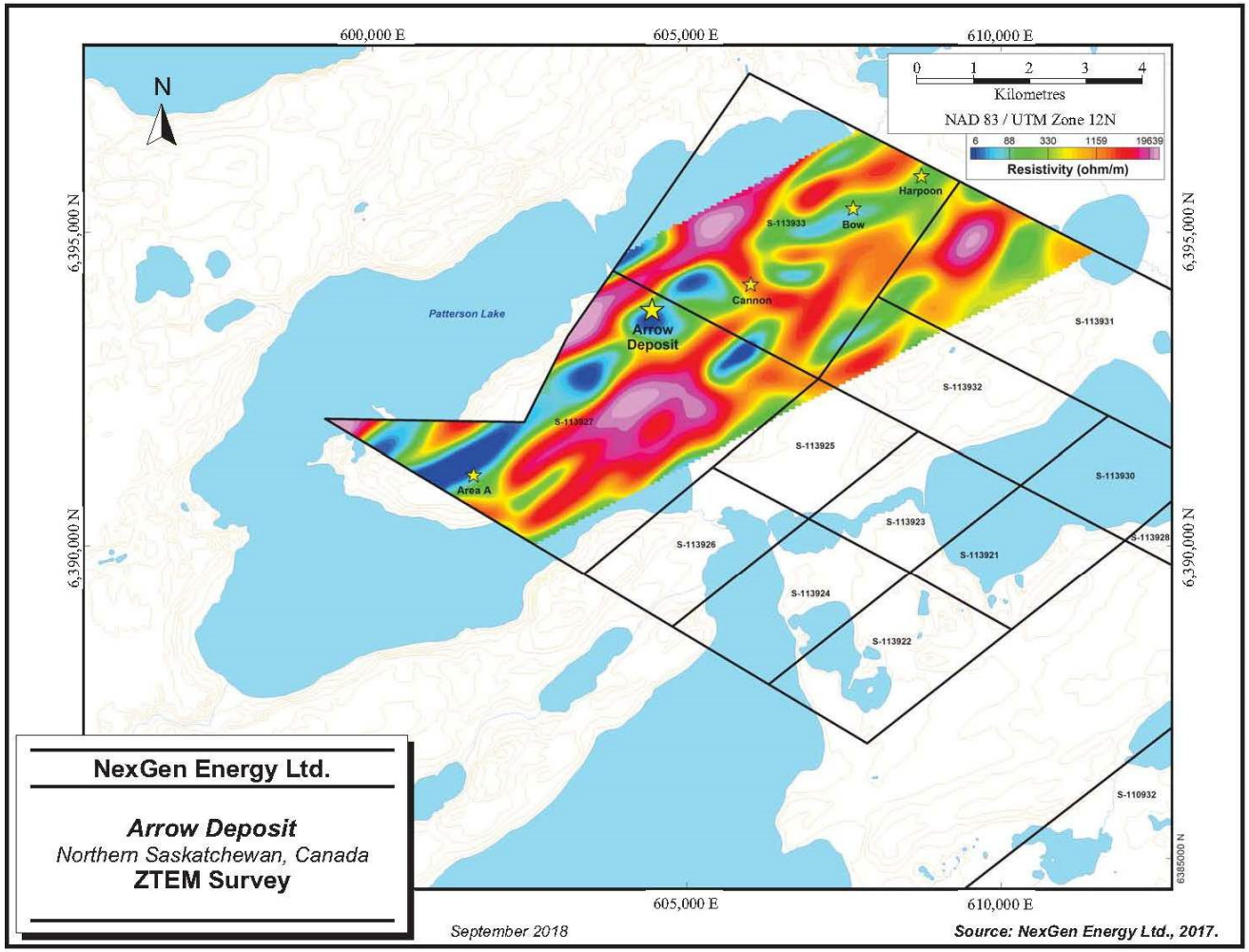
Figure 9-3). These smaller features, showing both relatively high and low gravity responses, can be the result of hydrothermal alteration in both sandstones and basement rocks. The 2016 airborne survey positively identified the gravity anomaly associated with the Arrow deposit and correlated very well with the ground gravity survey previously completed by NexGen. This indicates that airborne gravity can be an effective regional exploration tool in the search for basement-hosted uranium mineralization in the Athabasca Basin.

Figure 9-1: 2014 VTEM Survey



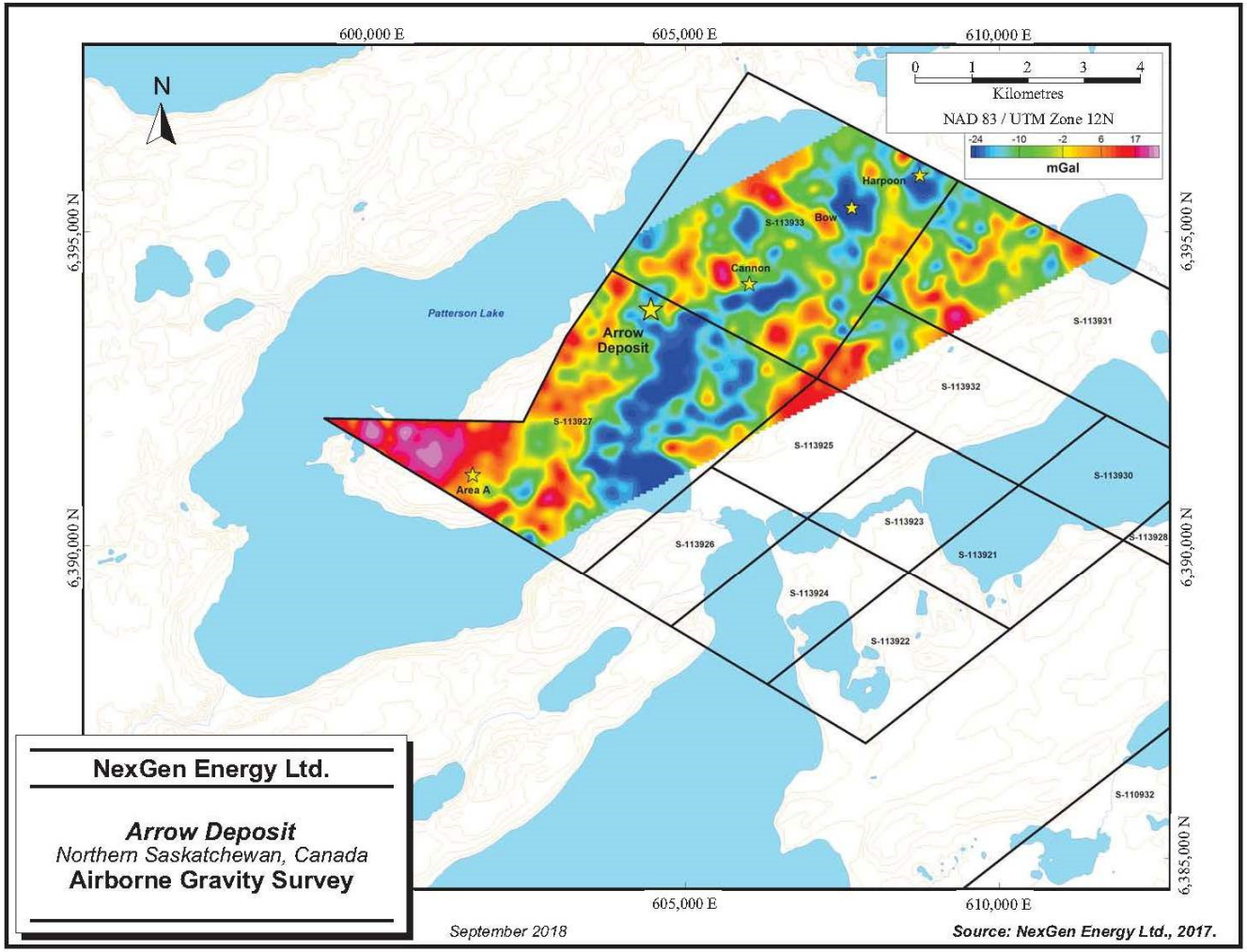
Note: Figure courtesy NexGen, 2016.

Figure 9-2: 2016 ZTEM Survey



Note: Figure courtesy NexGen, 2017.

Figure 9-3: Airborne Gravity Survey



Note: Figure courtesy NexGen, 2017.

9.5 Exploration Potential

A number of uranium-anomalous occurrences have been identified within the Project area (refer to Section 7.4). Geophysical surveys indicate a number of geophysical anomalies that may warrant further exploration.

In mid-2018, step-out drilling completed outside the current Mineral Resource envelope identified a new uranium-mineralized shear zone, the A0 shear, and intersected uranium mineralization in two areas to the northeast and northwest of the Arrow deposit.

Drilling that focused on an under-explored area to the northeast boundary of the currently defined A2 high-grade domain at variable elevations was completed in November, 2018. Mineralization between the A2 and A3 shears was identified, and the drill program was considered to demonstrate the continuity of high-grade mineralization beyond the currently-defined A2 high-grade domains.

9.6 Comments on Section 9

Exploration to date has identified the Arrow deposit and a number of regional targets. The Project area retains significant exploration potential.

10.0 DRILLING

10.1 Introduction

As of the effective date of this report, NexGen and its predecessors have completed 593 holes totalling 302,021 m. From 2013 to the effective date of this report, NexGen has completed 555 holes totalling 296,681 m of drilling. Drilling completed in the Project area is summarized in Table 10-1. Locations of drill collars for the 2013–2017 NexGen programs is included as Figure 10-1. Diamond drilling on the Rook I 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 were essentially the same for all drill programs (refer also to discussion in Section 11). Deposit dimensions are provided in Section 7.

10.2 Drill Methods

All NexGen drilling has used core methods. Two contractors have been used, Guardian Drilling Corp (2013) and Aggressive Drilling Ltd. (2014–2017). Core has been drilled predominantly in NQ diameter except for geotechnical holes which utilized HQ diameter, and AQ when directional drilling technology was being used.

Directional core drilling technology was used from 2015–2018, which allows for precise controlled deviation of drill holes and multiple branches drilled from one main pilot hole. The drilling method allows for both precise pierce point control (within three metres) and saves significant drilling metres. Directional drilling was completed by Tech Directional Services Ltd.

All holes within the Project area are cemented from the bottom of the hole to approximately 5 m above the top of the Devonian sandstones, if present, or to the top of bedrock if not present.

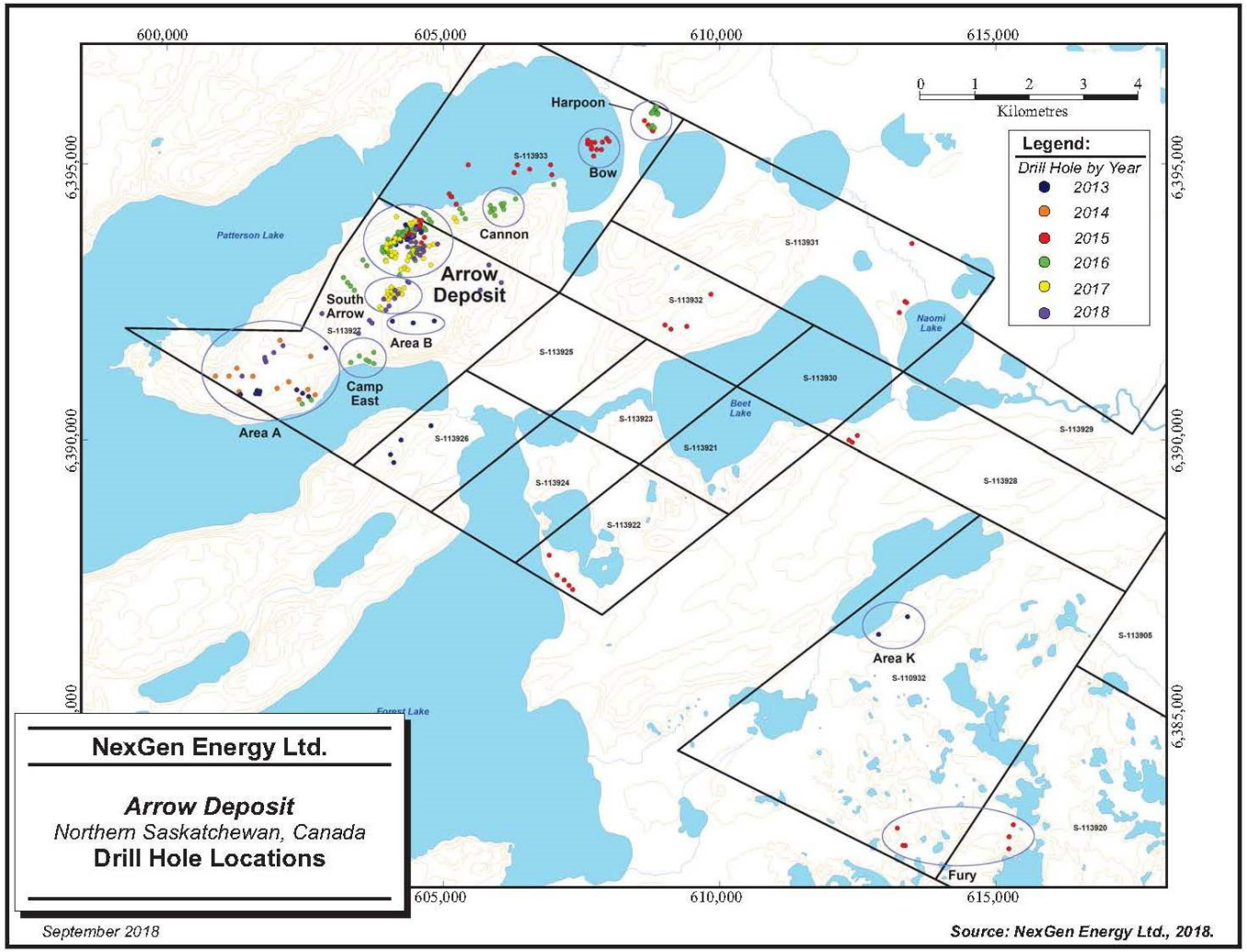
Table 10-1: Drill Table

Year	Season	Target Area	Company	Contractor	No. of Holes	Metres Drilled
1977	Winter	SW-2	Kerr Addison Mines - SMDC JV	Bradley Bros.	1	124
1978	Winter	SW-2	Canadian Occidental Petroleum Ltd.	Canadian Longyear	2	290
			Hudson Bay Exploration and Development Co. Ltd.	Midwest Drilling	1	297
1978 subtotal					**3**	**587**
1979	Winter	SW-2	Canadian Occidental Petroleum Ltd.	Canadian Longyear	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 subtotal					**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 subtotal					**10**	**1,319**
2013	Fall	Area A	NexGen Energy Ltd.	Guardian Drilling	13	3,029
2014	Winter	Arrow	NexGen Energy Ltd.	Aggressive Drilling	8	4,642
		Dagger	NexGen Energy Ltd.	Aggressive Drilling	3	963
		Area A	NexGen Energy Ltd.	Aggressive Drilling	5	1,591
	Summer	Arrow	NexGen Energy Ltd.	Aggressive Drilling	26	16,094
		Area B	NexGen Energy Ltd.	Aggressive Drilling	4	1,182
		Dagger	NexGen Energy Ltd.	Aggressive Drilling	1	413
		K	NexGen Energy Ltd.	Aggressive Drilling	2	558
		Area A	NexGen Energy Ltd.	Aggressive Drilling	3	885
2014 subtotal					**52**	**26,328**
2015	Winter	Arrow	NexGen Energy Ltd.	Aggressive Drilling	24	12,550

Year	Season	Target Area	Company	Contractor	No. of Holes	Metres Drilled
		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 subtotal					**115**	**54,574**
2016	Winter/Spring	Arrow	NexGen Energy Ltd.	Aggressive Drilling	69	35,494
		Arrow Trend	NexGen Energy Ltd.	Aggressive Drilling	2	1,746
		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 subtotal					**175**	**96,993**
2017	Winter	Arrow	NexGen Energy Ltd.	Aggressive Drilling	56	34,271
		Arrow Trend	NexGen Energy Ltd.	Aggressive Drilling	1	994
		South Arrow	NexGen Energy Ltd.	Aggressive Drilling	2	1,792

Year	Season	Target Area	Company	Contractor	No. of Holes	Metres Drilled
		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 subtotal					**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
2018 subtotal					**54**	**30,208**
Total all drilling					***593***	***302,021***
NexGen total drilling					***555***	***296,681***

Figure 10-1: Drill Collar Location Plan



Note: Figure courtesy NexGen, 2018.

10.3 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 with standard 1.5 m length (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.

Each core box was initially surveyed with a Radiation Solutions RS-120 scintillometer to determine if any boxes contained mineralization. A threshold of 500 counts per second (cps) was used for Arrow core, and 300 cps for core from elsewhere in the Project area. All mineralized core boxes above the threshold, plus a box before and after, was taken to the "hot" shacks for logging and sampling. All other core was moved to be processed in the "cold" logging shacks.

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

Logging and sampling information was entered into a Microsoft Access database template on a laptop computer which was integrated into the Project master digital database on a daily basis.

10.4 Recovery

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

10.5 Collar Surveys

The collar locations of drill holes were spotted and surveyed by differential base station global positioning system (GPS) using the UTM Zone 12N NAD83 reference datum. The drill holes have a concise naming convention with the prefix "AR" denoting "Arrow" or "RK" denoting "Rook I" followed by two digits representing the year and the number of the drill hole.

In general, most of the drilling was completed in both northwest and southeast directions with drill holes spaced approximately 15 m to 50 m apart based on directional drilling orientation.

10.6 Downhole Surveys

The trajectory of all drill holes was determined during drilling with a Reflex instrument in single point mode, which measured the dip and azimuth at 30 m intervals. Both immediately below casing and after completion, all holes at the Arrow deposit were surveyed via Stockholm Precision Tools north-seeking gyro, which measured the dip and azimuth continuously down hole.

10.7 Sample Length/True Thickness

The mineralization in the Arrow deposit is sub-vertical and true width is estimated to be from 30% to 50% of reported core lengths based on currently-available information.

10.8 Comments on Section 10

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.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Methods

Three types of samples are collected for geochemical analysis:

- Point samples taken at nominal spacing of 5 m and meant to be representative of the interval or of a particular rock unit
- Composite samples in Athabasca sandstone where 1 cm long pieces are taken at the end of each core box row over 10 m intervals (five to seven pieces normally for a sample)
- 0.5 m and 1.0 m samples taken over intervals of elevated radioactivity and 1–2 m beyond the radioactivity.

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

The bags containing the split samples were placed in lidded buckets for transport by NexGen personnel to Saskatchewan Research Council Geoanalytical Laboratories (SRC) in Saskatoon, Saskatchewan.

11.2 Density Determinations

NexGen personnel performed bulk density measurements on full core on site using standard laboratory techniques. In mineralized zones, bulk density was measured from samples at 2.5 m intervals, where possible (i.e., approximately 20% of all mineralized samples). Pieces of core were sealed in cellophane wrap and then weighed in air and weighed submerged in water. Bulk density was then calculated from the resulting data. In order for density to be correlated with uranium grades across the data set, each density sample directly correlated with a sample sent to SRC for assay (i.e., downhole intervals are the same for density samples and assay samples).

A total of 5,083 (Arrow: 4,991, South Arrow: 92) bulk density measurements were collected on drill core samples from the main mineralized zones to represent local major

lithologic units, mineralization styles, and alteration types. Samples were collected on full core which had been retained in the core box prior to splitting for sampling.

NexGen carried out correlation analyses of the bulk density values against uranium grades which indicate that a strong relationship exists between density and uranium grade (%U_3O_8). The relationship at Arrow can be represented by the following polynomial formula which is based on a regression fit:

$$y = 0.0002x^2 + 0.0181x + 2.4686$$

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.

Figure 11-1 is a logarithmic plot showing density versus grade for Arrow.

11.3 Analytical and Test 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. No umpire laboratories have been used on the project to date.

11.4 Sample Preparation and Analysis

11.4.1 SRC Sample Preparation

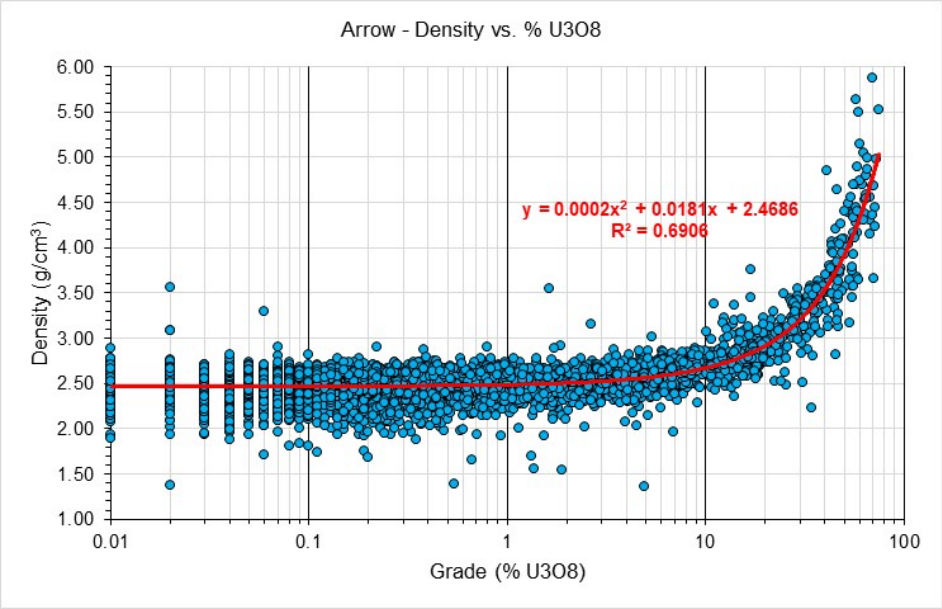
SRC crushed each sample to 60% passing -10 mesh and then riffle split the sample to 200 g with the remainder retained as coarse reject. The 200 g sample was milled to 90% passing -140 mesh.

11.4.2 Geochemical Analyses and Assay

All samples were analyzed at SRC 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 were analyzed using ICP-MS. Samples with anomalous radioactivity were analyzed using ICP-OES.

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

Figure 11-1: Logarithmic Plot of Bulk Density Versus Uranium Grade, Arrow Deposit



Note: Figure prepared by RPA, 2018.

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

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

11.4.3 PIMA Analyses

Samples were regularly collected in areas of clay alteration for clay mineral identification using infrared spectroscopy. Samples were typically collected 5 m intervals and consisted of centimetre-sized pieces of core selected by a geologist.

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

11.5 Quality Assurance and Quality Control

NexGen's QA/QC program includes:

- Standard reference materials (SRM): Determination of accuracy
- Duplicate samples: Determination of precision/repeatability
- Blank samples: Screen cross-contamination between samples during preparation and analyses.

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

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

11.5.1 Standard Reference Materials

SRMs were obtained from the Canadian Centre for Mineral and Energy Technology (CANMET). The individual SRM inserted into the sample stream was selected based on the core scintillometer measurements.

SRMs were considered to fail when more than three standard deviations (±3SD) from the mean of the measured values for each type of material were returned. Results within ± two standard deviations (±2SD) were considered acceptable.

A small but negative bias for SRMs BL5 and BL2A was noted for the period 2013–2018, suggesting that the SRC declared value of SRM BL4a was 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 the RPA considered that the BL4a reference material is extremely homogeneous with repeatable results.

On average, less than 1% of SRM samples were outside the precision limits. One sample from BL-4a returned values in excess of 3SD from the respective mean; however, because that sample plotted only just above the 3SD threshold, NexGen passed the respective batch.

Table 11-1: QA/QC Protocols

QA/QC Type	Insertion Frequency	Acceptance Criteria
Blank	1 in 40	Assay > 10% detection limit
Field duplicate	1 in 50	Relative difference ≤ ±20%
SRM	1 in 54	95% of samples ≤ ±2 std. dev ≤ 1% of samples ≥ ±3 std. dev
Check assay	100 per month sent to two external laboratories	Relative difference ≤ ±10% Std. dev ≤ 15% Difference between means ≤ 5% $R^2 \geq 0.9$

RPA considers that there is 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 2018 support the use of samples assayed at the SRC laboratory during this period in Mineral Resource estimation.

11.5.2 Blanks

Blank material samples consisted of pieces of rose quartz obtained from Deptuck's Landscaping & Supplies of Saskatoon, Saskatchewan.

Blanks were considered to have failed when results were >10 times the lower detection limit. Two samples failure occurred. However, as all other QA/QC samples from those sample batches passed, NexGen chose not to take corrective steps and the batches were passed.

11.5.3 Duplicate Samples

Field duplicates, pulp duplicates, and crush duplicates were submitted to SRC. The results were as expected with acceptable repeatability.

SRC also completed laboratory duplicate analysis on one 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 were completed on half cores of entire samples via wax methods. The results were as expected with acceptable repeatability.

11.5.4 SRC Internal QA/QC Program

Quality control was maintained for all analytical apparatus at SRC with certified reference materials 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 were regularly analysed in duplicate. All quality control results had to be within specified limits otherwise corrective action was taken. If there was a failure in a QA/QC analysis, the entire batch was reanalysed.

All processes performed at the laboratory were subject to a strict audit program, which was performed by approved trained professionals.

11.6 Databases

NexGen maintains a complete set of drill hole data plus other exploration data for the entire Property in a Microsoft Access database. RPA was supplied with an individual drill hole database for the Arrow Deposit on the Property by NexGen. The Arrow resource database dated May 25th, 2018 includes drill hole collar locations (including dip and azimuth), assay, and lithology data from 406 drill holes totalling 240,334 m of drilling. Of the 406 holes completed, 14 drill holes were abandoned before reaching their target depth, are considered restarts, and were not used in the resource estimate. The wireframe models representing the mineralized zones are intersected in 317 of 392 drill holes.

11.7 Sample Security

As each hole was being drilled, drilling contractor personnel placed the core in wooden boxes at the drill site and sealed the core boxes with screwed-on wooden lids. Core was then delivered to the Rook I core processing facility by the contractor twice daily. Only the contractor and NexGen geological staff were authorized to be at drill sites and in the core processing facility. After logging, sampling and shipment preparation, samples were 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 were sent securely via electronic transmission to NexGen. These results were provided as a series of PDF files and an Excel spreadsheet.

11.8 Comments on Section 11

Results of the QA/QC program were well documented by NexGen. RPA has relied on documentation provided by NexGen in addition to audit of the QA/QC data. RPA considers the QA/QC protocols in place for 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.

12.0 DATA VERIFICATION

12.1.1 Site Visit and Core Review

Mr. Mark Mathisen, CPG, visited the Project from January 19 to 20, 2016 and January 22 to 25, 2017 during the winter drill programs in connection with the Arrow Mineral Resource estimate. RPA visited several active drill sites and targets. RPA reviewed core handling, logging, sample preparation and analytical protocols, density measurement system, and storage procedures. RPA examined core from drill holes AR-14-30, AR-15-57c3, AR-15-62, AR-16-98c1, AR-16-106c1, and AR-16-111c1 and compared observations with assay results and descriptive log records made by NexGen geologists. As part of the review, RPA verified the occurrences of mineralization visually and by way of a hand-held scintillometer.

As part of the data verification process, RPA also:

- Reviewed the Leapfrog model parameters and geological interpretation
- Reviewed how drill hole collar locations are defined and inspected use of the Devico directional drilling
- Observed data management system and obtained master database
- Obtained SRC laboratory certificates for all drilling assays.

12.1.2 Database Validation

RPA performed the following digital queries:

- Header table: searched for incorrect or duplicate collar coordinates and duplicate hole IDs
- Survey table: searched for duplicate entries, survey points past the specified maximum depth in the collar table, and abnormal dips and azimuths
- Core recovery table: searched for core recoveries greater than 100% or less than 80%, overlapping intervals, missing collar data, negative 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
- The data were exported from a Microsoft Access database and imported into a Vulcan database.
  - The 2018 Vulcan database utilized a similar design as the Microsoft Access Resource database
  - Quality control completed in Access, validation completed in Vulcan and Leapfrog.
  - Implemented a density hierarchy:
    - SRC density values (laboratory results)
    - NexGen density values (field results)
    - Calculated values (polynomial regression).

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

No significant issues were identified.

12.1.3 Independent Verification of Assay Table

The drilling database contains a total of 65,811 assays used for estimating the 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. For 2018, all 3,530 samples added to the database were checked against their batch certificates. For older data, another 5,035 samples were randomly selected and checked against assay certificates. In all, a total of 13% (8,565 samples) of the drilling database were checked with minimal errors found and all are most likely due to rounding. RPA believes this indicates that the integrity of the database is more than sufficient for an accurate resource.

12.2 Comments on Section 12

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 reliable and appropriate to support a Mineral Resource estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

SRC was contracted to do a metallurgical test program using samples from the Arrow deposit.

The metallurgical test program included a bench test program, a pilot plant and paste backfill testing. The metallurgical test program described was developed and performed under Wood's supervision.

13.2 Metallurgical Testwork

13.2.1 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.

Ten additional samples of localized deposit areas were also tested:

- Five individual zones, A1 to A5
- Two extra-high-grade zones,
- One high-grade Mo/U zone.

Thirteen composite samples were prepared, and used for quantitative evaluation of materials by scanning electron microscopy (QEMSCAN), leaching, potential acid generation (PAG) and tailings preparation for paste backfill tests.

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

Five composite samples were subject to SAGDesin and Bond ball mill index testwork.

One large batch leach test was completed on a medium-grade 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 high-, medium- and low-grade 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 high-, medium- and low-grade 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 medium-grade 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-high-grade and one high Mo/U ratio composites' leach discharges using optimized leaching conditions.

Solvent extraction tests included solvent extraction 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 medium-grade 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 high-, medium- and low-grade samples. Separating funnel shakeout tests using standard conditions were performed on the eight settling filtrates of zones A1 to A5, the extra-high-grade and high Mo/U samples.

Using bench optimization tests results a flowchart for treating large batch pregnant leach solution was developed and relevant large bench scale experiments were carried out, including:

- Organic protonation
- Extraction
- Acid scrubbing
- Strong acid stripping
- Gypsum precipitation

- Yellowcake 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 to 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 1st 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 2nd 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 U 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 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 solvent extraction 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 solvent extraction (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 yellowcake precipitation tests.

One gypsum precipitation test (SX20) was carried out with a diluted loaded strip solution, which was obtained by combined the 1st, 2nd, and 3rd loaded strip solutions from SX19 and was used for gypsum precipitation and filtration tests.

Two yellowcake 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 Mo and 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.

13.2.2 Pilot Plant Testing

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

The feed of the first pilot leaching test (medium-grade pilot) was the medium-grade composite sample. The medium-grade composite sample represented mineralization studied in the 2017 PEA. The sample contained 2.03% U_3O_8, 315 ppm Mo and 37 ppm As. The total weight was 409.3 kg. The feed was ground to 100% passing 300 µm using a 1 ft x 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 then the medium-grade 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 x 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 medium-grade sample was used for all of 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
- Solvent extraction
- Gypsum precipitation, settling and leaching
- Yellowcake precipitation and settling
- Effluent treatment and settling.

13.2.3 Testwork 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 medium grade composite sample. There were no other gold grains in any other samples.

Grinding

Grinding test results indicate a medium hardness deposit. Average 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 hr residence time. In bench tests, a few samples benefited from longer residence times between 8–12 hrs.

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 medium-grade 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 high-grade samples had an extraction of 93.5% (the other six tests all had extractions of over 99.5%) and one of the seven medium-grade samples had an extraction of 96.5% (the other six tests all had extractions over 98.4%).

Arsenic Scrubbing

Arsenic wash efficiency was low (27.8%). However, the arsenic concentration in the organic is very low and is only 9 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 yellowcake 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, magnesium and phosphorus). Most of the contamination in the yellowcake 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 heavy media separation was hard to perform on the fine-grained material. No significant gold can be recovered by gravity and no further testwork 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 medium-grade sample resulted in uranium leaching rates at 8 hr 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 hr 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. Fe, Al, Ca, Mg, Na, K, Mn, V, Cu, Zn, Co, Ni, and As) 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% uranium 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 ASTM C967-13 and client specification requirements, yellowcake 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 below were below penalty concentration limits.

- Fe was 0.23 to 0.37%:U, penalty level 0.15%:U, reject level 1.0%:U (ASTM) and 0.5%:U (client)
- Mg was 0.08 to 0.11%:U, penalty level 0.02%:U and reject level 0.50%:U

- SiO2 was 0.71 to 0.85%:U, penalty level 0.50%:U, reject level 2.5%:U (ASTM) and 2.0%:U (client)
- 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 yellowcake sample calcined at 800ºC was acceptable without penalty (<0.01%). Non-calcined yellowcake 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 yellowcake within the accepted product specifications.

Calciner tests were conducted on uranyl peroxided yellowcake produced in the pilot test. Different calcining temperatures were used. X-ray diffraction analysis indicated that most of the uranyl peroxide is U_3O_8 by 600ºC. The initial uranyl peroxide sample was off spec with respect to fluorine. The fluorine assay for yellowcake calcined at 800ºC 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.

Paste Testing

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 LPSD show that 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 unconfined compressive strength (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.

13.3 Recovery Estimates

The average recovery estimate used in the 2018 Pre-feasibility Study was determined from the pilot testing. Pilot leach testing had uranium extractions of 99.3%. The washing efficiency in the counter current decantation was very high at >99.6%. All other unit operations in the pilot testing had uranium recoveries of >99.6%.

Section 17.3 provides additional discussion on recovery estimates within the plant. Uranium recovery was estimated by evaluating the recovery of the individual circuits and combining these into an overall recovery. Total net uranium 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 2018 Pre-feasibility Study 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 Mo-bearing minerals present, however, Mo may occur in chalcopyrite and galena solid solutions. Similarly, there were no As-bearing minerals identified.

13.6 Comments on Section 13

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

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

RPA was supplied with an individual drill hole database for the Arrow deposit in MS Access format, with a close-out date of May 25, 2018. The database included 406 drill holes, of which 14 were not used in estimation as they did not reach target depth. The wireframe models representing the mineralized zones were intersected in 317 of 392 drill holes.

14.2 Geological Models

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

High grade domain models were created using a grade intercepts limit ≥ 1 m with a minimum grade of 5% U_3O_8, although lower grades were incorporated in places to maintain continuity and to maintain a minimum 1 m thickness. Low-grade domain models were created using a lower-grade intercept limit ≥1 m, with a minimum grade-thickness product of 0.1%m, or 2 m at 0.05%.

RPA considers the selection of 0.05% U_3O_8 to be appropriate for construction of 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 underground operating scenario and is consistent with other known deposits in the Athabasca Basin.

Sample intervals with assay results less than the nominated cut-off grade (internal dilution) were included within the mineralized wireframes if the core length was <2 m or allowed for modelling of grade continuity.

In total, 162 wireframes were constructed within the Arrow A1, A2, A3, A4 and A5 shear zones. This included 10 high-grade wireframes were contained within four low-grade enveloping wireframes. Due to a limited number of drill holes, it was not possible to fully differentiate between the A4 and A5 shears; thus, mineralized intercepts in the A5 shear zone were grouped into the A4 shear for Mineral Resource estimation purposes.

Figure 14-1 is a view through the isometric model showing the low-grade and high-grade domains.

14.3 Exploratory Data Analysis

Samples within the mineralized wireframes were examined using histograms and probability plots. Figure 14-2 is a box-and-whisker plot of the U_3O_8 grades in the major domains.

14.4 Grade Capping/Outlier Restrictions

RPA is of the opinion that the influence of high-grade uranium assays must be reduced or controlled, and uses a number of industry best practice methods to achieve this goal, including capping of high-grade values. Assessing the influence of outliers involves a number of statistical analytical methods to determine an appropriate capping value including preparation of frequency histograms, probability plots, decile analyses, and capping curves. Using these methodologies, NexGen geologists examined the selected capping values for each of the 162 mineralized domains in the Arrow deposit.

High-grade outliers were capped at 2%, 3%, 4%, 6%, 8%, 10%, 15%, 25% and 40% U_3O_8 in the domains, resulting in a total of 422 (1.8%) capped assay values (Table 14-1). An example plot set for the A2-HG Domain is provided in Figure 14-3.

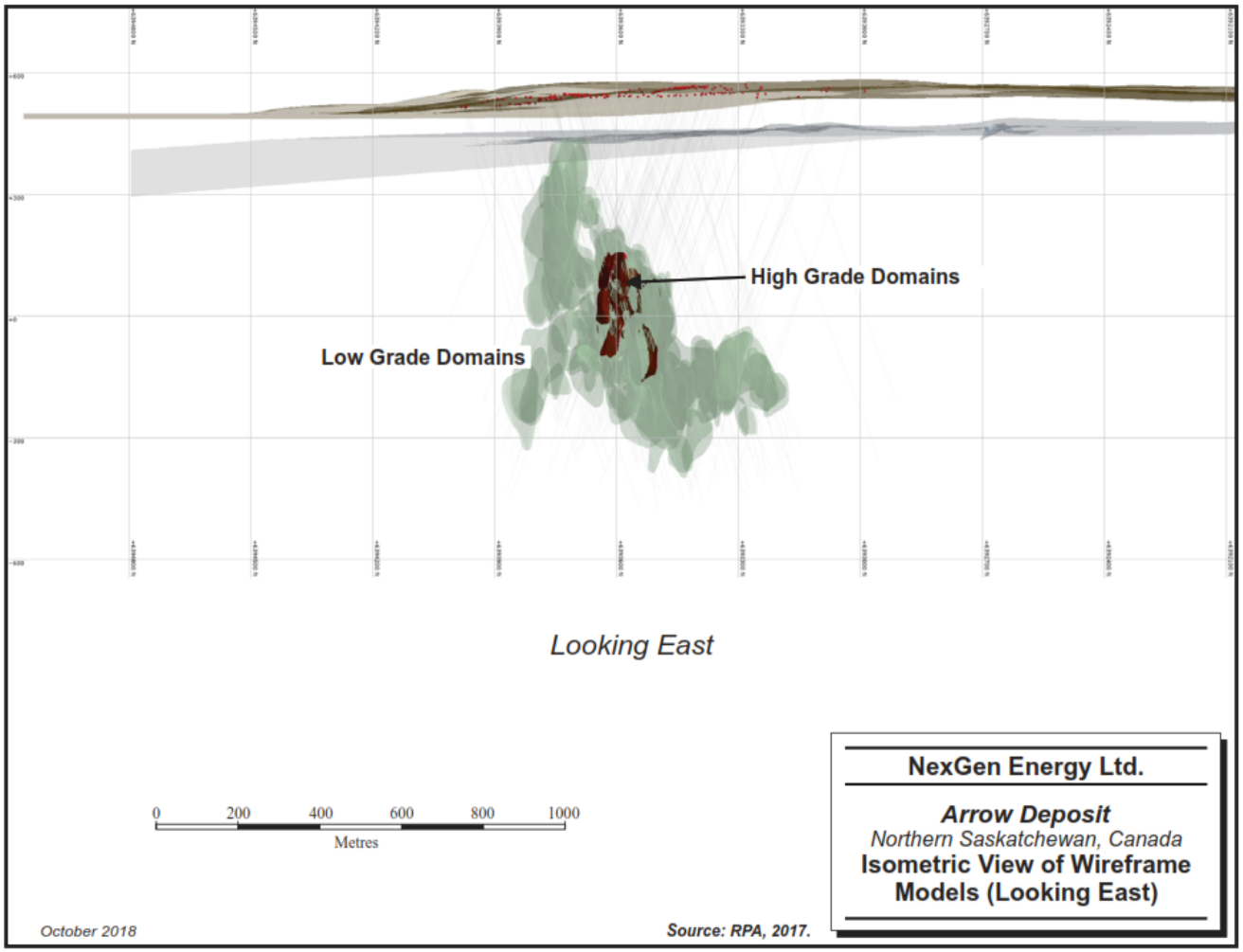
No capping was applied to the high-grade domains with the exceptions of HG-8 and HG-10.

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

14.5 Density Assignment

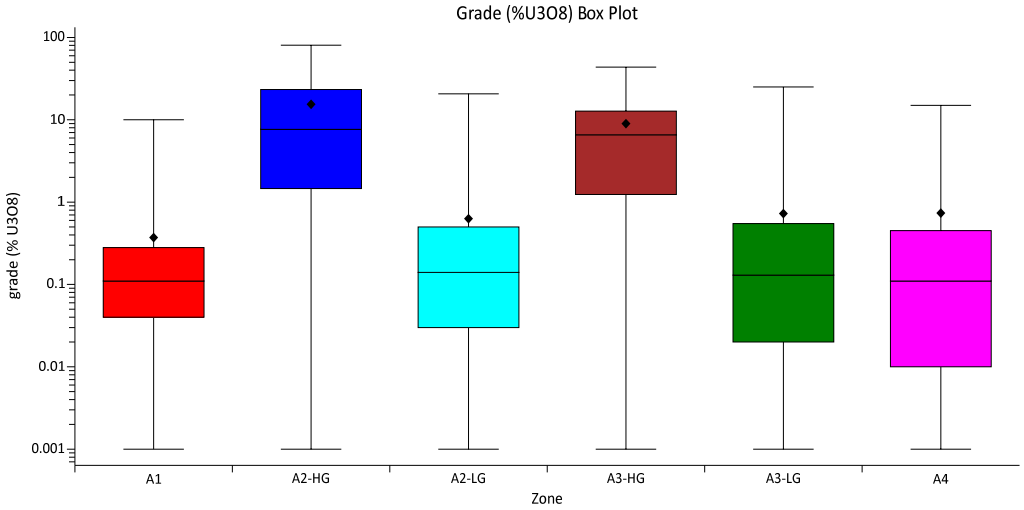
The uranium grade was used to estimate the density of each sample with the polynomial formula provided in Section 11.2. Densities were then interpolated into the block model to convert mineralized volumes to tonnage and were also used to weight the uranium grades interpolated into each block.

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



Note: Figure prepared by RPA, 2017.

Figure 14-2: Box Plots by Domain

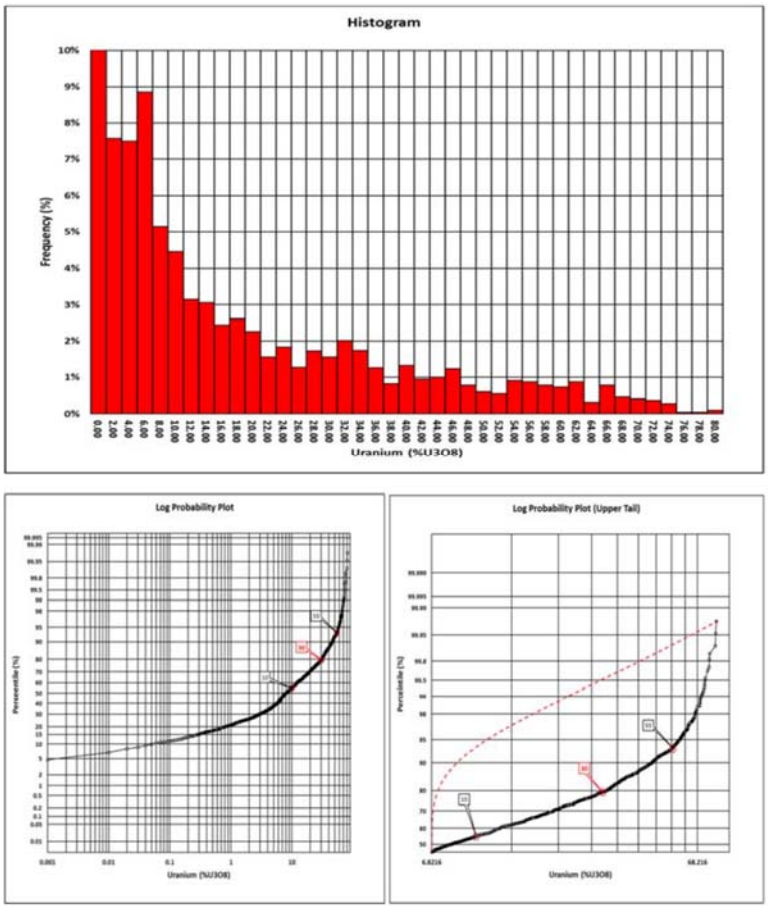


Note: Figure prepared by RPA, 2018.

Table 14-1: 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	3,481	46	1.3
A2-HG	25-HG8	2,052	3	0.1
A2-LG	2, 3, 4, 6, 8, 10 and 15	10,573	197	1.9
A3-HG	40-HG10	290	1	0.3
A3-LG	2, 3, 4, 8, 10 and 15	4,729	86	1.8
A4	2, 3, 6, 8 and 15	2,048	74	3.6
Grand Total		***23,173***	***407***	***1.8***

Figure 14-3: Example Probability and Histogram Plots, A2-HG Domain



Note: Figure prepared by RPA, 2018.

14.6 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 range from 15 cm to 3.0 m within the wireframe models, with 83% of the samples taken at 0.5 m intervals. Given this distribution, and considering the width of the mineralization, NexGen chose to composite to one metre lengths, which RPA agrees is appropriate for Mineral Resource estimation.

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, located at the bottom of the mineralized intercept, were excluded from the composite database.

14.7 Variography

NexGen generated downhole, omni-directional, and directional variograms using the one-metre U_3O_8 composite values located within the A2-HG mineralized domains.

The variograms were used to support search ellipsoid anisotropy, linear trends observed in the data, and Mineral Resource classification decisions. The downhole variograms suggests a relative nugget effect of approximately 10%. 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 20–40 m. Ranges for the high-grade domain also varied from 15–30 m.

RPA recommends additional variography and trend analyses as new drill hole data become available.

To aid in the evaluation of grade continuity, trend analysis, and classification, RPA generated a series of total grade x thickness (GT) contours for selected individual wireframes. A strong correlation was noted with the plunge direction observed in the variography analysis.

14.8 Block Model

Two sub-block models were created by NexGen geologists in Vulcan 10.1. Sub-blocking was used to give a more accurate volume representation of the wireframes using a parent block size of 4 m (along strike) by 4 m (across strike) by 4 m (bench height) and sub-blocks that measured 1 m (along strike) by 1 m (across strike) by 1 m (bench height). A whole block approach was used whereby the block was assigned to the domain where its centroid was located. The models fully enclose the modelled resource wireframes and are oriented with an azimuth of 57°, dip of 0.0°, and a plunge of 0.0° so as to align with the overall strike of the mineralization within the given model area.

14.9 Estimation/Interpolation Methods

For the A2 and A3 HG domains (excluding A2 HG-8), the 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 function, which allowed the search ellipsoid to stay subparallel to the orientation of the mineralized zone trend.

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 162 domains, only A1 low-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. Search ellipses were oriented with the major axis oriented parallel to the dominant northeasterly trend of the domains. The minor axis was oriented horizontally, normal to the major axis (across strike), and the semi-major axis was oriented with a plunge range of 0° to -53° and dip ranging from -76° to -90°.

For the first pass, the variables density (D) and grade multiplied by density (GxD) were interpolated using ordinary kriging (OK) in the A2-HG domains (excluding A2-HG8) and A2-LG domains 206 and 213 (high-grade domain enveloping domains) and using inverse distance weighting to the second power (ID2) on all remaining mineralized domains. Estimates used a minimum of 2–3 to a maximum of 50 composites per block estimate with a maximum of two composites per drill hole. Hard boundaries were used to limit the use of composites between domains. Block grade (GxD_D) was derived from the interpolated GxD value by dividing that value by the interpolated density (D) value for each block.

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

In order to reduce the influence of very high-grade 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 (high yield restriction). The threshold grade levels were chosen from the basic statistics and from visual inspection of the apparent continuity of very high grades within each domain, which indicated the need to limit their influence to approximately half the distance of the main search.

Table 14-2 presents the interpolation strategy by domain.

14.10 Block Model Validation

RPA validated the block model using the following methods:

- Swath plots of composite grades versus OK, inverse distance to the third power (ID3) and nearest-neighbour (NN) grades in the X, Y, and Z directions (Figure 14-4 to Figure 14-6)
- Volumetric comparison of blocks versus wireframes
- Visual inspection of block versus composite grades on plan, vertical and long section
- Parallel secondary estimation using ID3
- Statistical comparison of block grades with assay and composite grades

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

14.10.1 Volume Comparison

Wireframe volumes were compared to block volumes for each zone at the Arrow and South Arrow deposits. This comparison is summarized in Table 14-3 and results show that there is good agreement between the wireframe volumes and block model volume with the difference being less than 1%.

Table 14-2: Block Estimate Search Strategy by Domain

Domain	Estimation Type	Cap %U_3O_8	High Yield Restriction %U_3O_8	Unfold	Bearing	Plunge	Dip	Major	Semi	Minor
1	OK	N/A	N/A	HG01_PROJECTION.tetra	0	0	0	60	30	15
2	OK	N/A	N/A	HG02_PROJECTION.tetra	0	0	0	60	30	15
3	OK	N/A	N/A	HG03_PROJECTION.tetra	100	0	0	60	30	15
5	OK	N/A	N/A	HG05_PROJECTION.tetra	160	0	0	50	60	15
7	OK	N/A	N/A	HG07_PROJECTION.tetra	30	0	0	60	60	15
8	ID^2	25	N/A		225	0	88	100	100	20
9	ID^2	N/A	N/A	HG09_PROJECTION.tetra	0	0	0	100	100	20
10	ID^2	40	N/A	HG10_PROJECTION.tetra	0	0	0	100	100	20
11	ID^2	N/A	15	HG11_PROJECTION.tetra	0	0	0	100	100	20
13	ID^2	N/A	30	HG13_PROJECTION.tetra	0	0	0	100	100	20
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
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

Domain	Estimation Type	Cap %U_3O_8	High Yield Restriction %U_3O_8	Unfold	Bearing	Plunge	Dip	Major	Semi	Minor
113	ID^2	N/A	N/A		238	0	90	100	100	20
114	ID^2	N/A	1		238	0	90	100	100	20
115	OK	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
201	ID^2	4	N/A		238	0	90	100	100	20
202	ID^2	4	N/A		238	0	90	100	100	20
203	ID^2	4	N/A		238	0	90	100	100	20
204	ID^2	2	1		238	0	90	100	100	20
205	ID^2	2	N/A		238	0	90	100	100	20
206	OK	15	N/A		53	20	90	75	55	30
207	ID^2	4	3		238	0	90	100	100	20
208	ID^2	8	2		232	0	82	100	100	20
209	ID^2	2	1		238	0	90	100	100	20
210	ID^2	10	N/A		232	0	86	100	100	20
211	ID^2	10	5		230	0	84	100	100	20

Domain	Estimation Type	Cap %U_3O_8	High Yield Restriction %U_3O_8	Unfold	Bearing	Plunge	Dip	Major	Semi	Minor
212	ID^2	2	1		238	0	90	100	100	20
213	OK	10	N/A		53	0	90	75	50	25
214	ID^2	3	N/A		238	0	90	100	100	20
215	ID^2	2	1.5		238	0	90	100	100	20
216	ID^2	N/A	2		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	2	N/A		238	0	87	100	100	20
221	ID^2	2	1		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	6	4		221	0	84	100	100	20
226	ID^2	2	0.5		239	0	85	100	100	20
227	ID^2	2	1		234	0	81	100	100	20
228	ID^2	2	1.5		238	0	90	100	100	20
229	ID^2	2	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	N/A		238	0	90	100	100	20
233	ID^2	N/A	1		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

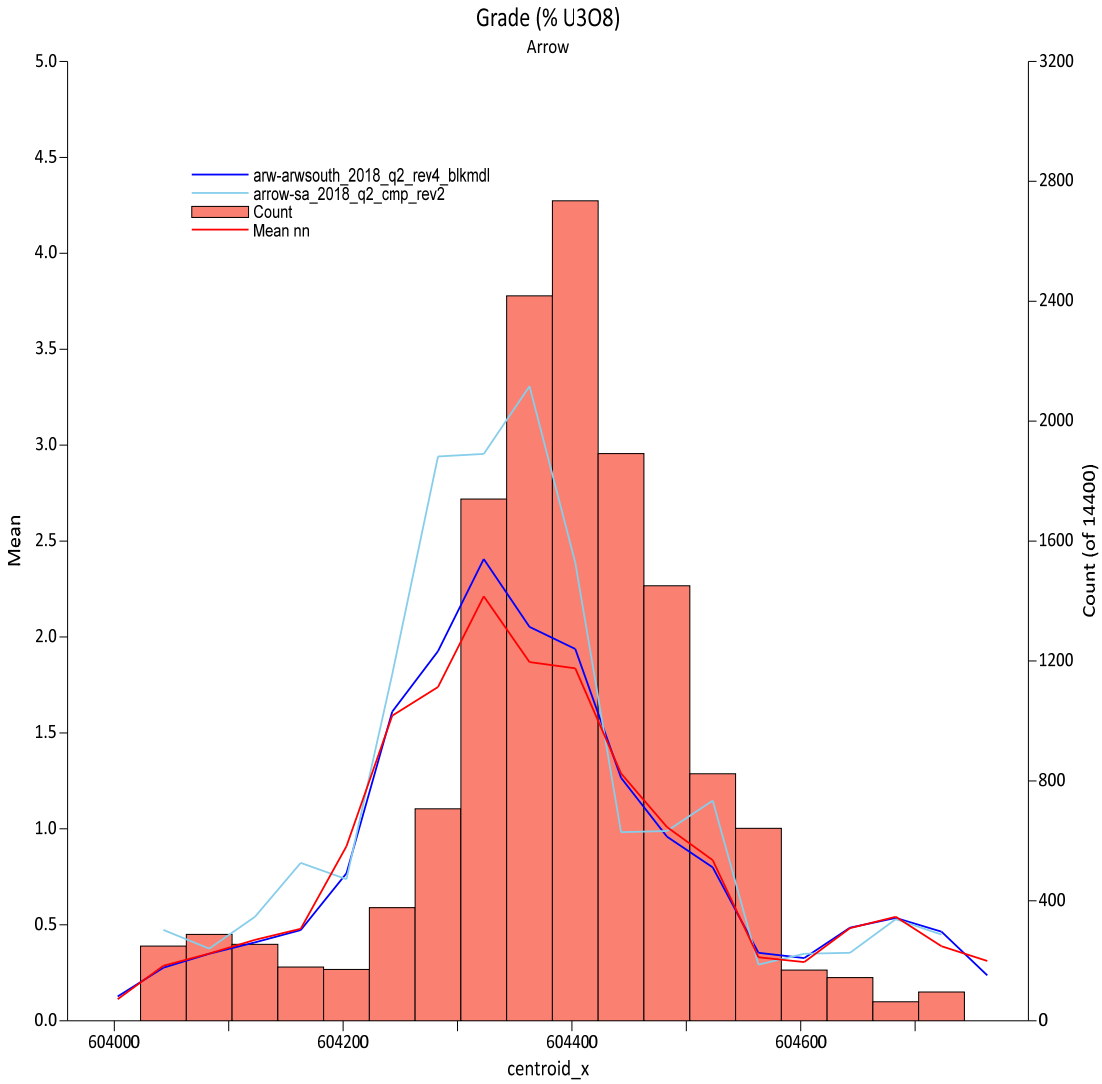
Domain	Estimation Type	Cap %U_3O_8	High Yield Restriction %U_3O_8	Unfold	Bearing	Plunge	Dip	Major	Semi	Minor
236	ID^2	3	N/A		234	0	83	100	100	20
237	ID^2	2	0.5		232	0	90	100	100	20
238	ID^2	N/A	N/A		238	0	-80	100	100	20
239	ID^2	N/A	N/A		238	0	90	100	100	20
240	ID^2	2	0.5		238	0	90	100	100	20
241	ID^2	3	N/A		246	0	88	100	100	20
242	ID^2	6	5		238	0	90	100	100	20
243	ID^2	N/A	N/A		238	0	90	100	100	20
244	ID^2	2	1		230	0	83	100	100	20
245	ID^2	2	N/A		236	0	84	100	100	20
246	ID^2	N/A	N/A		238	0	90	100	100	20
247	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	6	5		238	0	90	100	100	20
250	ID^2	2	N/A		230	0	90	100	100	20
251	ID^2	N/A	N/A		238	0	90	100	100	20
252	ID^2	N/A	N/A		238	0	90	100	100	20
253	ID^2	2	1		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

Domain	Estimation Type	Cap %U_3O_8	High Yield Restriction %U_3O_8	Unfold	Bearing	Plunge	Dip	Major	Semi	Minor
301	ID^2	10	6		238	0	90	100	100	25
302	ID^2	4	2		242	0	88	100	100	20
303	ID^2	N/A	1		238	0	90	100	100	20
304	ID^2	2	1		238	0	88	100	100	20
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	3	N/A		238	0	90	100	100	20
308	ID^2	N/A	1		231	0	90	100	100	20
309	ID^2	25	15		241	0	86	100	100	20
310	ID^2	N/A	2		236	0	87	100	100	20
311	ID^2	2	1		227	0	88	100	100	20
312	ID^2	10	8		238	0	90	100	100	20
313	ID^2	15	8		227	0	85	100	100	20
314	ID^2	2	N/A		238	0	90	100	100	20
315	ID^2	N/A	1		231	0	-85	100	100	20
316	ID^2	2	1		238	0	90	100	100	20
317	ID^2	N/A	N/A		231	0	-87	100	100	20
318	ID^2	15	8		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	N/A	1		233	0	90	100	100	20
322	ID^2	8	2		235	0	90	100	100	20
323	ID^2	N/A	2		238	0	90	100	100	20
324	ID^2	N/A	N/A		237	0	90	100	100	20

Domain	Estimation Type	Cap %U_3O_8	High Yield Restriction %U_3O_8	Unfold	Bearing	Plunge	Dip	Major	Semi	Minor
325	ID^2	2	1.5		240	0	87	100	100	20
326	ID^2	8	N/A		229	0	90	100	100	20
327	ID^2	2	1.5		226	0	90	100	100	20
328	ID^2	8	N/A		242	0	85	100	100	20
329	ID^2	2	1		239	0	86	100	100	20
330	ID^2	2	1		229	0	83	100	100	20
331	ID^2	2	1.5		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	2	N/A		222	0	87	100	100	20
335	ID^2	N/A	N/A		238	0	90	100	100	20
401	ID^2	N/A	N/A		238	0	90	100	100	20
402	ID^2	6	N/A		235	0	88	100	100	20
403	ID^2	15	13		232	0	80	100	100	20
404	ID^2	N/A	N/A		242	0	80	100	100	20
405	ID^2	8	2		235	0	90	100	100	20
406	ID^2	N/A	N/A		230	0	90	100	100	20
407	ID^2	6	N/A		233	0	85	100	100	20
408	ID^2	8	3		230	0	85	100	100	20
409	ID^2	3	1		225	0	80	100	100	20
410	ID^2	6	4		227	0	85	100	100	20
411	ID^2	3	N/A		238	0	90	100	100	20
412	ID^2	2	N/A		238	0	90	100	100	20
413	ID^2	2	1		235	0	75	100	100	20

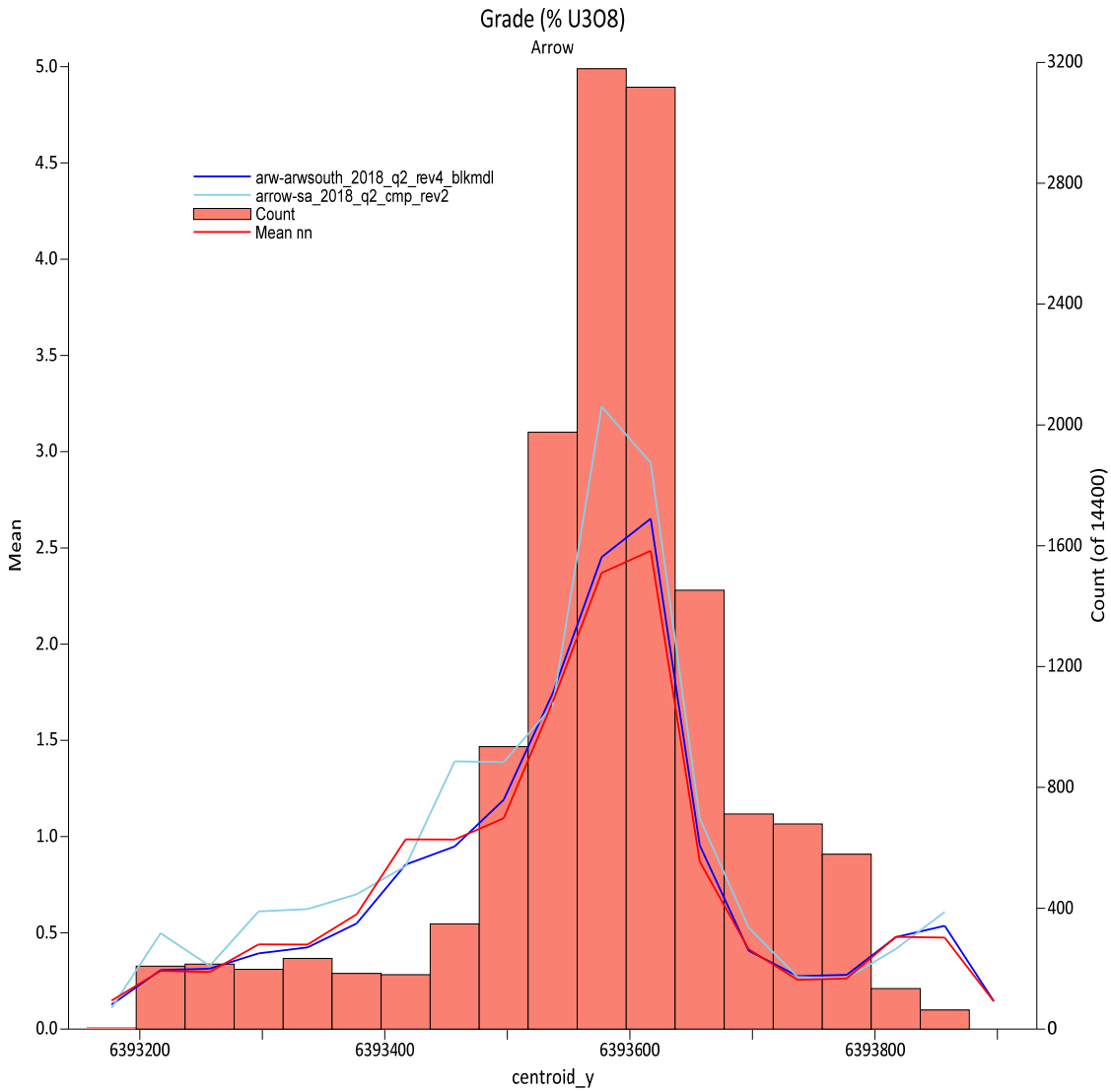
Domain	Estimation Type	Cap %U_3O_8	High Yield Restriction %U_3O_8	Unfold	Bearing	Plunge	Dip	Major	Semi	Minor
414	ID^2	8	2		233	0	85	100	100	20
415	ID^2	3	1		232	0	87	100	100	20
416	ID^2	3	1		232	0	87	100	100	20
417	ID^2	N/A	N/A		239	0	85	100	100	20
418	ID^2	8	6		231	0	85	100	100	20
419	ID^2	6	4		231	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
900	ID^2	2	1		234	0	80	100	100	20

Figure 14-4: East–West (x) Swath Plot of Arrow Deposit



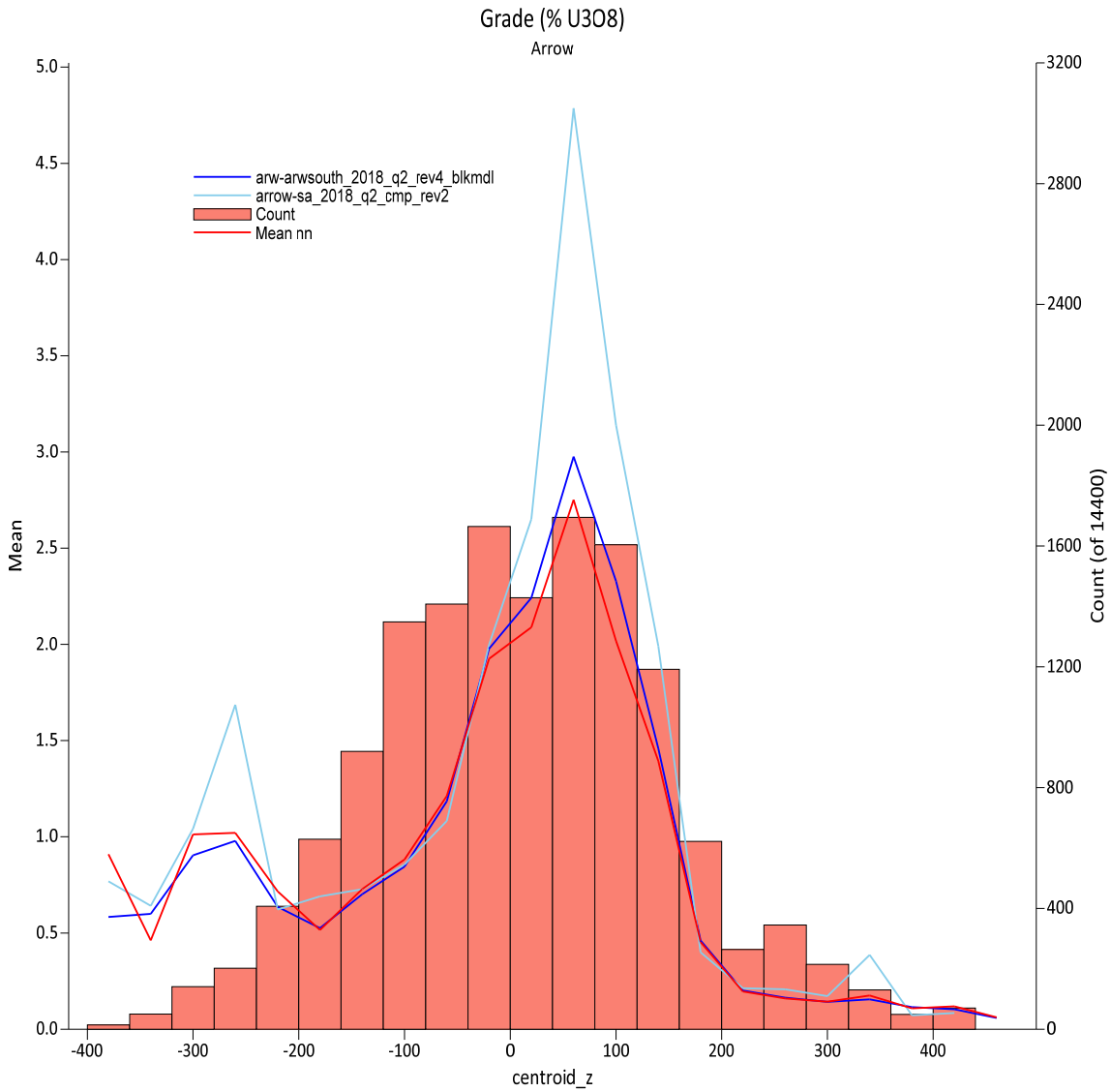
Note: Figure prepared by RPA, 2018.

Figure 14-5: North–South (Y) Swath Plot of Arrow Deposit



Note: Figure prepared by RPA, 2018.

Figure 14-6: Vertical (Z) Swath Plot of Arrow Deposit



Note: Figure prepared by RPA, 2018.

Table 14-3: Volume Comparison

Zone	Wireframe Volume (m^3)	Block Model Volume (m^3)	% Difference
A1	1,493,924	1,493,869	0.00
A2-HG	163,148	163,254	0.07
A2-LG	1,515,437	1,515,316	-0.01
A3-HG	66,157	66,148	-0.01
A3-LG	1,231,930	1,231,628	-0.02
A4	546,212	545,982	-0.04
Total	**5,016,809**	**5,016,197**	**-0.01**

14.10.2 Visual Comparison

Block grades were visually compared with drill hole composites on cross-sections, longitudinal sections, and plan views. The block grades and composite grades correlate very well visually within the Arrow Deposit. Figure 14-7 is a cross section and Figure 14-8 is a level plan showing blocks and drill hole composites colour coded by grade within the A2-HG zone.

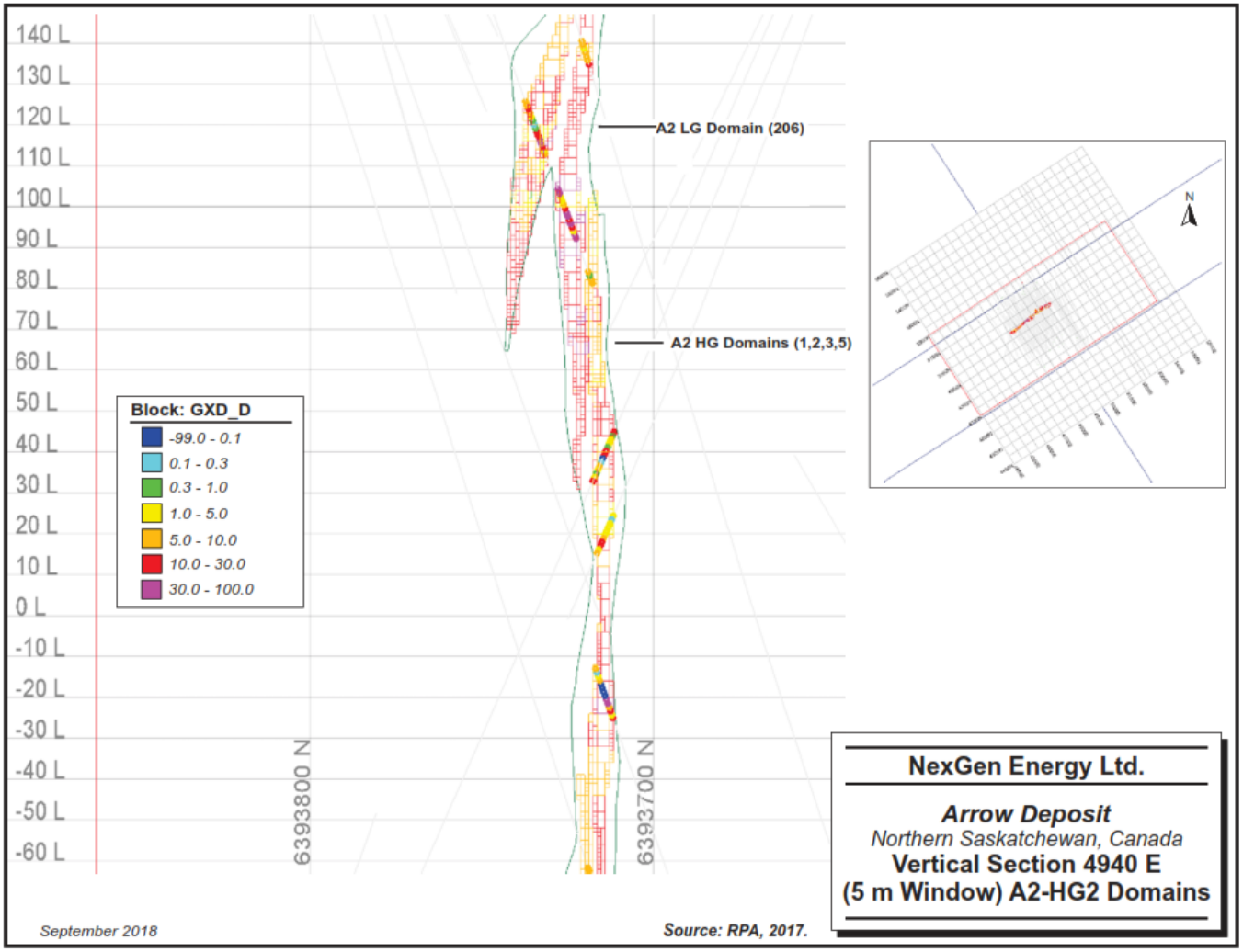
14.11 Reasonable Prospects of Eventual Economic Extraction

To fulfill the requirement to meet "reasonable prospects for eventual economic extraction", RPA estimated a potential underground mining cut-off grade using assumptions based on historical and known operating costs for mines operating in the Athabasca Basin.

Commodity prices used for reserves are based on consensus, long term forecasts from banks, financial institutions, and other sources. The commodity prices used for Mineral Resources are slightly higher than those used for Mineral Reserves.

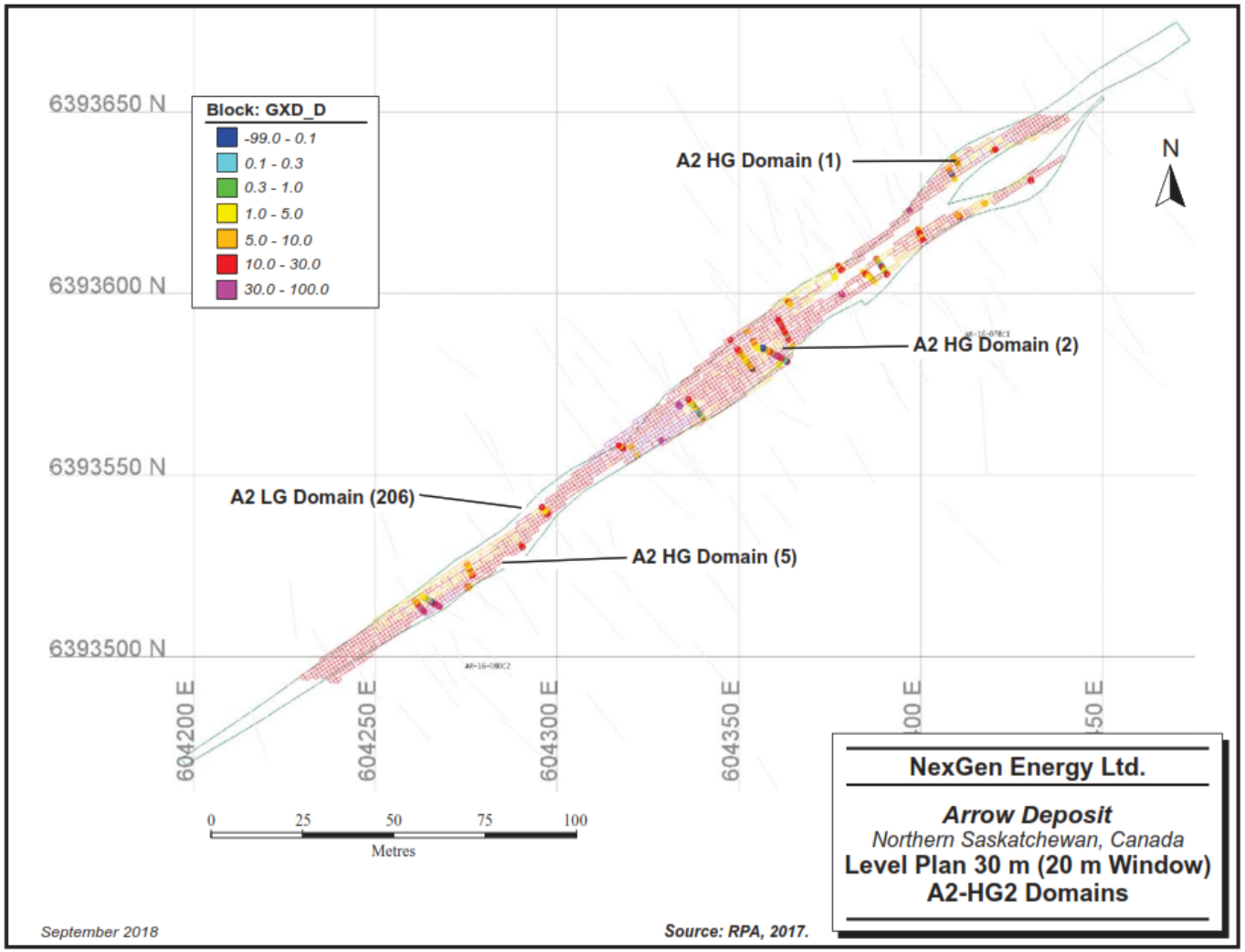
The breakeven cut-off grade was estimated by RPA using a price of US$65/lb U_3O_8 and based on assumptions for process plant recovery, total operating cost, and incremental component of operating cost. The estimated cut-off grade of 0.25% U_3O_8 is in line with the cut-off grade of 0.25% at Cameco's Eagle Point mine, which is basement-hosted mineralization that is interpreted by NexGen to be geologically-similar to the mineralization at Arrow and Arrow South.

Figure 14-7: Vertical Section 4940E (5 m Window) A2-HG Domains



Note: Figure prepared by RPA, 2017.

Figure 14-8: Level Plan 30m (20 m Window Looking Down)



Note: Figure prepared by RPA, 2017.

Table 14-4 shows the input parameters to the cut-off grade estimate. Table 14-5 and Table 14-6 show the sensitivity of the Arrow block model to various cut-off grades. RPA notes that, although there is some sensitivity of average grade and tonnes to cut-off grade, the contained metal is less sensitive.

A minimum mining width of 1 m was assumed using underground mining methods.

The input parameters to the Mineral Resource cut-off are based on results from the PEA completed in 2017.

14.12 Classification of Mineral Resources

Mineral Resources for the Arrow deposit are classified into Indicated and Inferred categories based on studies conducted by Clayton V. Deutsch Consultants Inc. (Deutsch) for NexGen in 2017. The principal aim of this work was to establish a geostatistical simulation workflow for uncertainty as a function of drill hole spacing to support decisions related to future drilling and classification. In Figure 14-9, the red curve represents the probability of the metal content for a nominal quarterly production volume (assuming 1,500 t/d) to be within 15% predicted. This is based on the combined uncertainty of thickness, grade, and density. The areal limit uncertainty was also quantified. The yellow shaded region represents a probability target between 75% and 90% that corresponds to a square drill hole spacing of 32 m and 24 in the plane of vein that would be consistent with an Indicated Resource (Deutsch, 2017).

Deutsch further recommended that additional high-resolution models of uncertainty be evaluated to support mine planning and economic sensitivities.

Indicated and Inferred categories were assigned based on the following parameters:

- Indicated Mineral Resources: Defined by 25 m by 25 m drill hole spacing and a NN distance of ≤30 m with strong geological continuity between drill hole intercepts
- Inferred Mineral Resources: Defined by a drill hole spacing that is >25 m by 25 m and a NN distance of ≤100 m with reasonable continuity assumed between holes.

Table 14-4: Arrow Deposit Cut Off Grade Calculation

Item	Quantity
Price in US$/lb U_3O_8	US$50
Process plant recovery	95%
Mining cost per tonne	US$180
Processing cost per tonne	US$120
G&A cost per tonne	US$32
Total operating cost per tonne	US$332
Break-even cut-off grade	0.25%

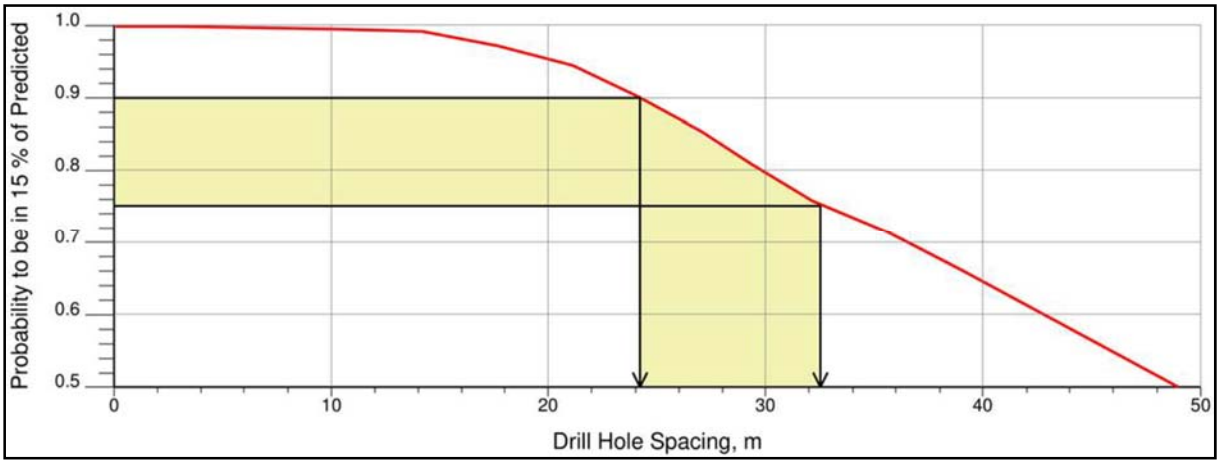
Table 14-5: Indicated Mineral Resource Sensitivity to Cut Off Grade, Arrow Deposit

Cut-off (% U_3O_8)	Tonnes (t x 1,000)	Grade (% U_3O_8)
0.25	2,883	4.04
0.30	2,785	4.17
0.50	2,255	5.06
1.00	1,037	10.15

Table 14-6: Inferred Mineral Resource Sensitivity to Cut Off Grade, Arrow Deposit

Cut-off (% U_3O_8)	Tonnes (t x 1,000)	Grade (% U_3O_8)
0.25	4,844	0.86
0.30	4,185	.095
0.50	2,677	1.27
1.00	1,092	2.09

Figure 14-9: Drill Spacing Analysis



Note: Figure prepared by Deutsch, 2017.

14.13 Mineral Resource Statement

The Mineral Resource estimate is reported using the 2014 CIM Definition Standards.

The effective date of the Mineral Resource estimate is May 25, 2018. Estimated block model grades are based on chemical assays only. The Mineral Resources were estimated by NexGen and audited by RPA. The Qualified Person for the estimate is Mr. Mark Mathisen, CPG, an RPA employee.

Table 14-7 summarizes Mineral Resources based on a US$50/lb uranium price at a cut-off grade of 0.25% U_3O_8.

14.14 Factors That May Affect the Mineral Resource Estimate

Factors that may affect the resource estimate include: 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 reasonable prospects for eventual economic extraction; assumptions as to social, permitting and environmental conditions; and additional infill or step out drilling or results obtained from the planned 2019 drill program.

Table 14-7: Mineral Resource Estimate

Classification	Zone	Tonnage (Tonnes)	Grade (U_3O_8 %)	Contained Metal (U_3O_8 lb)
Indicated	A2-LG	1,244,000	0.79	21,700,000
	A2-HG	459,000	17.88	181,000,000
	A3-LG	1,005,000	0.70	15,500,000
	A3-HG	175,000	9.94	38,400,000
Indicated Total		***2,883,000***	***4.04***	***256,600,000***
Inferred	A1	1,510,000	0.72	23,900,000
	A2-LG	1,293,000	0.70	19,900,000
	A2-HG	5,000	12.78	1,400,000
	A3-LG	1,233,000	1.10	30,000,000
	A3-HG	1,000	8.11	200,000
	A4	801,000	0.92	16,300,000
Inferred Total		***4,844,000***	***0.86***	***91,700,000***

Notes to accompany Mineral Resource table:

1. The estimate effective date is May 25, 2018. The Qualified Person for the estimate is Mr. Mark Mathisen, CPG, an RPA employee.
2. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Mineral Resources are reported at a cut-off grade of 0.25% U3O8. The cut-off grade includes considerations of metal price, process plant recovery, and mining, processing and general and administrative costs.
4. Mineral Resources are estimated using a long-term uranium price of US$50 per lb U_3O_8 and a minimum mining width of 1 m. Tonnes are based on bulk density weighting
5. Totals may not sum due to rounding.

14.15 Comments on Section 14

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 that are not discussed in this Report.

In RPA's opinion, the estimation methodology is consistent with standard industry practice and the Indicated and Inferred Mineral Resource estimates for Arrow and South Arrow are considered to be reasonable and acceptable.

15.0 MINERAL RESERVE ESTIMATES

15.1 Introduction

The Mineral Reserves consist of selected portions of the Indicated Mineral Resources that are above a 0.25% U_3O_8 cut-off grade. 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. It was assumed that both transverse stope and longitudinal retreat stope mining methods would be used.

15.2 Mineral Reserves Statement

The Mineral Reserve estimate is reported using the 2014 CIM Definition Standards.

The effective date of the Mineral Reserve estimate is May 25, 2018. The Qualified Persons for the estimate are Mr. Jason Cox, P.Eng., and Mr. David Robson, P.Eng., M.B.A., employees of RPA.

Table 15-1 summarizes Mineral Reserves based on a US$45/lb uranium price at a cut-off grade of 0.25% U_3O_8.

15.3 Factors that May Affect the Mineral Reserves

Factors that may affect the Mineral Reserve estimate include: 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 can be permitted; assumptions as to social, permitting and environmental conditions; and additional infill or step out drilling or results obtained from the planned 2019 drill program.

15.4 Underground Estimates

15.4.1 Throughput Rate and Supporting Assumptions

The assumed mining rate is 1,100 t/d. Additional information is provided in Section 16.

15.4.2 Stope Sizing

Stope sizing is discussed in Section 16.

Table 15-1: Mineral Reserve Estimate

Category	Tonnage (kt)	Grade (% U_3O_8)	Contained Metal (Mlbs U_3O_8)
Proven	—	—	—
Probable	3,433.0	3.09%	234.1
Total Proven and Probable	***3,433.0***	***3.09%***	***234.1***

Notes to accompany Mineral Reserves table:

1. The estimate effective date is May 25, 2018. The Qualified Persons for the estimate are Mr. Jason Cox, P.Eng., and Mr. David Robson, P.Eng., M.B.A., employees of RPA.
2. Mineral Reserves include transverse and longitudinal stopes, development, and incremental ore.
3. Stopes and ore development were estimated at a cut-off grade of 0.25% U_3O_8. Incremental ore is material between 0.03% U_3O_8 and 0.25% U_3O_8 that must be extracted to access mining areas. A grade of 0.03% U_3O_8 is the regulatory limit for what is considered benign waste and material that must be treated and stockpiled in an engineered facility.
4. Mineral Reserves are estimated using a long-term metal price of US$45/lb U_3O_8, and a 0.75 US$/C$ exchange rate. A minimum mining width of 3.0 m was applied for all longhole stopes. The density varies according to the U_3O_8 grade in the block model. Waste density is 2.464 t/m_3.
5. Totals may not sum due to rounding.

15.4.3 Dilution and Mine Losses

Dilution has been included in the Mineral Reserve estimate through the following:

- For primary transverse stopes, end-wall dilution of 0.5 m is applied to each stope shape, for a total of 1.0 m. Sidewall dilution has not been included for primary stopes, as this would be removing mineralized material from secondary stopes
- For secondary transverse stopes, additional zero-grade fill-dilution of 13% was applied
- For longitudinal stopes, hanging wall and footwall dilution of 0.5 m is applied to each stope shape, for a total of 1.0 m
- For both longitudinal and transverse stopes, the grade of the dilution is based on what was contained in the block model.

Applying the rules above to the longhole stopes resulted in total rock dilution of approximately 31%. Fill dilution, applied to secondary stopes only, represents an additional 13% applied to secondary stope tonnes.

Extraction (mining recovery), used in the 2018 Pre-feasibility Study is 95% for longhole mining and ore development.

Post-blast ore sorting is not considered in the 2018 Pre-feasibility Study, and represents an upside potential area for further head grade control and improvement.

15.4.4 Cut-off Criteria

Based on the Mineral Resource and cut-off grade calculation, stope shapes of >0.25% U_3O_8 were considered in the mine plan (refer to Section 16). After completion of the cash flow model, the initial cut-off grade was validated to ensure that it still aligned with the initial assumptions. The input parameters to the cut-off grade calculation is presented in Table 15-2.

Based on the initial cut-off grade assessment, RPA selected material with a cut-off grade of 0.25% as the basis for which mine planning would occur. RPA used 0.25% U_3O_8 as the input parameter for designing stopes (see Section 16). RPA then visually evaluated stopes with marginal economics as to whether they would be included in the mine plan.

The inputs to the Mineral Reserve cut-off grade included some factors that were developed during the 2017 PEA.

RPA notes that a nominal amount of material between 0.03% U_3O_8 (the regulatory limit between benign waste and mineralized material) and 0.07% U_3O_8, which is uneconomic to process, has been included in the mine plan.

15.5 Comments on Section 15

Mineral Reserves are reported using the 2014 CIM Definition Standards.

As NexGen is listed on the NYSE, RPA tested the Mineral Reserves using a three-year trailing average for the price of U_3O_8. The Mineral Reserves have demonstrated economic viability using a three-year trailing average U_3O_8 price of US$33/lb U_3O_8.

Although the deposit is relatively insensitive to cut-off grade, RPA recommends that a detailed cut-off grade and grade-control program be evaluated in future studies.

Table 15-2: Cut-Off Grade Input Parameters

Parameter	Units	Value	Notes
Mine and mill production	kt	455	
Daily throughput	tpd	1,300	
Uranium grade	% U_3O_8	3.40%	
Uranium recovery	%	97.6%	
Uranium price	US$/lb	45.00	
Exchange rate	C$/US$	0.75	
Uranium price (Canadian)	C$/lb U_3O_8	60.00	
Saskatoon profit royalty payments	%	7%	
Net uranium price realized	C$/lb U_3O_8	55.80	
Contained uranium	lbs	33,286,940	
Payable uranium	%	100.0%	
Payable uranium	lbs	33,286,940	
Value			
Uranium	C$ '000	1,857,411	1
Total	C$ '000	1,857,411	
Unit transport charges	C$ / t prod	740.00	2
Transportation charges	C$ '000	11,173	
Net smelter return	C$ '000	1,846,238	
Value per tonne	C$/t ore	4,058	
Value per % U_3O_8	***C$/% U_3O_8***	***1,193***	
Operating Costs			
Unit underground mining	C$/t proc	130.00	3
Unit processing	C$/t proc	110.00	3
Unit G & A	C$/t proc	70.00	3
Underground mining	C$ '000	59,150	
Processing	C$ '000	50,050	
G & A	C$ '000	31,850	
Total operating costs	*C$ '000*	*141,050*	
Sustaining capital unit costs	C$/t proc	31	3
Sustaining capital costs	C$ '000	14,105	

Parameter	Units	Value	Notes
Tailings storage unit costs	C$/t proc	24	3
Tailings storage	C$ '000	10,920	
Mining incremental @ 60%	C$/t	78	4
Processing	C$/t	110	
G&A	C$/t	70	
Incremental cut-off grade	C$/t	258	
Processing and G&A (pit/portal discard)	C$/t	180	
Var. processing @ 50% + tailings storage	C$/t	55	5
Cut-off grades			
Operating costs and sustaining capital	% U_3O_8	0.31%	6
Operating costs + tailings storage	% U_3O_8	0.28%	7
Incremental (var. mining, proc., G&A, tailings)	% U_3O_8	0.24%	8
Pit/portal discard (proc., G&A, tailings)	% U_3O_8	0.17%	9
Var. processing + tailings storage	% U_3O_8	0.07%	10

Notes to accompany cut-off table:

1. Values are based on the recovered value
2. Assumes delivery of yellowcake to a refinery in Ontario, Canada.
3. Cost assumptions based on the PEA completed in September 2017 and are consistent with the November 2018 PFS.
4. Incremental mining costs assume that 40% of mining costs are fixed annually, and 60% are variable based on tonnage.
5. Variable processing costs assume that 50% of total processing costs are fixed annually, and 50% are variable based on tonnage.
6. This cut-off grade considers the metal needed to cover total operating costs and sustaining capital costs (including excavating tailings chambers).
7. This cut-off grade considers the metal needed to cover total operating costs, and the excavation of tailings chambers; however, it excludes the cost of sustaining capital development.
8. This cut-off grade considers the metal needed to cover variable mining costs, processing, G&A, and the excavation of tailings chambers.
9. Surface cut-off grade (often referred to as the "pit/portal discard") considers that the material has already been mined and brought to surface, and the metal content must be sufficient to cover processing, G&A, and tailings excavations costs.
10. This cut-off grade is only applicable when there is excess capacity in the process plant, and fixed processing costs and G&A costs are already incurred.

16.0 MINING METHODS

16.1 Overview

The vertical extent of the Mineral Reserves extends from approximately 355 m below surface to 655 m below surface.

The main objectives of the mine design for the Arrow deposit were:

- Ensure minimization and effective management of radiation exposure to operating personnel
- Develop and design the mine in a way that results in the likely regulatory approvals
- Provide a safe working environment for all site personnel
- Achieve buy-in from local stakeholders
- Minimize the mine environmental footprint
- Minimize mine construction complexity
- Maximize operational reliability
- Maximize the extraction factor of high-grade veins
- Minimize the operating and capital costs necessary to achieve safe production.

16.2 Geotechnical Considerations

Geotechnical analysis and designs were carried out by BGC Engineering Inc. (BGC).

16.2.1 Overview

BGC identified eight geotechnical domains in the Arrow deposit area:

- Domain 1 (D1): Overburden (sand, cobbles, boulders)
- Domain 2 (D2): Cretaceous (siltstone/mudstone/sandstone/coal seams)
- Domain 3 (D3): Devonian (sandstone)
- Domain 4 (D4): Athabasca (sandstone)
- Domain 5 (D5): Paleo-weathered basement (gneiss)
- Domain 6 (D6): Faults and shear zones (gneiss, altered gneiss, possibly mineralized)

- Domain 7 (D7): Altered basement (altered gneiss, possibly mineralized)
- Domain 8 (D8): Unaltered basement (gneiss).

Domains 2 to 4 represent sedimentary geological units and would be encountered during shaft sinking only. Domains 5 to 8 are basement lithologies that have undergone variable intensity of weathering and/or hydrothermal alteration. Within Domain 8 is a distinct hanging wall domain that constrains the hanging wall of the A3 to A5 mineralized zones.

16.2.2 Access

Multiple shafts have been sunk throughout the Athabasca Basin by contractor companies, and the technical and operational capacity exists to sink shafts in the vicinity of the Arrow deposit.

Where possible, all shafts should be in areas with the least overburden cover, minimal sedimentary unit thickness, relatively flat unconformity, and away from prominent faults or shear zones, as these factors can negatively impact shaft sinking rates, decrease stability, and increase groundwater inflow risks.

Expected challenges in shaft construction include:

- Saturated and unconsolidated overburden at shaft foundation and sinking locations
- Very poor to fair quality sedimentary rock underlying the overburden, potentially resulting in significant groundwater inflows during shaft excavation if left unmanaged
- The unconformity contact and paleo-weathered zone consist of poor-quality rock, improving to fair/good quality rock at greater depth. Groundwater inflow rates from this domain are unknown; however, given increased fracture and proximity to the sandstone aquifer above, inflows are anticipated.

Initial shaft sinking will be through domains D1 through D4, which at the proposed shaft locations consist of 40 m of overburden, 60 m of sedimentary rock, and 25 m of paleo-weathered basement rock with a combined thickness of 125 m. These domains consist of poor to very poor-quality rock masses; however, once these have been artificially frozen, they are not anticipated to be problematic.

16.2.3 Crown Pillar

The crown pillar is referred to as the rock mass separating the unconformity from the uppermost mine workings. Based on the information used in the 2018 Pre-feasibility Study, the minimum distance between the shallowest mine excavation and the unconformity is approximately 250 m. This drastically reduces the risks associated with the pillar and therefore has not been investigated in detail.

16.2.4 Stope Dimensions

Based on the stope stability assessments, the maximum recommended supported transverse stope back dimensions are outlined in Table 16-1. The stope back dimension recommendations for longitudinal stopes is outlined in Table 16-2. The transverse stope width is parallel to mineralization and the stope length is perpendicular to mineralization. For stope walls the maximum recommended unsupported and supported longitudinal and transverse stope wall dimensions are summarized in Table 16-3.

BGC has divided the mine blocks in elevation m(asl) as follows, based on approximations from the PEA (RPA, 2017). Although the PFS mine plan stopes are now greatly reduced in vertical extent, reference can be made to Figure 16 2: Transverse Longhole Mining Method Schematic.

- Block B between 145 to 235 m, 'Upper Block'
- Block C between 0 to 145 m, 'Upper Block'
- Block D between -125 to 0 m, 'Lower Block'.

The recommended stope dimensions are as follows:

- Transverse stopes recommended supported span is 10 m to 15 m, at a stope length of 30 m to 15 m, respectively. The strike length is controlled by the hanging wall stability. Increased strike lengths are possible with reduced level spacing
- Longitudinal stopes in Mine Blocks B, C, and D, which have both supported and unsupported back spans (cable bolt supported when span greater than 7.5 m), the recommended stope length is limited by the unsupported walls at 20 m, considering the median Q' and N' for unsupported stopes. Stope lengths greater than 20 m would require cable bolt support in the walls. This stope length will increase with reduced level spacing.

Table 16-1: Recommended Transverse Stable Supported Back Dimensions

Stope Width (m)	Supported Stope Length (m)		Recommended Supported Stope Back Length (m)
	Lower	Upper	
10	20	>50	25
15	***10***	***18***	***15***

Table 16-2: Recommended Longitudinal Stable Unsupported and Supported Back Dimensions

Mine Block	Stope Width (m)	Stope Length (m)				Recommended Unsupported Stope Back Length (m) [1]	Recommended Supported Stope Back Length (m)
		Unsupported		Supported			
		Lower	Upper	Lower	Upper		
B, C, D	4	22.5	>50	>50	>50	22.5	>50
	6	7.5	>50	>50	>50	25	>50
	8	***5.0***	***>50***	***>50***	***>50***	***10***	***>50***
	10	5.0	15.0	30.0	>50	- [1]	30
	15	4.0	10.0	17.5	30.0	- [1]	20

Note: Spans > 7.5 m require cable bolts based on kinematic analysis, so no unsupported spans are noted. Highlighted cells indicate for B, C and D, 90% of the longitudinal spans are <8 m. These are considered the representative maximum spans.

Table 16-3: Recommended Stable Length Dimensions for Transverse and Longitudinal Stopes

Mine Block	Mine Method	Stope Length (m)				Recommended Unsupported Stope Length (m)	Recommended Supported Stope Length (m)
		Unsupported		Supported			
		Lower	Upper	Lower	Upper		
B, C, D	Longitudinal	15.0	25.0	35.0	>50	20	35
C, D	Transverse	7.0	12.0	16.0	25.0	15[1]	20

Note: Recommended dimensions with median Q' in altered basement. Indicates 'typical' stope length, i.e., that supported by median data sets.

All stope dimensions assume that good blasting practices will be employed to enhance stability by minimizing damage to the walls and stope backs. It is also assumed that the paste backfill will be of good quality and placed in a timely manner, and that the stope backs will be supported with cable bolts as required, with hanging wall/footwall cables supported where locally required.

Risks to achieving the anticipated design include undefined large-scale fault or shear structures, groundwater pressures in weak zones that cannot be effectively de-pressurized, the presence of adversely oriented discontinuities which could impact stope stability (although the evidence was presented to demonstrate that unfavorable conditions exist for the hanging wall, backs and footwall of all stopes considered), more pervasive and extensive poor to very poor quality ground conditions within and proximal to mineralization than is indicated in the current data, and the ability (or lack) of uranium paste tailings to consistently develop adequate strengths.

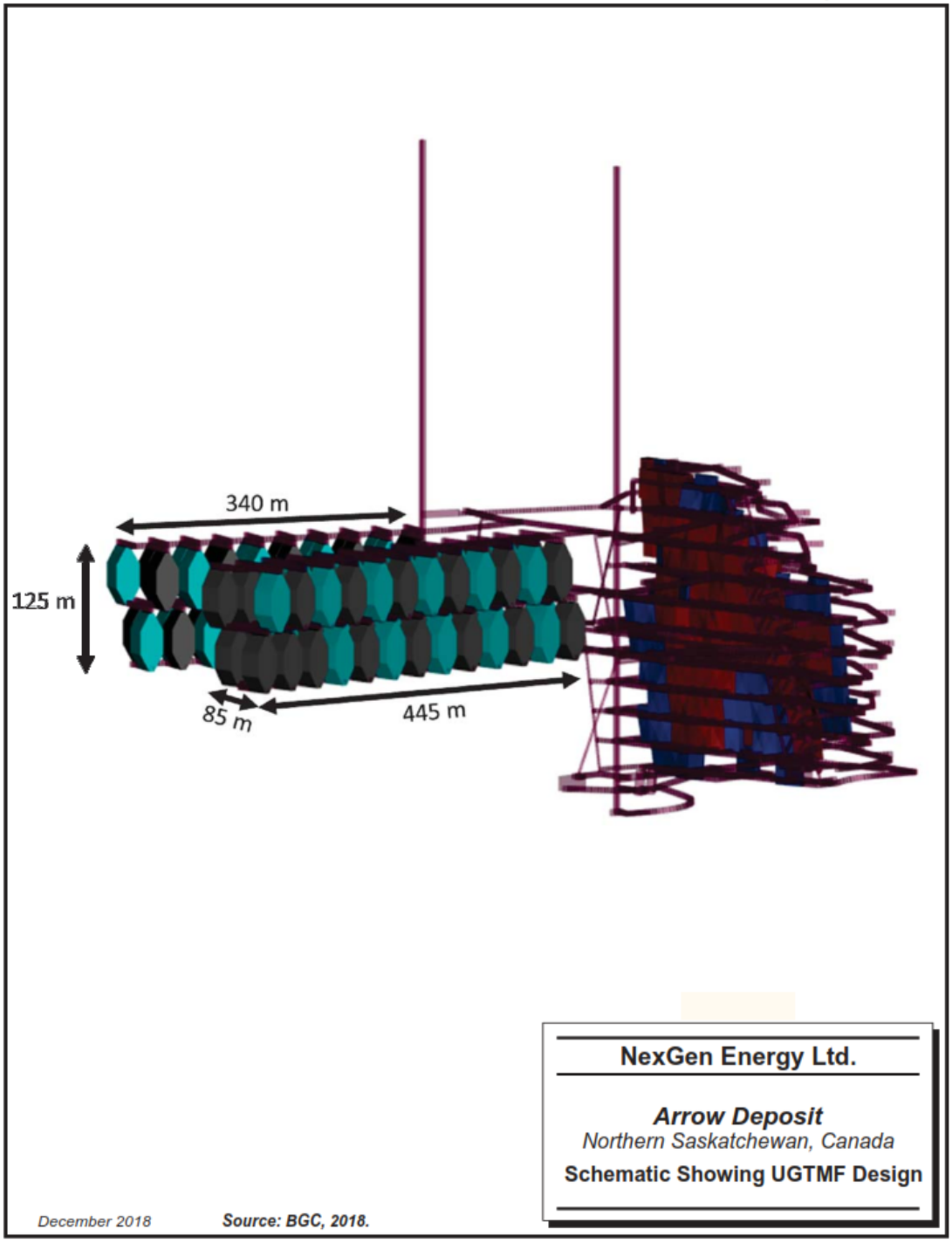
16.2.5 Underground Tailings Management Facility

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 of the deposit, in an area that is interpreted to have relatively minimal alteration or fault or shear structures.

Risks with the geotechnical assumptions are that there are minimal drill hole data in the area. The location proposed in the 2018 Pre-feasibility Study has not been investigated on a local basis, and the closest hole with RMR' data is on the proposed western boundary of the facility. As a long-term facility, the functional service life will have an impact on the required level of ground support and assessment of the blasted (and presumably unsupported) excavation walls. Also, the stability and hydraulic conductivity of the rock mass in and proximal to the underground tailings management facility (UGTMF) are likely to require relatively conservative design approaches due to the long service life which will require detailed site investigations and monitoring programs.

The 2018 Pre-feasibility Study proposes 88 diamond-shaped excavations each approximately 25 m wide by 25 m long by 60 m high. The excavations will be arranged in a regular pattern with 45 m rib pillars separating primary excavations, and 10 m between primary and secondary openings. The UGTMF design is shown in Figure 16-1.

Figure 16-1: Schematic Showing UGTMF Design



Note: Figure prepared by BGC, 2018.

The top of the excavations will be approximately 250 m below the unconformity. Sequencing of chamber construction will be important so that a new chamber is not constructed until the adjacent cell is backfilled and cured, to maintain stability. In order to excavate to the proposed dimensions, the paste will have to be "self-supporting", i.e., will require some cement and strength to provide effective confinement and support of the cell walls. Additional information is included in Section 16.5 and Section 16.6.

16.2.6 Backfill

Backfill of mined stopes is planned to use a combination of process waste, cement, potential fillers (such as fly ash), and water. Tailings not used for paste backfill will be stored in the UGTMF.

A by-product of the processing will be other waste products such as gypsum and effluent precipitates. The 2018 Pre-feasibility Study proposes to store the gypsum and effluent precipitates in secondary chambers within the UGTMF, comprising approximately 50% of the total chamber volume. A cement-based paste will be used at the base of the cells in order to maintain confinement and contain the unconsolidated gypsum. RPA noted that gypsum settlement is anticipated so it will be important for future work to assess the optimal UGTMF cell excavation and fill sequencing.

The stope stability assessments, and subsequent recommended stope dimensions, require that the backfill is tight to the walls.

High-strength backfill will be required in the primary and secondary stopes immediately above the sill pillar to allow for undercutting and working under engineered fill (for sill pillar recovery and mining of the stope below the pillar).

There will be only one sill pillar in the 2018 Pre-feasibility Study mine plan, the base of which is at elevation 30 masl (refer to Section 16.4).

The backfill for the upper primary and secondary stopes is designed at a lower strength for vertical exposure stability within open stopes only. The required paste fill strength is based on similar operations and past experience to provide minimum strength requirements for the transverse and longitudinal stopes.

16.3 Hydrogeological Considerations

The proposed options for dewatering the mine are to use a "dirty water" system to the 500 m level, operating both submersible and centrifugal pump systems.

The dewatering system will be capable of collecting and removing all strata and operational process water from the mine infrastructure, ongoing developments, operational stopes, and shaft inflow, pastefill seepage and UPWNF seepage during the filling process, and collecting it at the main dewatering station sumps located at 500 m level. This water is considered to be "dirty water" (i.e., water with entrained solids).

All water received at the 500 m level sumps will be either be settled to allow solids drop out or suitably clarified utilizing ancillary equipment to produce clean water and a slimes/slurry. The clean water is to be pumped to the surface clean water holding facility while the slimes/slurry is to be pumped or otherwise transported to the UGTMF for containment underground.

16.4 Mining Method

16.4.1 Selection

Two longhole mining methods will be used to extract the ore:

- Transverse stope mining
- Longitudinal retreat stope mining.

Both longhole mining methods will use paste backfill to provide ground stability. This combination of longhole stoping using paste backfill will provide a combination of good productivity, high extraction, and stable back support.

16.4.2 Extraction

All stope designs were completed using the Deswik Stope Optimizer (DSO) tool. The longhole stope design parameters were based on 30 m stope heights for both transverse and longitudinal stopes. For transverse stopes, the width of the primary and secondary stopes was set at 15 m, while a maximum strike length was not specified. For longitudinal stopes, the width of the stopes was set at a minimum of 3.0 m (including 1.0 m of dilution), and no maximum width, and a strike length of 15 m. After an initial result from DSO, a secondary evaluation of stopes with heights of 10 m and 20 m was completed to capture material that was not converted into mine shapes during the first pass.

Transverse longhole mining will be done using primary and secondary stoping methods. The order in which stopes are extracted is largely driven by the head grade, with the overarching goal of achieving annual production of 30.0 M lbs U_3O_8. 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.

Figure 16-2 is a schematic showing the proposed transverse and longitudinal mining method. Figure 16-3 is a schematic showing a section of the mine looking north. Figure 16-4 is a schematic showing a section of the mine looking north. An example level plan is included as Figure 16-5.

16.4.3 Access

The Arrow deposit is planned to be accessed using two shafts. Both shafts will be located in the footwall of the deposit. They will both have an internal diameter of 6.5 m, and will be blind-bored concurrently, using conventional shaft sinking technologies.

The first shaft will be used as a production shaft, and for transportation of personnel and materials into the mine. Shaft #1 will be sunk to a depth of 658 m below surface. Shaft #1 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. Shaft #1 will have a permanent headframe and hoisting house.

The second shaft will be used as an exhaust ventilation shaft. Shaft #2 will be sunk to a depth of 533 m below surface. An emergency escapeway system will also be installed in Shaft #2.

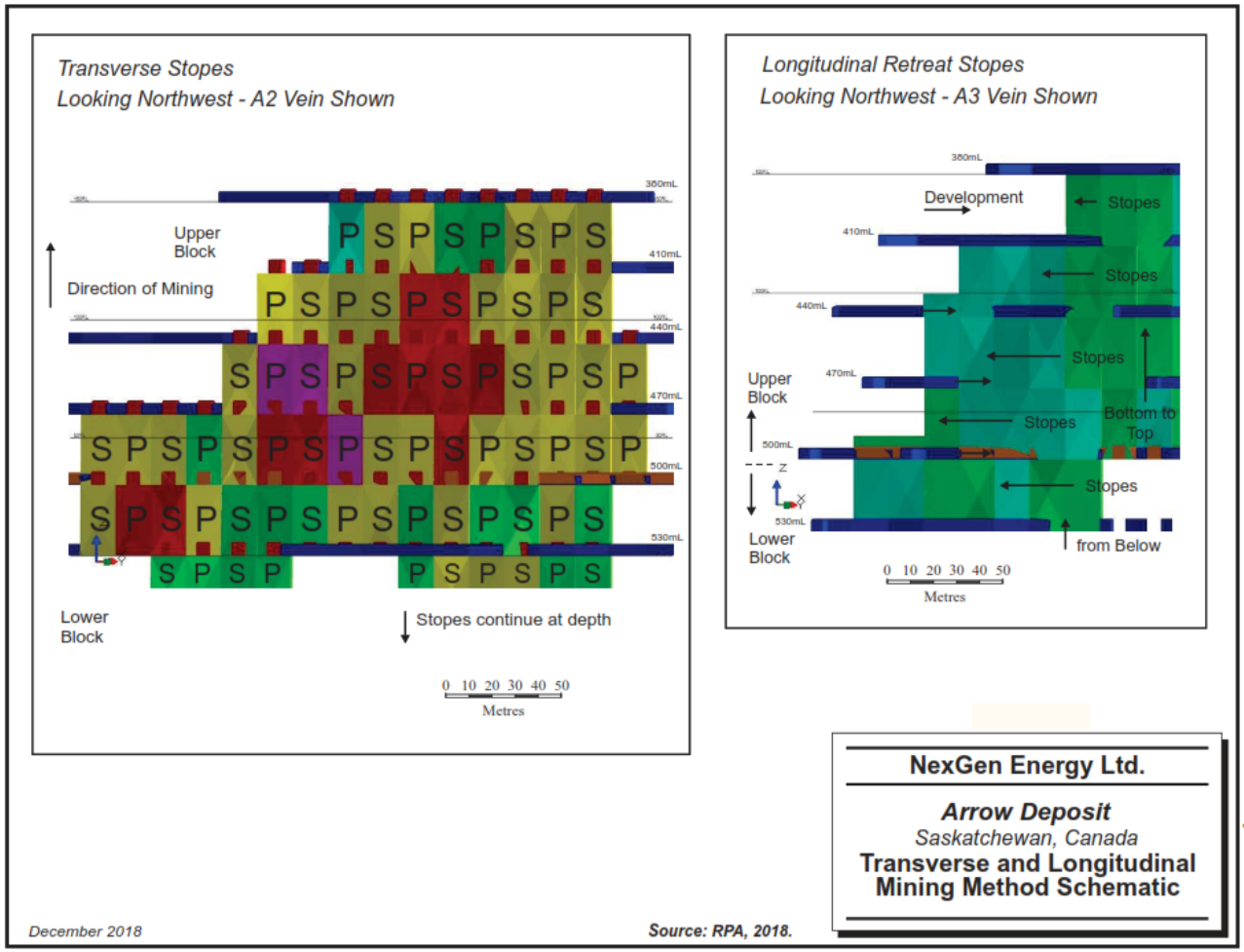
A total of eight shaft stations are included in the mine design (Table 16-4).

16.4.4 Lateral Development

Lateral development will be taken both from Shaft #2 and Shaft #1. Shaft #2 will be completed in advance of Shaft #1, and therefore much of the lateral development completed during the pre-production period will originate from Shaft #2.

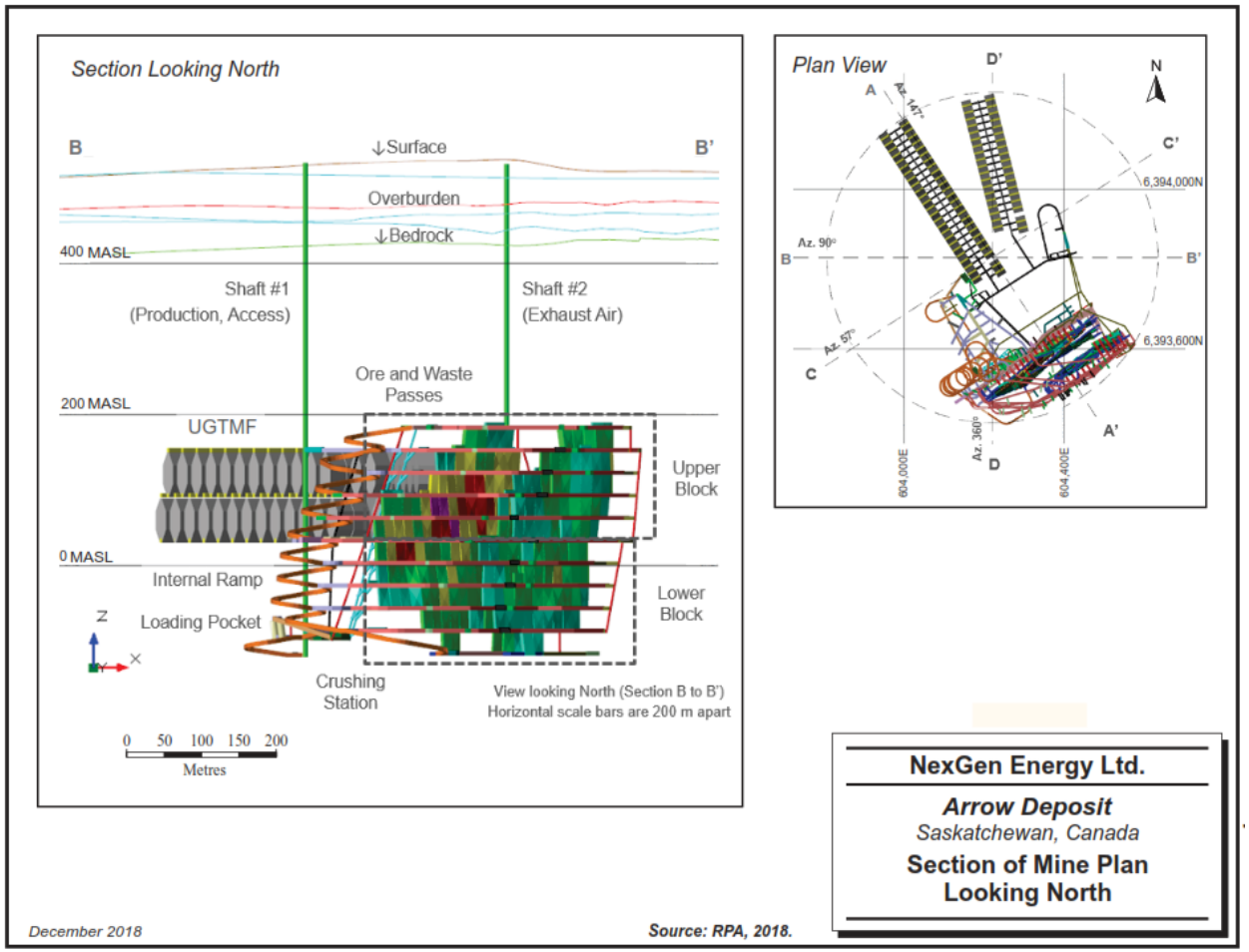
Level spacings at Arrow have been planned at 30 m intervals, sill to sill. There will be a total of 11 main levels in the mine (Table 16-5). In addition to the lateral development, there will be an internal ramp system that will connect all mining levels. Ramps will be driven at 5.0 m wide x 6.0 m high with a maximum gradient of 15%, and will have tapered gradients at the intersections of each level.

Figure 16-2: Transverse and Longitudinal Mining Method Schematic



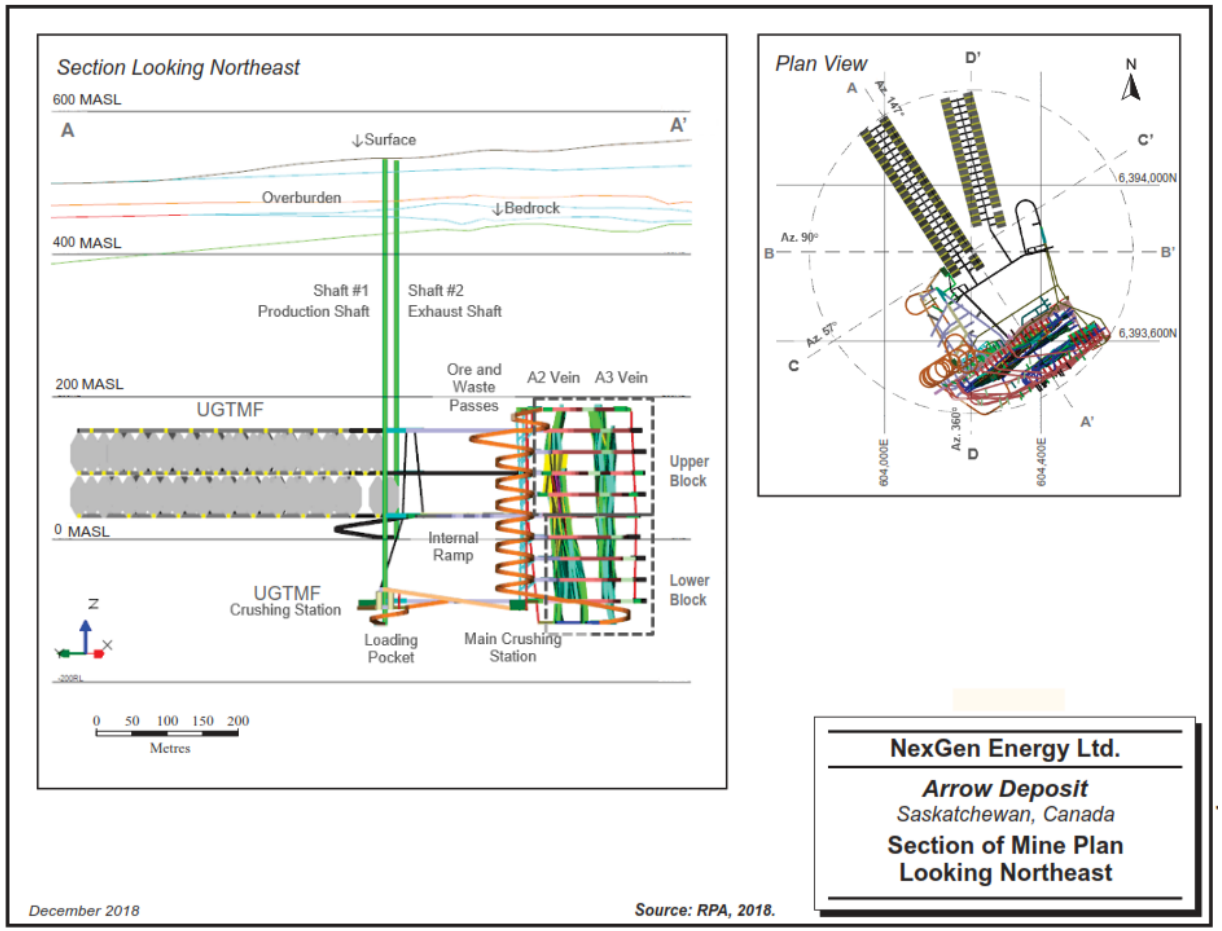
Note: Figure prepared by RPA, 2018.

Figure 16-3: Section of Mine Looking North



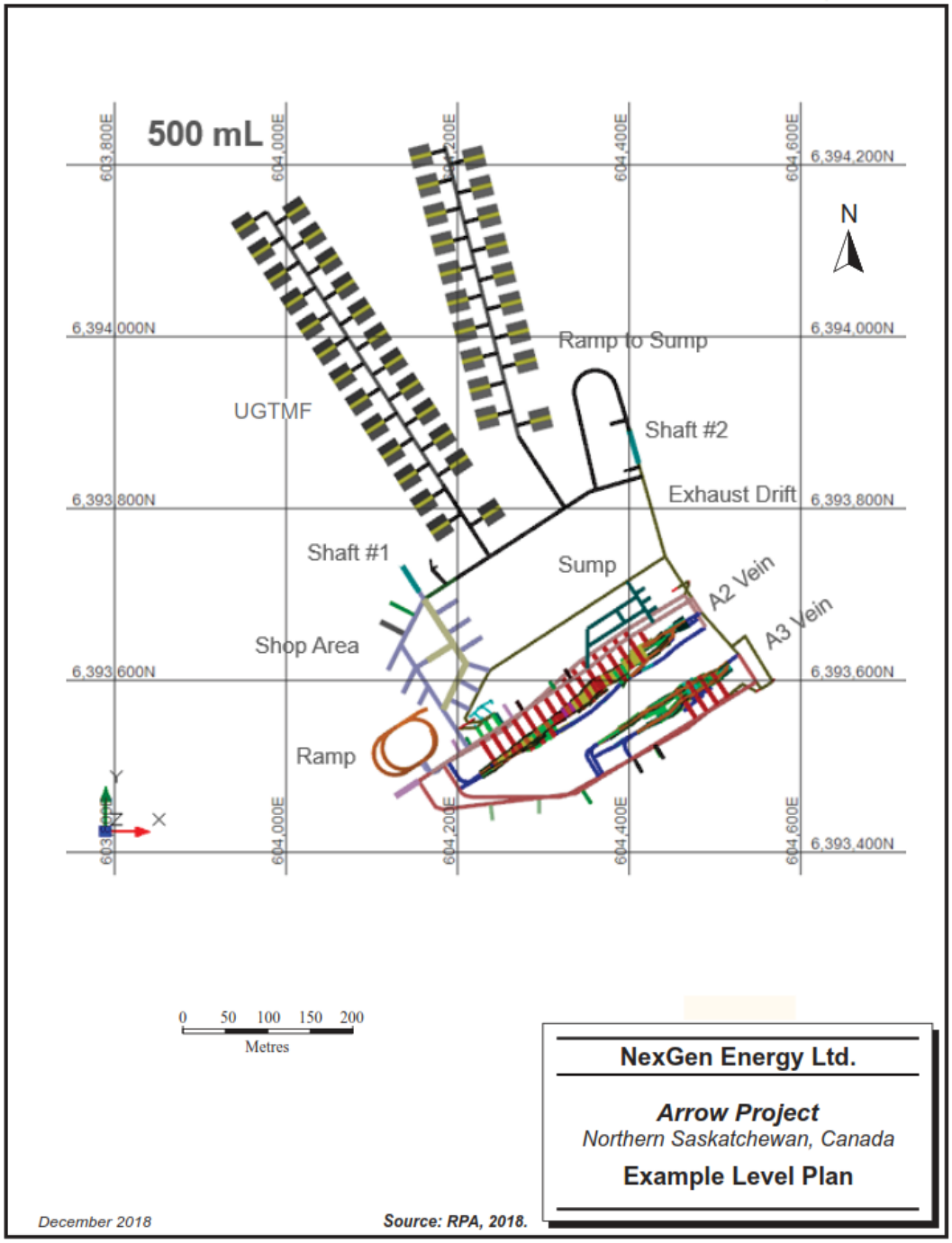
Note: Figure prepared by RPA, 2018.

Figure 16-4: Section of Mine Looking Northeast



Note: Figure prepared by RPA, 2018.

Figure 16-5: Example Level Plan



Note: Figure prepared by RPA, 2018.

Table 16-4: Shaft Stations

Shaft	Shaft Station Name	Approximate Depth from Surface (m)	Elevation (masl)	Purpose
#1	380 m level	385	150	Overcut of Block One
#1	500 m level	505	30	Main Level, undercut of Block One, overcut of Block Two
#1	620 m level	625	-92	Undercut of Block Two, loading pocket
#1	650 m level	658	-122	Shaft bottom, sump
#2	380 m level	383	150	Overcut of Block One
#2	440 m level	443	90	Overcut of UGTMF bottom lift
#2	500 m level	503	30	Main level, undercut of Block One, overcut of Block Two
#2	530 m level	533	0	Shaft bottom, sump

Table 16-5: Mine Levels

Mine Level	Approximate Depth from Surface (m)	Sill Elevation (masl)	Shaft #1 Station	Shaft #2 Station	Level Description
350 m level	355	180	N	N	Top level of mine, some minor stopes will also be taken with Uppers
380 m level	385	150	Y	Y	Shaft station on both Shaft #1 and Shaft #2, production level, overcut for UGTMF
410 m level	415	120	N	N	Production level
440 m level	445	90	N	Y	Production level, middle level of UGTMF
470 m level	475	60	N	N	Production level
500 m level	505	30	Y	Y	Shaft Station on both Shaft #1 and Shaft #2, as well as main mine level, undercut to UGTMF, production level
530 m level	535	0	N	Y	Shaft bottom on Shaft #2, production level
560 m level	565	-30	N		Production level
590 m level	595	-60	N	—	Production level
620 m level	625	-90	Y	—	Includes loading pocket, ore bins, underground crusher stations, production level
650 m level	655	-120	Y	—	Shaft bottom on Shaft #1, also deepest production level for mining area

Lateral development is based on a number of heading sizes (Table 16-6). All lateral development, including ore headings, will be driven using two-boom jumbos. Load–haul–dump vehicles (LHDs) will transport material to waste and ore passes located at each level. Underground haul trucks will not be required for material haulage in the current mine plan. However, if mining activities extend further away from the shafts and planned material handling systems, underground haul trucks could be required. Therefore, headings have been sized appropriately if haul trucks ever needed to be brought into the mine.

All lateral waste development will have arched back profiles. This design has been chosen to improve the stability of openings. The location of waste development has been chosen so as not to interfere with any major known fault structures or adverse ground conditions.

Lateral development will be concentrated in three years (Year -1, Year 1, and Year 2). This initial build-out of lateral development includes the internal ramp, material handling systems, UGTMF development, and a number of footwall and hanging wall access drifts. Beyond Year 2, the remaining lateral development will include a continuation of some footwall and hanging wall development, as well as the stope access drifts.

16.4.5 Ore Development

Longitudinal stope development will use a 5.0 m by 5.0 m heading profile, while transverse stope development will use a 6.0 m by 6.0 m profile. The decision to use a larger profile for the transverse stopes is based on a number of factors:

- Wider overcuts and undercuts will improve the ability for mining equipment (longhole drilling machines and scooptrams) to effectively drill and muck the stopes
- Greater width of drifts potentially results in lower radiation exposure to personnel, based on the tenets of time, distance, shielding, and ventilation.

When determining whether stopes would be developed as either transverse or longitudinal, one of the criteria was the estimated U_3O_8 grade of the development. One of the goals of the 2018 Pre-feasibility Study design was to minimize the amount of development greater than 4% U_3O_8. The 4% U_3O_8 threshold for lateral development was provided by Arcadis, a company with past experience providing radiological monitoring and modeling services to operating uranium mines in Saskatchewan.

Table 16-6: Headings

Heading Size (W x H)	Capital (CAP) or Operating (OP)	Waste (W) or Ore (O)	Level Description	LOM Total (m)
5.0 x 5.0	OP	W	Access to longitudinal retreat stopes	1,777
5.0 x 5.0	OP	O	Ore development in longitudinal retreat stopes	4,634
6.0 x 6.0	OP	W	Access to transverse stopes	3,330
6.0 x 6.0	OP	O	Ore development in transverse stopes	2,528
Total Operating				***12,269***
4.0 x 5.0	CAP	W	Ventilation drifts	2,167
5.0 x 6.0	CAP	W	Footwall and hanging wall access, internal ramp, and UGTMF access	22,559
5.0 x 7.0	CAP	W	Sumps, with high backs for overhead cranes, and catwalk access	688
6.0 x 6.0	CAP	W	Conveyor drifts, level access drifts	1,680
7.0 x 5.0	CAP	W	Refuge stations	70
8.0 x 8.0	CAP	W	Shop areas with overhead cranes, and room for equipment to drive past	213
MISC	CAP	W	Underground crusher stations with irregular shapes	49
Total Capital				***27,427***
Total Lateral Development				***39,696***

Over the course of the life-of-mine (LOM), there will be 939 m of lateral development with rounds having grades >4% U_3O_8. In addition to ore development, there will be some lateral development taken through paste fill, in order to extract the uppermost level of the Lower Block.

16.4.6 Non-Shaft Vertical Development

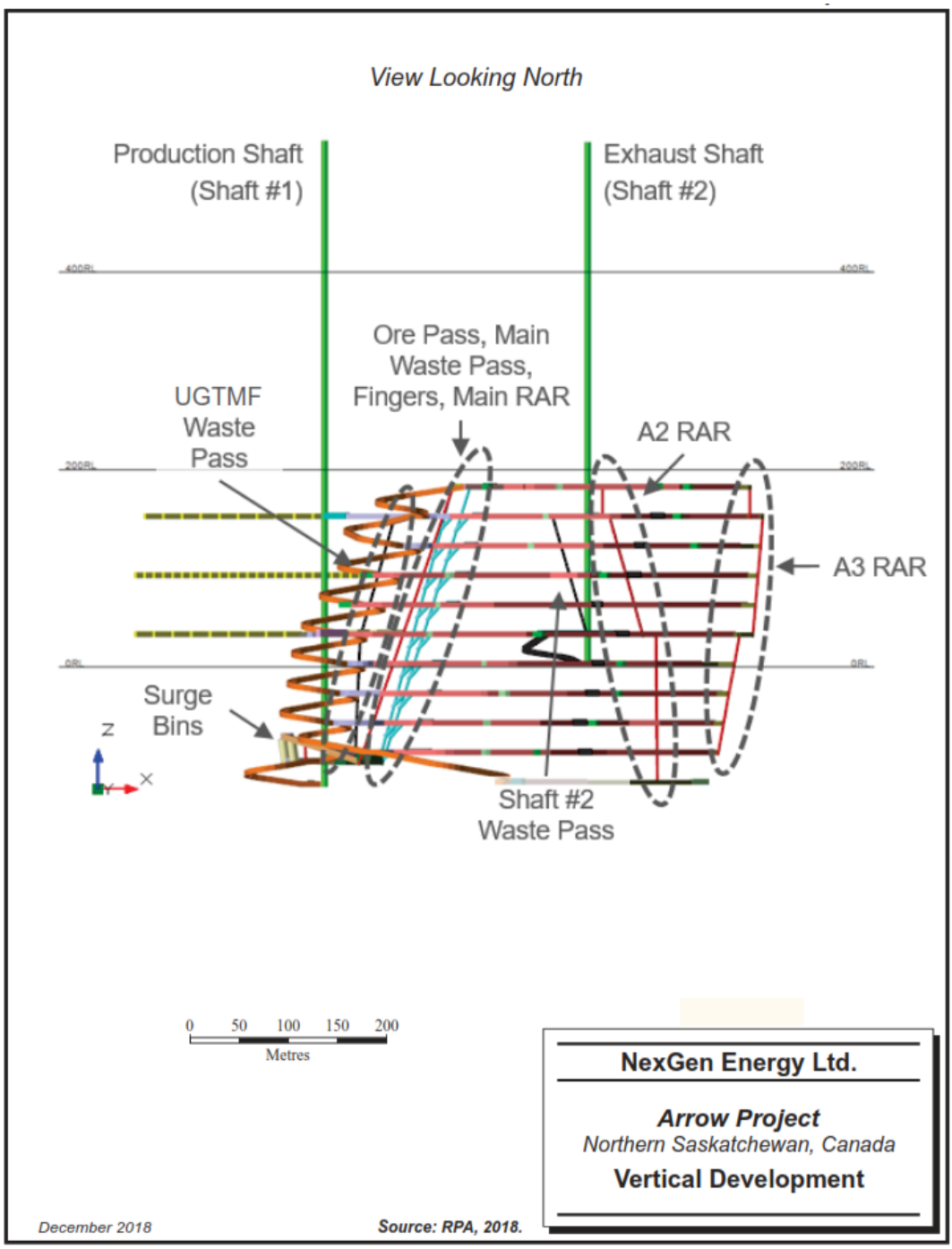
In addition to the two shafts connecting to surface, there will be several vertical developments at Arrow (Table 16-7; Figure 16-6).

Table 16-7: Non-Shaft Vertical Development

Name	Size	Excavation Method	Description	LOM Total (m)
Ore pass	3.0 m Ø	Raisebore	Pass to consolidate ore from mine into Main Crushing Station	296
Main waste pass	3.0 m Ø	Raisebore	Pass to consolidate waste from the mine into the Main Conveyor Belt	296
UGTMF waste pass	3.0 m Ø	Raisebore	Waste pass for material from UGTMF to UGTMF Crushing Station	246
Shaft #2 waste pass	3.0 m Ø	Raisebore	Used as a waste pass during early development, converted to a return air raise for the UGTMF	125
Main return air raise	3.0 m Ø	Raisebore	Return air raise for waste and ore pass, and southwest portion of mine	329
A2 return air raise system	3.0 m Ø	Raisebore	Return air raise for the A2 vein	307
A3 return air raise system	3.0 m Ø	Raisebore	Return air raise for the A3 vein	268
Finger connections	2.0 m x 2.0 m	Drill and blast	Dump point into ore or waste pass	300
Surge bin – ore	10.0 m Ø	Drill and blast	Surge bin to hold ore prior to skipping to surface.	25
Surge bin – waste	10.0 m Ø	Drill and blast	Surge bin to hold waste prior to skipping to surface.	25
Total Non-Shaft Vertical Development				***2,218***

Note: Ø = diameter

Figure 16-6: Non-Shaft Vertical Development



Note: Figure prepared by RPA, 2018.

In the footwall of the A2 vein, near the southwest end of the mining area, one ore and one waste pass (two total) will extend the vertical height of the deposit. A series of "fingers" will connect each level into the two passes. The ore and waste passes will also be connected to an exhaust ventilation raise to ensure that dust from the ore and waste passes does not enter into areas of the mine with operating personnel. There will also be a separate waste pass that will service the UGTMF.

Each of the A2 and A3 veins will have a series of exhaust ventilation raises at the northeast extent of the production area (refer to Section 16.6). Two vertical storage bins will be excavated near the loading pocket, to serve as surge bins for skipping ore and waste, and to provide operational flexibility is skipping material to surface.

A minimal amount of ground support has been planned for in the vertical development, as these areas are not intended to have personnel, and will be used exclusively for ventilation or material handling. RPA recommends that ground control plans for vertical development be evaluated at the next level of study.

16.5 Underground Tailings Management Facility

The UGTMF has been designed so that all solids waste streams generated from the process plant will be returned underground for long-term storage. Two separate products will be returned underground:

- Paste tailings
- Combined precipitates.

The paste tailings will contain a binder content ranging from 4% to 7% depending on the application and required uniaxial compressive strength (UCS). The binder will use a combination of Portland cement and ground slag.

The combined precipitates will be a mixture of three waste streams:

- Gypsum precipitate
- Effluent precipitate – low pH precipitate filter cake
- Effluent precipitate – high pH precipitate filter cake.

The UGTMF is designed as a series of open excavations, with pillars left between each excavation.

The UGTMF will occur in two lifts, and each lift will have two "wings" of chambers. Each "wing" will have chambers off both sides of the drift. This layout ensures that storage can be achieved with the minimum amount of capital development (see also discussion in Section 16.2.5).

The UGTMF will be accessed by three levels:

- 380 m level: overcut of top lift
- 440 m level: overcut of bottom lift, undercut of top lift
- 500 m level: undercut of bottom lift.

The dimensions of each excavation will be as follows:

- 60 m height (two mining levels)
- 25 m strike
- 25 m width
- 10 m pillar between excavations.

Each excavation will have a volume of 28,400 m^3, as the shape of each chamber has been optimized for ease of longhole drilling, blasting, and mucking. A total of 88 chambers will be required over the LOM. Within the 88 chambers, 47 will be used to store paste tailings, and 41 will be used to store combined precipitates. In the 41 chambers that will store combined precipitates, an engineered dam (or plug) of high strength paste tailings will be used as a bulkhead to prevent the combined precipitates from flowing into the drifts. During the chamber filling and during the consolidation process of the fill, chamber drains, pipes and valves will be installed to direct water back to the main u/g sump for water quality monitoring and effluent treatment on surface.

It is envisioned that at any given time, two chambers would be available for deposition, two chambers are in the process of mucking, another chamber is blasted however not mucked out, and another chamber is drilled off, however, not blasted. Fill times and "open exposure" times should be minimized wherever practical. Approximately 10 chambers will be completely cycled (i.e. developed, drilled, blasted, mucked out, and filled again) per year.

16.6 Backfill

The creation of paste tailings is directly proportional to the amount of material processed through the plant. For each tonne of processed material, 0.84 m^3 of paste tailings will be created, along with 0.31 m^3 of combined waste precipitates. The paste tailings will be mixed with a binder, to achieve a certain fill strength. The amount of binder will depend on the paste application, with primary transverse stopes and longitudinal stopes with adjacent stopes on strike requiring the highest level of binder. The combined precipitates will not receive any binder, and are therefore considered to be an unconsolidated mass. The deposition of the combined precipitates will be limited to the UGTMF, and possibly some secondary stopes or longitudinal stopes at the fringes of the production area. Based on a steady-state production rate, total fill demand is nominally 328,000 m^3 per year for paste tailings, and 121,000 m^3 per year for combined precipitates.

The paste fill system is designed to be operable 24 hours per day, although the plant will likely operate for less than this, given the time required to change filling locations underground. An underground backfill crew will install backfill lines, build barricades, divert paste to the ore drifts and UGTMF, and monitor filling progress.

It is assumed that the binder content will be a 50:50 mix of Portland cement and ground slag. The paste will be approximately 67% solids (by weight), and the combined precipitate will be approximately 55% solids (by weight).

The paste plant is proposed to be located in the vicinity of the process plant, and there will be three boreholes that transport the paste and combined precipitates into the mine, exiting at the 380 m level. Available stopes will be filled as the first priority. If no stopes are available to be filled, the material will be sent to the UGTMF. The UGTMF will have dedicated chambers for both paste tailings and combined precipitates.

16.7 Ventilation

16.7.1 Design Considerations

Ventilation design guidelines were used for design of the primary, secondary, and auxiliary ventilation systems to meet or exceed all regulatory requirements. Ventilation planning was also based on inputs from Arcadis, a specialized consultant in radiological monitoring. Airflow requirements address both radiation protection and diesel exhaust emission control.

Design guidelines and details used for the ventilation system are:

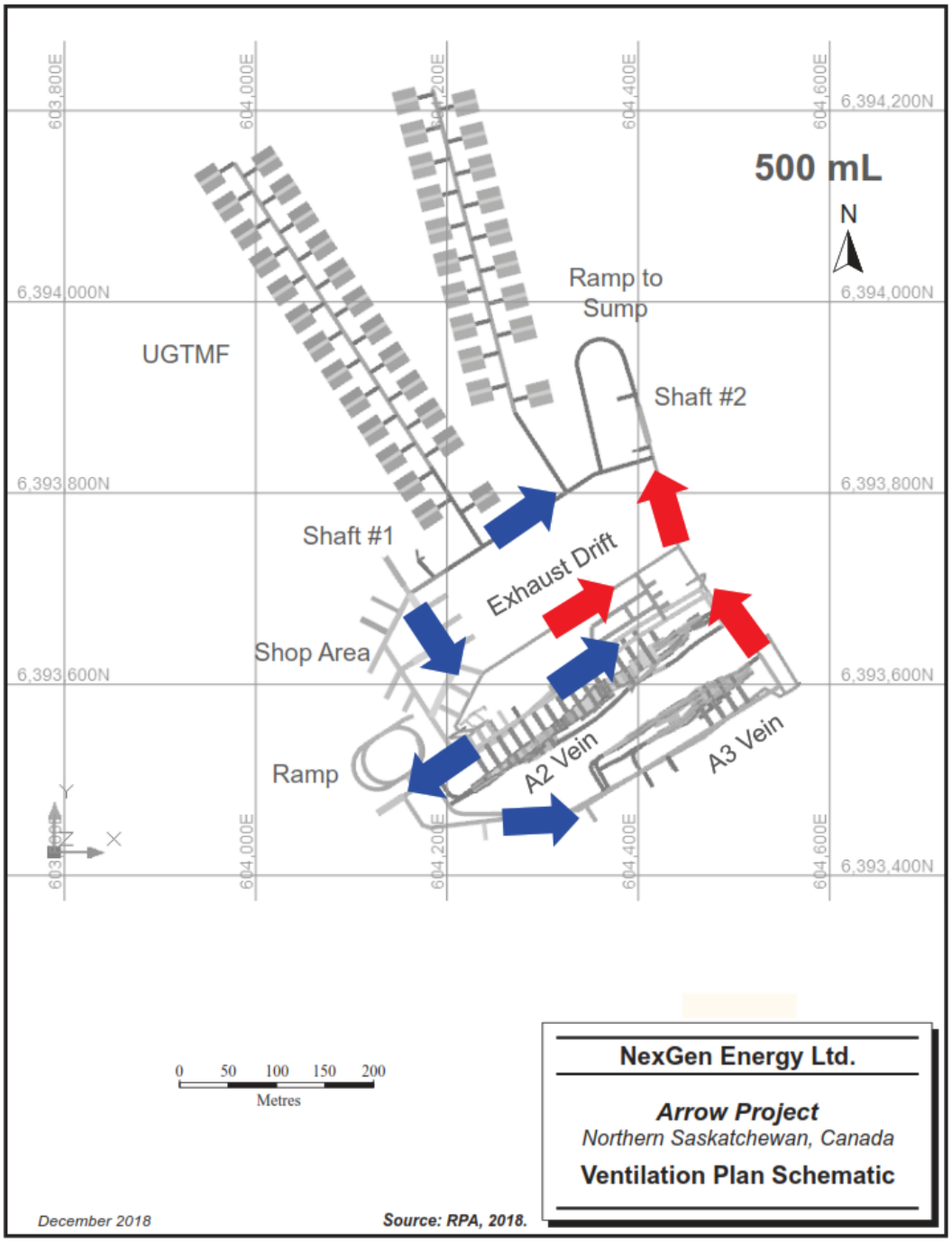
- Airflow exposed to a radiation source will be single pass air. During waste development and construction periods, where multiple headings are being developed simultaneously, some recovery of air between development headings and construction areas is allowable provided the CANMET engine rating requirements are met, and radiological limits are not exceeded
- General airflow throughout the mine will be from areas with a lower contamination potential to areas with a higher contamination potential
- Any leakage and short-circuiting that may occur within the ventilation systems and between systems should be from the airflow with the lower contamination potential to the airflow with the higher contamination potential
- Back-up systems will be established on critical system components where the failure of such component could compromise worker health and safety
- All ventilation procedures and designs are based on conventional ventilation technology using standard, commercially-available equipment
- Air quality will be monitored to ensure contaminant levels do not exceed regulatory limits as specified in the CNSC Regulations and Guidelines.

Computer modelling using Ventsim Visual was completed to create a model of the mine as part of the ventilation planning process. The model was used to determine the pressures required to move sufficient air through the mine workings and meet the airflow requirements.

The airflow determination by "activity allocation" was used to calculate the total airflow quantity required. The estimated calculated airflow total is 330 m^3/s.

The ventilation system is designed as a predominately negative or "pull" system delivering 330 m^3/s through Shaft #1. Fresh air will be distributed throughout the mine by three shaft stations from Shaft #1 and an internal ramp. The fresh air fans will provide sufficient pressure so that the headframe on top of Shaft #1 will have positive pressure over the outside air. The fresh air fan requirements are two parallel 223.7 kW (300 hp) fans at 165 m^3/s (350,000 cfm) each at a static pressure of 0.9 kPa. A heater house will also be installed at Shaft #1 fresh air fan installation. The exhaust fan requirements are two parallel 596.6 kW (800 hp) fans at 165 m^3/s (350,000 cfm) each at a static pressure of 2.3 kPa. Figure 16-7 is a schematic showing the planned ventilation system.

Figure 16-7: Ventilation Plan Schematic



Note: Figure prepared by RPA, 2018.

16.7.2 Primary Ventilation

Three access drifts will connect Shaft #1 to the three primary levels. A spiral ramp will connect the sublevels to each other and will also be used as the primary passageway for airflow to sublevels. Fresh air will enter each sublevel and flow through the stope access drifts to the return air raise located at the opposite side of the ramp. As the fresh air flows past the stope cross-cuts, it will be drawn into the stope by the auxiliary ventilation system as required. Each sublevel stope access drift will be connected to an internal return air raise.

16.7.3 Shaft #1 and Skipping Compartments

Two parallel fresh air supply fans will be located on surface at Shaft #1. These fans will ensure adequate airflow through the mine air heaters, provide for upcasting of the headframe, and supply down casting of Shaft #1.

The nature of the ore at the Project requires that air exposed to ore is considered radiation-contaminated and cannot be used in active working areas. This requires the separation of the Shaft #1 skipping compartment from the other compartments that are used for the passage of fresh air into the mine. It is planned that approximately 15 m^3/s of airflow will be required to ventilate the skip compartment. Air from the skip compartment will be used to ventilate the skip loading pocket, conveyor way and the underground crushing chamber. Contaminated air from these operations will be exhausted to Shaft #2.

The quantity of air entering the underground ore handling system will be controlled by a regulator and secondary fan. The regulator will be used to reduce the volume of air ventilating the underground ore handling system. The secondary fan will be used increase the airflow through the underground ore handling system if needed and ensure sufficient airflow to the underground ore handling system is always maintained.

16.7.4 Auxiliary Ventilation System

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. Active transverse stopes and some longitudinal retreat stopes will be equipped with ridged duct to capture contaminated air from the source (e.g., drilling) and the general heading. A ventilation fan will be drawing air from up to three stopes and delivering it to a return air raise. To minimize the potential for air recirculation, and therefore the buildup of

contaminate levels, mine level and sublevel ventilation flow rates will be controlled through the use of drift regulators and regulators inside the rigid ducting.

Air contaminated with radiation will not be reused or recycled in other parts of the mine.

Where the mine layout permits, flow-through ventilation will be used to ventilate infrastructure (e.g. sumps, maintenance shop) and some longitudinal retreat stopes during bolting drilling, mucking, and backfilling.

The ventilation on a given sublevel will be dynamic, requiring frequent modifications depending upon the location and nature of activities. The auxiliary ventilation system will be monitored and controlled through a ventilation on demand (VOD) system by the control room on surface and underground personnel. The operator of the system will oversee mine activity, radiation levels, airflow volumes and crucial parameters through a fully-integrated system.

16.8 Mine Services

16.8.1 Heating

Based on average monthly outside temperatures at the Project site, liquefied propane gas consumption and heating equipment capacity was estimated. The consumption was based on the mine air intake quantity of 330 m^3/s.

The coldest month in the area is January, which has an average low temperature of -32°C. Using a maximum temperature rise of 45°C, an estimated 50,000,000 BTU/hr heating system is required on the fresh air fans. A 50,000,000 BTU/hr heating capacity system would be capable of heating outside air as cold as -42°C up to +3°C.

16.8.2 Electrical

The main power supply will consist of an onsite captive liquefied natural gas (LNG) generator station with associated fuel storage, conversion, and transfer facilities. The underground distribution system will be at 13.8 kV. The total steady state load for the mine is estimated to be 9.43 MVA with a possible instantaneous maximum dynamic load of 16.5 MVA.

The mine power supplies will be provided in two phases:

- Phase 1: mine development; the mine development phase will include the sinking of two shafts; establishment of the surface sinking infrastructure facilities with all

power supplies derived from a temporary containerized generator station. The containerized generator station and power distribution and switchboards will be supplied and operated by the mining contractor

- Phase 2: mine production; the permanent energy supply will be provided by permanent LNG generators in an onsite generator station designed to cater for operational electrical loads of the process plant, mine operations, and surface facilities.

Services that will require power include: production operations; developments; the mine dewatering system; the underground tailings infill UGTMF; underground and surface mine ventilation systems; shafts and hoists; surface infrastructure including main ventilation; mine ore handling conveyors, crushers and vibros; and mine services and facilities, such as workshops, refuge stations, and lighting.

16.8.3 Communications and Instrumentation

Communications systems will be installed throughout the mine, and will be planned in a way that Arrow will transition to a mine that relies extensively on an advanced communications network, consisting of an automatic and emergency telephone, and a mine radio system.

Personnel and vehicle tracking will be incorporated into the system to permit the location of personnel, vehicles, and equipment underground. This system will improve safety in the mine, as well as productivity.

The mine control and monitoring system will be designed to allow specific items of the mine to be remotely controlled and monitored from a central control room at the surface of the mine. It will also allow environmental information collected throughout the mine to be displayed and monitored in the same surface control room. Environmental monitoring transducers will be positioned at strategic points throughout the mine. These transducers will connect either directly to remote outstations being used for equipment control, or may be connected into a transducer controller and transmitted back to a remote outstation.

Three refuge shelters will be employed underground and will be remotely monitored to safeguard the security of the shelters, and ensure that they remain ready for use.

Closed-circuit television (CCTV) will be employed around the mine surface and underground facilities which will involve a dedicated communications link between

cameras and monitors. Surface surveillance units may employ digital video recorders (DVRS), for security records and identification of persons and equipment.

Luminaires conforming to IP65 standard will be installed underground to provide general lighting at strategic locations.

16.8.4 Water Supply

The underground water supply will be provided from fresh water on surface, either from recycled water from the water retreatment plant, or water drawn from the surrounding environment.

The total underground water supply requirement is estimated to be approximately 46 m^3/hr, and is largely dependent on the level of mobile equipment operating in the mine. The water treatment plant will supply approximately 21 m^3/hr and the remainder will be fresh water.

16.9 Underground Mine Facilities

16.9.1 Material Handling

The ore handling system will begin with scoops loading muck in transverse and longitudinal retreat stopes. The scoops will tram muck to centrally-located ore and waste passes. Both the ore and waste passes have been located so that a tramming scoop has its loaded bucket downstream from the operator's cab. This ensures that dust from a loaded bucket will be carried away from the scoop operator. Each active level will be equipped with a scanner bay. Scoop operators will enter the scanner bay with a loaded bucket of material and through a simple signaling system the operator will be told if they are hauling ore or waste. The material will be dumped into the ore or waste pass adjacent to the scanner bay.

All levels and sublevels will be connected to the ore and waste pass except the 650 m level. Ore and waste from the 650 m level will be hauled by scoop to the 620 m level. The tramming distance is comparable to the longer tramming distances on other sublevels.

A ventilation raise will run parallel to the ore and waste passes. Fresh air will ventilate the ore pass, waste pass and sump by flowing past the dump points and directly into the return air raise which will be connected directly to the return air bypass drift on

500 m level. This configuration minimizes the contamination of clean air with airborne radiological contaminants from the ore passes and sump.

The bottom of the ore pass will be located on the 620 m level where a control system directs ore into the underground ore crushing system. Ore will be directed on to a grizzly equipped with a remotely-operated rock breaker. The underground crusher will reduce ore to <120 mm in size. The crusher has been sized for a throughput of 1,500 t/d, although it will be capable of substantially more than this, if required. Crushed ore will be loaded onto a conveyor and hauled to the shaft for skip loading.

The main waste pass will bypass the underground crusher, as this material will be exclusively development material. A control system will direct waste rock onto a separate grizzly and rock breaker. Broken and screened material will be loaded onto a conveyor and hauled to the shaft for skip loading. A second waste pass and underground crusher will be located at the UGTMF. At the UGTMF waste pass loadout, a separate grizzly, rock breaker, and crusher will reduce UGTMF waste to <120 mm for skip loading. UGTMF waste will be conveyed to the shaft where it will join the mine development waste.

Material conveyed to the shaft will be stored in two in-situ surge silos, one for ore and one for waste. Each surge silo will be sized to store 2,000 m^3 of material. A double flask loading pocket will be constructed just below the 620 m level to feed two 13 t skips. A loadout conveyor will feed the loading pocket measuring flask which will be volume- and weight-controlled for loading the skips. In lieu of a traditional shaft spill pocket, a shaft bottom ramp will be driven to allow a scoop to access the shaft bottom for clean-up. On surface, the ore will be dumped into bunkers where it will be held for material handling on surface.

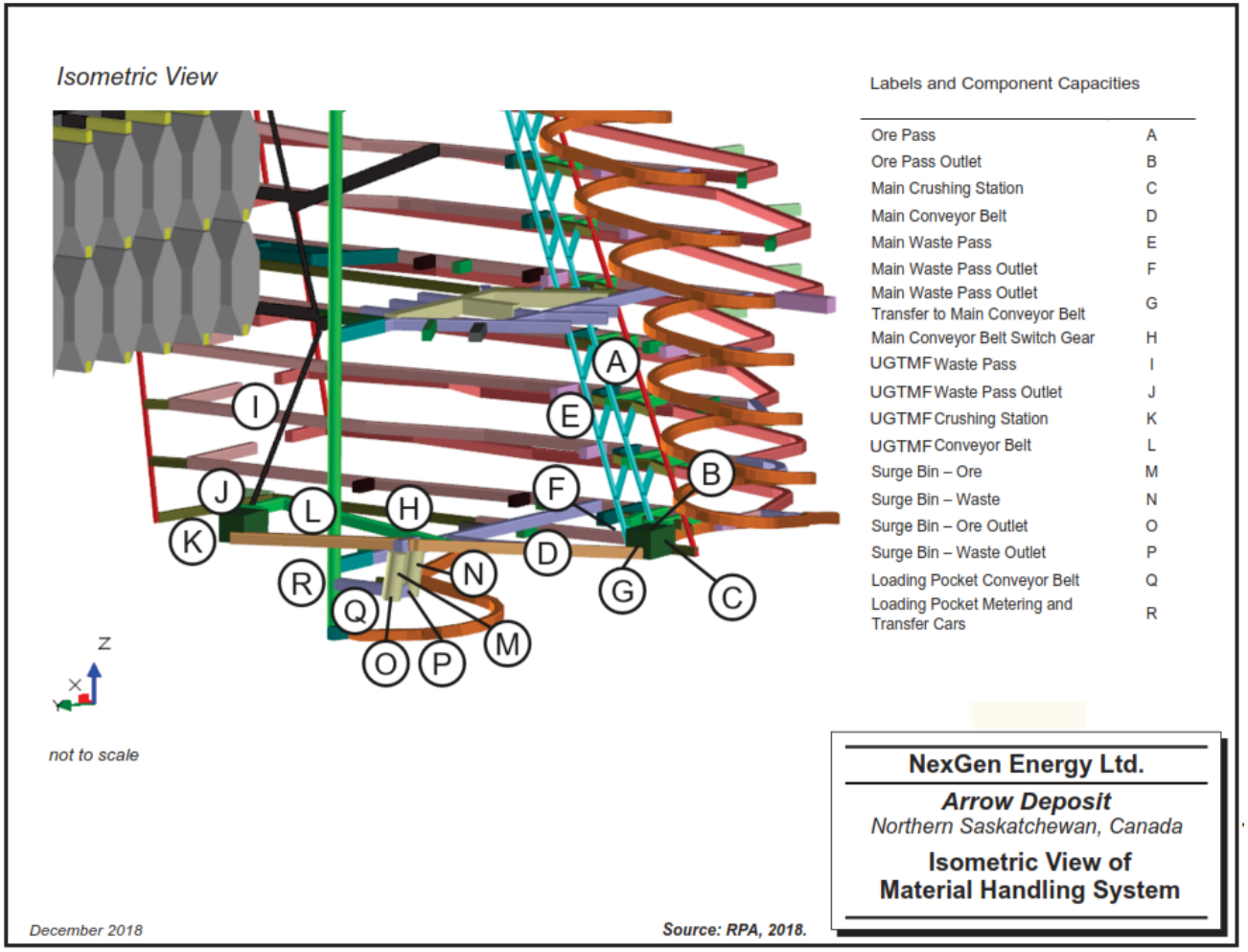
The delivery of paste fill and waste precipitates to the underground will be accomplished using three slick lines from surface to the 380 m level.

Figure 16-8 is a schematic layout showing the main material handling areas.

16.9.2 Maintenance and Fuel

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

Figure 16-8: Isometric View of Material Handling System



Note: Figure prepared by RPA, 2018.

The maintenance facility will be located on the 500 m level. It will consist of one service area, two repair areas, tool crib, warehouse, office, lubrication bay, electrical shop, welding bay, parking pads and wash bay. The maintenance facility was sized to accommodate the eight largest pieces of equipment at any one time.

Diesel fuel will be delivered from surface by underground fuel truck directly to both the underground equipment and a fuel bay to be located adjacent to the maintenance shop. Mining equipment that is less mobile (production drills and jumbos) will be re-fuelled and lubed in the workplace directly from a dedicated fuel/lube truck. Mining equipment that is more mobile (scoops, personnel carrier, etc.) will receive fuel from a self-contained fuel station in the fuel bay.

16.9.3 Explosive Magazines

Explosives storage for the Project will consist of three detonator/booster storage magazines and three explosives magazines located underground.

16.10 Surface Mine Facilities

Surface facilities required to support operations are discussed in Section 18.

16.11 Production Schedule

The target for planning production at Arrow was to achieve 30 M lbs of packaged U_3O_8 (i.e. after applying metallurgical recovery). To achieve the production target, the following constraints were applied:

- The process plant can handle a maximum of nominally 455 kt/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 m level area. For scheduling purposes, the mine was divided into two vertical mining blocks, the Upper Block, and Lower Block. Mining activities will commence from both the Upper Block and Lower Block, and in the A2 and A3 vein, for a total of four separate production areas. Having four separate production areas will provides operational flexibility for mine scheduling and sequencing. The daily ore production rate will range from 1,000–1,300 t/d, and will average to 1,100 t/d over the LOM.

In the area of transverse stopes, a primary and secondary stope system was used, to avoid leaving pillars. Primary stopes will be recovered first, followed by primary stopes on two vertical levels above, and secondary stopes on the original level. This sequence is primarily a result of geotechnical considerations.

The proposed mining schedule is included as Figure 16-9. Mining is planned to be conducted using two 12-hr shifts per day.

16.12 Mining Equipment

The Arrow mine 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; however, RPA has included operators for the equipment in the cost estimate. All the equipment listed will be supplied by a major mining equipment manufacturer, in new condition, via the underground mine contractor selected to complete lateral development.

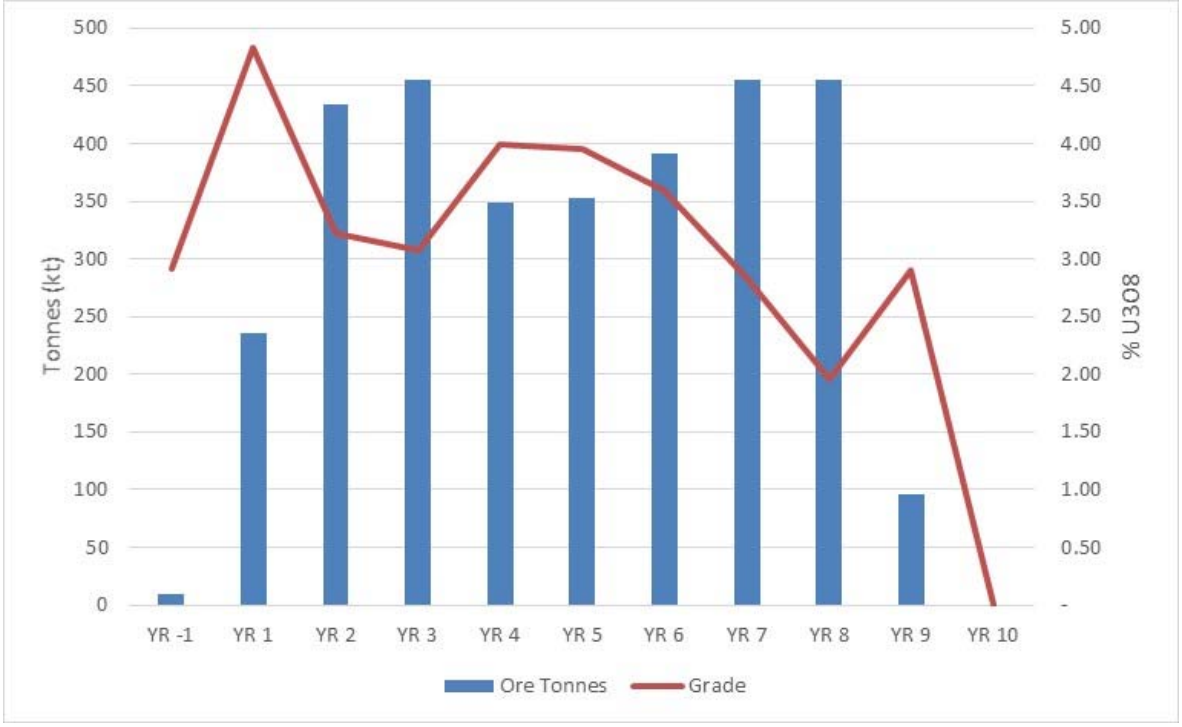
During initial construction, all levels will be captive from their respective shaft stations. Crews will begin development from Shaft #2 on the 380 m level, 440 m level, and 500 m level. Initially, crews will work to connect Shaft #1 and Shaft #2 on 380 m level and 500 m level. Following the connection of Shaft #1 and Shaft #2 on the primary levels, crews will work independently from to fully develop the underground mining infrastructure and development, allowing equipment free circulation throughout the mine.

During the captive development phase, each development area will have one of the following pieces of equipment:

- One two-boom jumbo
- One scooptram
- One bolter
- One scissor lift.

A raisebore machine will be used for development of ore and waste passes, and internal ventilation raises. The vertical development will not be in mineralized waste; therefore, the machine will not come into contact with mineralized material. It is assumed that the raisebore will be rented, and only remain on site during the pre-production period.

Figure 16-9: Planned Mining Schedule



Note: Figure prepared by RPA, 2018.

In total, there will be 44 pieces of equipment that will be operating in the mine during peak years. The mine will not use any haul trucks, as a series of ore and waste passes have been established for the movement of material, which will minimize scoop tramming distances. Once the mine goes into production, stopes will be backfilled with paste fill pumped to open stopes through pipes hung from the drift back. The use of remote-controlled equipment is assumed, so as to reduce radiation exposures.

Table 16-8 provides the list of equipment required at peak. Equipment requirements will vary over the LOM.

Equipment operating hours have been estimated based on first principles. Equipment that is specific to development or production fluctuates with changes to annual throughput, while some equipment is assumed to be operating at a certain rate regardless of how many tonnes or metres are being developed.

Table 16-8: Mobile Equipment Fleet

Description	Company	Model Number	Qty
Two Boom Jumbo	Epiroc	Boomer M2	4
Rock Bolter	Epiroc	Boltec M	3
Dev. Scoop	Epiroc	ST18	2
Pro. Scoop	Epiroc	ST18	3
Production Drill	Epiroc	Simba E7C	3
Cable Bolter	Epiroc	Cabletec M	2
ANFO Loader Truck	Normet	Charmec MF 605	3
Slot Drill	Epiroc	Easer L	1
Utility Scoop	Epiroc	LH203	1
Lube Truck	Normet	MF 400	1
Flat Deck Truck w. Crane	Normet	LF 130	1
Transmixer	Normet	Ultimec LF 600	2
Shotcrete Sprayer	Normet	Spraymec MF 050 D	2
Personnel Carrier	—	—	3
Scissor Lift	Normet	MF 540	3
Small Vehicle (Rad. Tech., etc.)	Kubota	RTV-X1100C	6
Grader	—	—	1
Mobile Rock Breaker and Scaler	—	—	1
Cassette Truck	Normet	MF 100 Multimec	1
Basket Truck	Normet	MF 905 Himec	1
Total			***44***

Notes to accompany Mobile Equipment Fleet table:

Manufacturers and models listed are examples of units appropriate for the required tasks, however, do not represent commitment to purchase those particular units.

16.13 Comments on Section 16

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 underground and active reclamation of the mine.

17.0 RECOVERY METHODS

17.1 Process Flow Sheet

A zero-based design approach was taken in the mill process design. The design aims to achieve the required throughput with the minimum redundancy, installed equipment and design allowances. Health, safety, and environmental aspects however are not compromised. There are only two instances where circuit design capacity is planned to be more than nominal. Grinding capacity has been increased by 20% more than nominal to allow for higher than the other circuit, maintenance requirements. The effluent treatment plant has also been designed for a more than nominal flow rate due to the possibility of having mine leakage as well as weather-related surges in effluent treatment requirements that must be considered.

Process design has been directed by the metallurgical test program results as well as knowledge that is from literature and Wood's experience with existing successful process methods.

The proposed process block diagram is included as 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

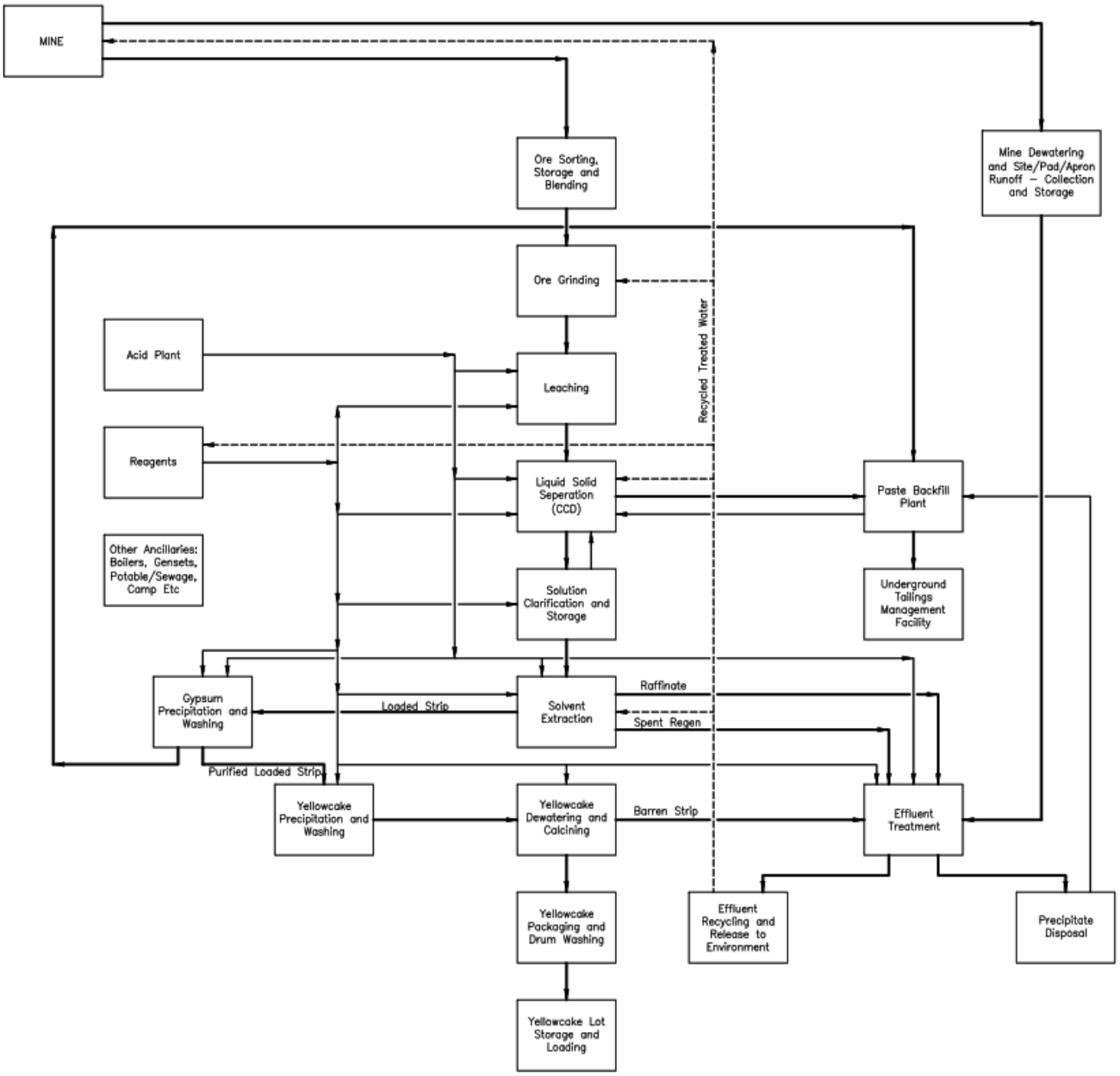
17.2.1 Ore Sorting and Storage

Ore will be crushed underground and will be suitably sized to feed the grinding mill. The hoisted ore will be loaded into an ore truck at the headframe. The truck will drive through a radiometric scanner to confirm ore grade and the delivery location of the ore on 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 semi-autogenous grind (SAG) mill at a prescribed rate that will be close to the ground ore feed rate to be fed to mill leaching. The ore will be weighed on the belt as well as given a gamma radiation scan to check uranium content.

Figure 17-1: Process Block Flow Diagram



Note: Figure prepared by Wood, 2018.

Table 17-1: Production Design Requirements

Production Criteria	Units	Quantity
Ore feed rate	t/a	456,300
	t/op day	1,300
Ore feed grade	% U_3O_8	3.09
Maximum feed grade to mill	% U_3O_8	5.0
Plant uranium recovery	%	97.64
Production rate	lb U_3O_8/a	30,000,000
Operating time	hr/a	8000
Availability	%	95

Water will be added into the SAG mill feed as well as the ore to provide the target % solids content in the mill. SAG mill discharge will report by gravity to feed the ball mill. The ball mill will also be fed recycled oversize particles from a classification cyclone that will be situated above the ball mill. Water can be added to the ball mill feed and or to the ball mill discharge to maintain the target % solids composition of slurries in the circuit. Ball mill discharge will report 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 (100% passing 300 µm) as well as the target 50% solids composition and will report by gravity to a ground ore storage tank. The ore storage tank will be mechanically agitated and will provide surge capacity between the grinding circuit and the leaching circuit and a degree of ore blending. The grinding circuit will have tonnage capacity over that required for leach feed. This will allow the grinding circuit to fill the ore storage tank. When full, the grinding circuit can be shut down to provide short periods (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 to feed the prescribed solids rate to the leaching circuit. The leaching circuit will consist of six mechanically-agitated tanks that will be connected in series. The flow between each tank will be by gravity. The discharge of each tank will be from a baffled upcomer to ensure minimal solids short circuiting in each tank. The tanks in total will provide the target 10-hour residence time to oxidize and dissolve the uranium.

The tanks will be heated with steam spargers to 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 25 to 35 g/L acid content in the leaching tank discharge while the peroxide will be added to maintain the target >450 meV oxidation reduction potential (ORP).

If there are times when low iron ore is being leached, ferric sulphate will be available to provide supplemental ferric iron which will be required to provide faster oxidation of U^{+4} to U^{+6}. (U^{+6} is soluble in the acidic solution while U^{+4} is not).

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 relatively small mix tank. Overflow solution from the counter-current decant (CCD) 2 will report by gravity into this mix tank as well. The mixed slurry will be pumped to the center well feed of CCD 1 along with a flocculant flow (flocculant enhances settling). The overflow of CCD 1 (pregnant aqueous solution) will report to a pumpbox and will be pumped to feed a pin bed clarifier. Underflow from CCD 1 will be removed by a variable speed pump that will be controlled by the density of the underflow stream as well as the solids load level in the thickener. Underflow will be pumped to a small mix tank where it will be mixed with the gravity overflow of CCD 3. This mixed solution will be pumped to the feed well of CCD 2 where it will be treated with flocculant. In a similar manner, 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 tails centrifuge feed tank in the paste backfill plant.

Wash water will be fed into the feed mix tank of CCD 6. The wash water will be made up of:

- Priority 1: tails centrifuge centrate (if un neutralized)
- Priority 2: a percentage of solvent extraction (SX) raffinate flow (recycle as much acid as possible)
- Priority 3: acidized fresh water.

If tails centrifuge centrate has been neutralized (see paste plant description), raffinate flow will becomes the priority and tails centrate the second. 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 introduced into this tank to meet the target uranium content in the pregnant aqueous solution that will be overflowing from CCD 1.

The overflow of CCD 6 will flow by gravity to the mix tank feeding CCD 5. The solutions will pass from one thickener to the next in the opposite direction as the solids slurry, i.e. from CCD 6 to feed CCD 5 etc. Circuit performance will be based on a combination of the amount of uranium in the feed solution, the amount of wash water to be added to CCD 6 feed and 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 by gravity to the SX feed tank. The feed tank will have capacity to feed the SX circuit for about two 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 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 (usually about 6–12% amine reagent by volume).
- Isodecanol that will be introduced into the solution to enhance the separation of aqueous and organic after mixing ceases. Isodecanol is typically added to about 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 1 mixer where it will be mixed

with organic solution from extraction 2 settler. As the organic and the aqueous are intimately mixed, the tertiary amine in the organic will hold onto the uranyl sulphate and remove it from the aqueous. After mixing, the mixer will discharge the solutions into a settler unit where the solution will separate into the lighter organic floating on top of the heavier aqueous. Some of the organic in an extraction settler will be returned to its mixer. Therefore, in the mixer, the ratios between organic and aqueous can be controlled by recirculating organic.

The aqueous that has settled out from extraction 1 will be fed to extraction mixer 2 where it will be met with a counter-currently moving organic flow from extraction settler 3. In this counter current flow, pregnant aqueous will be fed into mixer 1 and will discharge as barren raffinate from settler 4. Conversely, barren organic will be fed into mixer 4 and will discharge from settler 1 as loaded organic (high uranium content organic). Barren raffinate from settler 4 will be pumped to the raffinate tank. Periodically, the organic that accumulates on the raffinate tank surface will be skimmed off to return to the extraction circuit. Much of the raffinate will report to the CCD 6 feed tank where it will be recycled. As much of the raffinate as possible will be recycled to capture the acid that is contained in the raffinate. Recycling of raffinate will, however, increase the circulating load of contaminant elements. This elevation of contaminant levels will result in the need to bleed some of the raffinate to the effluent treatment circuit. The raffinate tank will be able to hold about two hours of raffinate generation.

Loaded organic at this point 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 flow of acidic water that will be flowing counter-current to the organic. The acid solution will wash some elements such as arsenic from the loaded organic as well as washing any small "bubbles" of aqueous that may have escaped extraction 1 with the loaded organic. In both of the two acid wash mixer settlers, aqueous will be recirculated to obtain the target organic to aqueous ratio in the mixers. Scrubbed loaded organic will have a high concentration of uranium and much lower concentrations of contaminating elements. Some elements such as molybdenum can, however, go with the uranium into the organic flow to some 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 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 four hours of loaded strip as it is generated. The loaded strip will be very 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. Once again 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-currently 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 will be used to strip the loaded organic.

Most of the washed organic will report to the barren organic tank. A portion, however, 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 a pH of about 9. This will strip the barren organic of elements such as 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 (from incomplete strip performance), much of it will be lost to the spent regeneration solution. The spent regeneration solution will report to the effluent treatment circuit. It is expected that the stripping of the loaded organic into the loaded strip will be 99.6% efficient (U lost to the regeneration stream is about 0.4%).

17.2.7 Gypsum Precipitation and Washing

Lime will be added to increase the pH of the loaded strip solution to remove the acid in the strip solution in preparation to precipitating uranium. Loaded strip solution will be pumped from the loaded strip tank into the first reactor tank of a train of seven tanks. The flow will report from one tank to the other by gravity. Lime slurry will be added to

each tank to very gradually bring pH up in small steps to a final 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 four hours.

Gypsum solids will be present at a relatively high percentage as it will discharge the last reactor tank 7 to the gypsum washing circuit. Before feeding into gypsum centrifuge 1, the gypsum reactor discharge will be diluted with centrate from gypsum centrifuge 2. Gypsum centrifuge 1 will spin off the loaded strip solution and provide cake at an estimated 60% solids. The gypsum centrifuge 1 centrate will report to the gypsum clarifier. Flocculant will be added to the clarifier feed to assist with solids settling. The clarifier will settle the small quantity of centrate solids and provide a clear low suspended solids feed (purified loaded strip solution) for feed to the next circuit.

The small amount of solids from the gypsum clarifier underflow will report in a periodic flow to the gypsum centrifuge 1 feed along with the gypsum reactor 7 feed flow and the gypsum centrifuge 2 centrate. If the flow of gypsum is low from the gypsum precipitation tanks, some or all of the reactor 7 discharge can report to the clarifier to thicken up the slurry before feeding gypsum centrifuge 1. The gypsum cake from centrifuge 1 will be dropped into mechanically agitated discharge tank 1. Centrate from gypsum centrifuge 3 will be added into this tank as well. Solids will be washed with the centrate and fed as a slurry to gypsum centrifuge 2. Gypsum centrifuge 2 centrate will return to feed centrifuge 1. The cake of centrifuge 2 will drop into discharge tank 2 and will be slurried with a small amount of acidized water. The pH in the circuit must be maintained at 3 to ensure no uranium precipitation. The tank slurry will then be pumped to feed gypsum centrifuge 3.

Fresh water will be added onto the beach area of the solid bowl centrifuge 3 to wash the gypsum one last time before it will be discharged as cake. The cake from gypsum centrifuge 3 will report to a conveyor that routes the washed gypsum to the paste plant. Centrate from gypsum centrifuge 3 will report to centrifuge discharge tank 1 to slurry centrifuge 1 cake and to feed centrifuge 2.

17.2.8 Yellowcake Precipitation and Washing

Purified loaded strip from the gypsum clarifier overflow will report to yellowcake (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 the pH at 2.8 to 3.5. Tank 1 will flow by gravity to YC precipitation tank 2. In tank 2, the reaction will be completed and time will be given for the precipitate particles to grow. Total residence time in the YC precipitation tanks will be four hours.

YC precipitation tank 2 will overflow into YC wash tank 1 where the yellowcake slurry will be mixed with the overflow of YC wash thickener 2. Wash tank 1 discharge will be pumped to YC was thickener 1 where it will be treated with flocculant to assist good settling in the thickener. The thickened underflow of YC wash thickener 1 (50 % solids), will be pumped to YC wash tank 2 where it will be mixed with calciner scrubber discharge water, and centrate from the YC centrifuge. YC wash tank 2 will be pumped to feed YC wash thickener 2. Flocculant will be added to the thickener feed to assist good settling. Overflow will report to the YC wash tank 1. The underflow of the wash thickener 2, which will be 50% solids, will be mixed with fresh water in an inline mixer. The slurry will then be fed to the YC centrifuge. In the solid bowl of the YC centrifuge, water can be added to wash the yellowcake further. The cake will discharge the centrifuge 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 solids settling performance will be required in YC wash thickener 1 to ensure that yellowcake solids do not escape with the barren strip to feed the effluent treatment system and result in uranium recovery loss. The barren strip tank can contain about four hours of barren strip generation.

17.2.9 Yellowcake Drying/Calcining and Packaging

Yellowcake from the YC centrifuge will be fed to the YC calciner by the YC conveyor. 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 yellowcake, a small amount of volatile contaminants will also be driven out of the calcine (e.g. fluorine).

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 ID 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 Tailing Neutralization and Paste Plant

CCD thickener 6 underflow slurry will be pumped to the paste plant and into the tails centrifuge feed tank. The feed tank can contain about two hours of CCD 6 slurry. The solution in the slurry will have 12 to 15 g/L acid as well a small concentration of dissolved uranium. The solution could be neutralized in the tails centrifuge feed tank in which case the acid will be destroyed to produce gypsum and uranium will be precipitated and report to the paste. However, acid and some uranium from the slurry can be also returned to the CCD circuit if tails are neutralized after centrifuging by the following process:

- Un-neutralized residue slurry from the feed tank will feed the tails centrifuge.
- Water will be added onto the centrifuge cake beach to wash as much acid to the centrate as possible.

Discharge cake from the tails centrifuge will be 77–85% solids as it reports to a mixing screw conveyor. Slaked lime (calcium hydroxide) will be mixed into the cake in

proportions to render the cake basic. Binder will then be introduced into the cake as it is fed into the paste mixer. The concentration of the binder added is dependent on the type of backfill required by the mine operations. Different requirements need different backfill strengths. Water will also be added into the paste mixer to ensure that the mixed paste has the prescribed moisture content (67% solids in one metallurgical test).

Discharge from the paste mixer will report to a positive displacement concrete-type pump. The pump will pump the paste through a pipe system that reports to the underground tails management facility.

17.2.11 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. 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 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 (ORP is a measure of the cleanliness of the water and its ability to break down contaminants) in the first-stage water treatment to keep arsenic in an arsenate form. This will make arsenic precipitation more efficient. Sometimes, however, it will be beneficial to inject air into the reactors to provide oxygen to the system to ensure elements do not reduce. Air will be injected into the agitator blade area, will help to remove 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 will react with the sulphate that is plentiful in the first-stage water treatment to form barium sulfate. The radium in the effluents will act similarly to barium 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 two hours at nominal flow. All of the reagents can be added into either the first and 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 low suspended solids 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 a feed tank for the first-stage water treatment pressure filter. The pressure filter will consolidate the precipitate into a cake and will discharge it onto a conveyor that reports to the paste plant. The filtrate from the filter will be pumped back to the first-stage water treatment clarifier feed.

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, metals will be precipitated. More ferric sulphate as well as barium chloride will be added to precipitate more of the oxyanions as well as radium. If more sulphate is required, or pH needs to be tuned down, sulphuric acid will be available. 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 one hour. At the nominal flow rate residence time will be two hours. Precipitated solids will be removed from the second-stage water treatment clarifier as the first-stage water treatment clarifier to feed the second-stage water treatment pressure filter. The solid cake will also be collected by a conveyor and will be fed to the same drop pipe as the first-stage water treatment system. The second-stage water treatment pressure filter filtrate will be pumped back to feed the second-stage water treatment clarifier. The second-stage water treatment clarifier will overflow into the pH adjustment tank. Dilute sulphuric acid will be added to bring the effluent to pH 6.5 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 and mill process 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 an adequate source of treated water for recycling.

17.2.12 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 normal mine water discharge. As water is retained in the pond, suspended solids will settle. Water that runs off from potentially-contaminated site areas such as the ore storage pads and from potentially-contaminating uses such as dry and laundry and maintenance shops, will discharge to the feed settling pond.

A contingency pond will be available for accommodating pond cleaning and maintenance procedures 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 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.

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 all of the parameters of concern will be assayed. If all of the assays are in the required ranges, approval will be given for the pond to be discharged into the environment. As the pond is discharged to the environment, another composite sample will be taken that will represent the discharge to the environment. The assays of this composite sample will be reported as required to the control agencies. If assays are not as required 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 recovery is forecast to be 97.6% (Table 17-2).

Table 17-2: Uranium Recovery from Plant

Circuit	Recovery (%)
Leach extraction	99.3
CCD washing efficiency	99.5
SX extraction efficiency	99.9
Stripping/regeneration efficiency	99.6
Gypsum precipitation efficiency	99.77
Gypsum washing efficiency	99.75
Yellowcake precipitation efficiency	99.9
Barren strip, calciner, packaging	99.9
Overall Mill % U Recovery	***97.6***

17.4 Energy, Water, and Process Materials Requirements

17.4.1 Water

Water consumption was estimated for the different mill processes, and totals about 152.9 m^3/hr. Opportunities to recycle water to the mill and reduce fresh water consumption were identified, totalling approximately 67 m^3/hr.

17.4.2 Reagents

Reagents will include:

- Sulphur
- Sulphuric acid (94% H_2SO_4)
- Unslaked lime (CaO)
- Hydrogen peroxide (H_2O_2)
- Flocculant
- Kerosene
- Tertiary amine
- Isodecanol

- Sodium carbonate (Na_2CO_3)
- Magnesia (MgO)
- Ferric sulphate ($Fe_2(SO_4)_3$).

17.4.3 Energy

Energy requirements for the Project are discussed in Section 18.9. The process plant is estimated to require 7 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

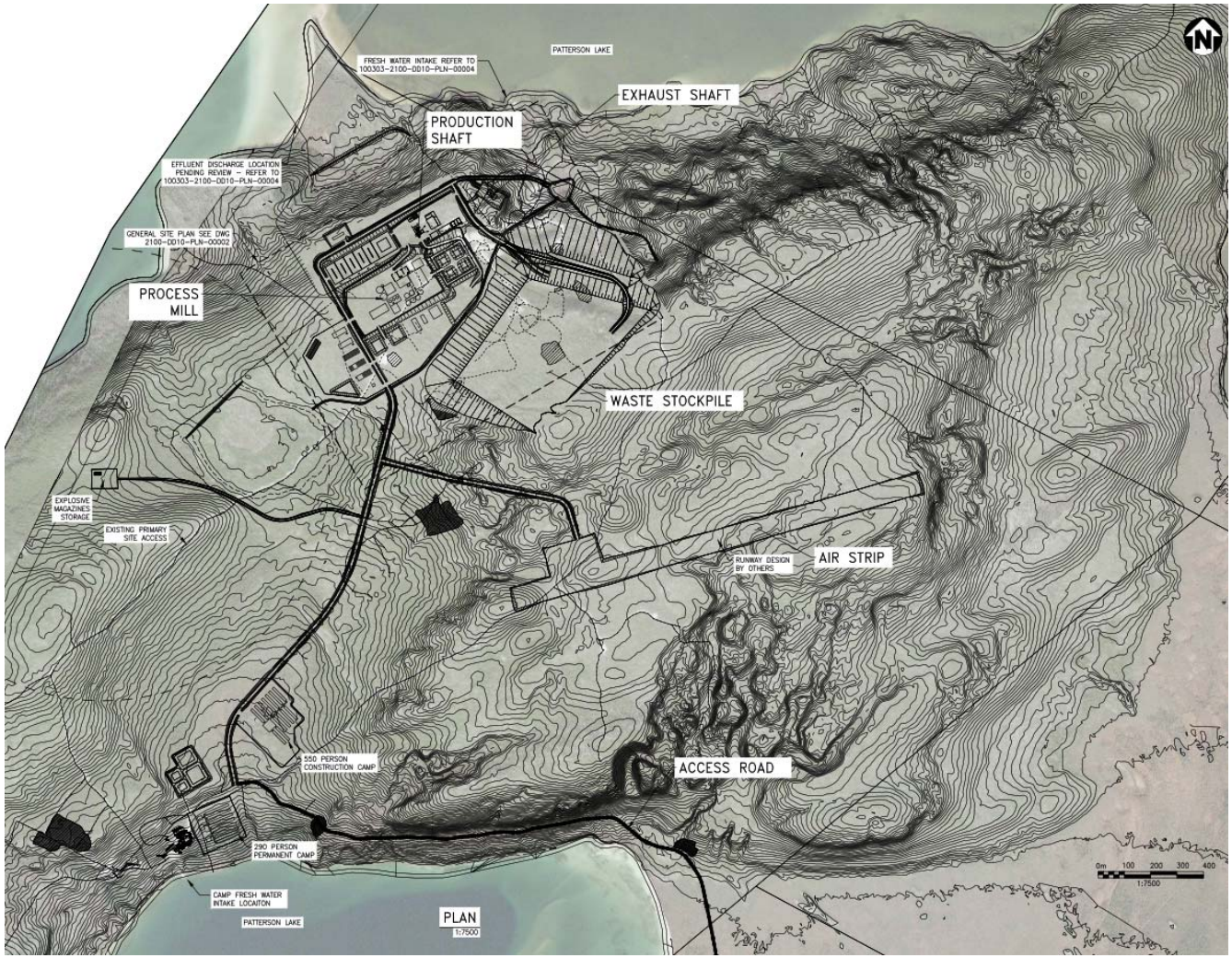
18.1 Introduction

The key infrastructure contemplated for the Project includes:

- Underground mine with two shafts
- Underground infrastructure including: material handling systems, maintenance facilities, fuel bay, explosives magazine, ventilation, paste backfill, electrical and communications facilities, underground water supply, dewatering facilities
- Underground tailings management facility
- Surface support infrastructure for the mine, including: headframes and hoist facilities, surface magazine, liquid natural gas (LNG) facilities, power plant, and ventilation fans
- Process plant and associated analytical laboratory
- Surface waste rock storage facility for clean, special waste and mineralized waste.
- Permanent and construction accommodation camps
- Mine support buildings, including permanent accommodations, maintenance, warehouse and security buildings
- Water management facilities, including storm water runoff pond and six process ponds
- Airstrip.

The layout of the planned surface infrastructure is provided in Figure 18-1. Underground mine layouts were discussed in Section 16.

Figure 18-1: Proposed Surface Infrastructure Layout Plan



Note: Figure prepared by Wood 2018.

18.2 Roads and Logistics

The access road to site from Hwy 955 began construction in 2016 and was completed in 2017 with improvements to the surface composition. The road alignment varies to best fit the existing land to avoid steep slopes and excessive embankment fills. As a result, there are sharp turns and often blind spots that will create hazards for new traffic flows to and from the Project. Clearing is anticipated for some corners to minimize safety concerns. The road may need to be sprayed with water trucks during dry conditions to avoid rutting, dust and surface rework.

The road plan on site is designed to allow for intermittent closures while maintaining access to all major infrastructure. Currently there is an access trail from the existing camp site to the future plant site which accommodates exploration activities. This road may be used to mobilize the site preparation contractor to site, however it will not be considered for use in the final design. Regular traffic roads will have a 6 m wide traffic surface (3.0 m per lane) with 1.0 m wide shoulders totalling 8 m wide top gravelled surface. The road from the camp to site will be constructed from a new alignment to these specifications.

18.3 Stockpiles

No long-term stockpiles are envisaged in the mine plan.

18.4 Waste Storage Facilities

The planned clean waste rock facility is discussed in Section 20. Any radiologically contaminated waste will be disposed of in the underground workings.

18.5 Tailings Storage Facilities

No surface tailings management facility is included in the 2018 Pre-feasibility Study design. The Project plan is to store all processed waste underground in an underground tailings management facility (see Section 16.5).

18.6 Water Management

Water management is discussed in Section 20.

18.7 Built Infrastructure

18.7.1 Mine Island

The term "mine island" is used to refer to the prepared area surrounding the production shaft that will house the majority of the surface infrastructure required to support underground mining. The surface of the pad will be approximately 150 m by 65 m.

18.7.2 Administration Building, Laboratories, and Dry

The administration building will be a two-story building located at the northeast corner of the process plant, and will include:

- Offices and cubicles for the staff
- Safety and first aid facilities, including an ambulance and fire-truck parking bay
- Change rooms, lockers, showers, and laundry for the plant and underground workers
- Metallurgical laboratory for analyzing effluent, ore, and product samples

It is anticipated that there will be office and cubical space for approximately 32 people in the administration building. There will also be two meeting rooms, kitchen, and a lunch room for staff to use. The female and male dry areas are sized for 37 and 174 workers respectively based on other similar underground uranium projects. The overall administration complex is projected to cover 37 m x 32 m, and will be separated from the mill complex by a firewall.

18.7.3 Maintenance and Warehouse Building

The Project's maintenance and warehouse storage will be in the one building that will be located to the west of the acid plant. The maintenance shop will occupy the western half of the 30 m × 70 m 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. Two drive-through maintenance bays will be located on the west end of the building, one with a wash bay and one with a 10 t overhead crane. Allowances were made to equip the shop appropriately with items such as welders, band saws, and other small tools. This building will also house additional office space and lunch room facilities for warehouse and maintenance personnel.

The warehouse side of the building will stock supplies and equipment that are required for the ongoing plant operation and maintenance. The warehouse will include a truck

receiving platform and a 7.5 t overhead crane. Larger spare parts that are not susceptible to freezing will be stored in either fenced warehouse storage or the coverall-style cold storage building that will be located nearby.

18.7.4 Site Security

The site gatehouse will be a 12 ft x 60 ft modular building, and will include washroom facilities and water storage for security personnel. Gate arms will be used to control site access.

18.7.5 Fire Protection

A standard deep buried interconnected firewater loop will be installed, and will encircle the process plant and the production shaft. Fire is an inherent risk in solvent extraction plants that must be managed. The solvent extraction plant will have it's own specially designed fire suppression system.

18.8 Camps and Accommodation

The permanent camp is assumed to host about 290 rooms in seven dormitories that will be oriented perpendicular to the topographic slope. The camp administration office and boot room will form the entrance to the camp. Additional facilities will include laundries, recreation room, and camp kitchen and dining area.

The construction camp will be located in proximity to the permanent camp in order to minimize infrastructure such as piping. It is planned to accommodate approximately 550 persons in 13 dormitories.

18.9 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 is approximately 200 km away. Due to the high capital costs associated with running a power line to site, the 2018 Pre-feasibility Study design includes on on-site power plant.

The power requirement for site is estimated to be 14 MW (Table 18-1). In order to meet the site power requirement with a N+2 design requires nine generators. The plant will be fuelled by LNG which will be trucked to site.

Table 18-1: Power Requirements Projection

Description	Power Consumption (MW)
Mine	7.0
Process	6.3
G&A	0.7
Total	***14.1***

Note: G&A = general and administrative

18.10 Water Supply

Water from Patterson Lake will be pumped to a storage tank in the headframe building for the production shaft. From there the water will be pumped to the effluent treatment plant where it will be treated as fresh water. This will be piped to the administration building for the workers dry and laboratory and for fresh water to the plant.

A separate water intake will be located south of the existing camp and will supply a separate potable water system for the camp facilities. The treated potable water will be piped to both the construction camp and the permanent camp.

18.11 Comments on Section 18

Infrastructure considerations outlined are sufficient to support the proposed mine plan.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

19.1.1 Overview

This general overview of the uranium industry is abstracted from the World Nuclear Association website, and has not been independently verified by the QP.

Production from world uranium mines now supplies 90% of the requirements of power utilities. Primary production from mines is supplemented by secondary supplies, formerly chiefly from ex-military material but also from the products of recycling and stockpiles built up in times of reduced demand. World mine production has expanded significantly since about 2005.

Over two-thirds of the world's production of uranium from mines is from Kazakhstan, Canada and Australia. An increasing amount of uranium, now 50%, is produced by in situ leaching. After a decade of falling mine production to 1993, output of uranium has generally risen since then and now meets almost all the demand for power generation.

The World Nuclear Association notes that:

"About 445 reactors with combined capacity of over 390 GWe, require some 75,000 tonnes of uranium oxide concentrate containing 63,000 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 – over the 20 years from 1970 there was a 25% reduction in uranium demand per kWh output in Europe due to such improvements, which continue today. Each GWe of increased new capacity will require about 150 tU/yr of extra mine production routinely, and about 300-450 tU for the first fuel load".

"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 PV sources."

"The perception of imminent scarcity drove the spot price for uncontracted sales to over US$ 100 per pound U_3O_8 in 2007 but it has settled back to under $20 over the four years to mid-2017. Most uranium however is supplied under long-term contracts and the prices in new contracts have, in the past, reflected a premium of at lease $10/lb above the spot market".

The website notes that price fluctuations are common; however, the price does not indefinitely stay below the cost of production, nor does it remain at very high levels for longer than it takes for new producers to enter the market and supply-side concerns to subside (Figure 19-1).

19.1.2 Current Market Activity

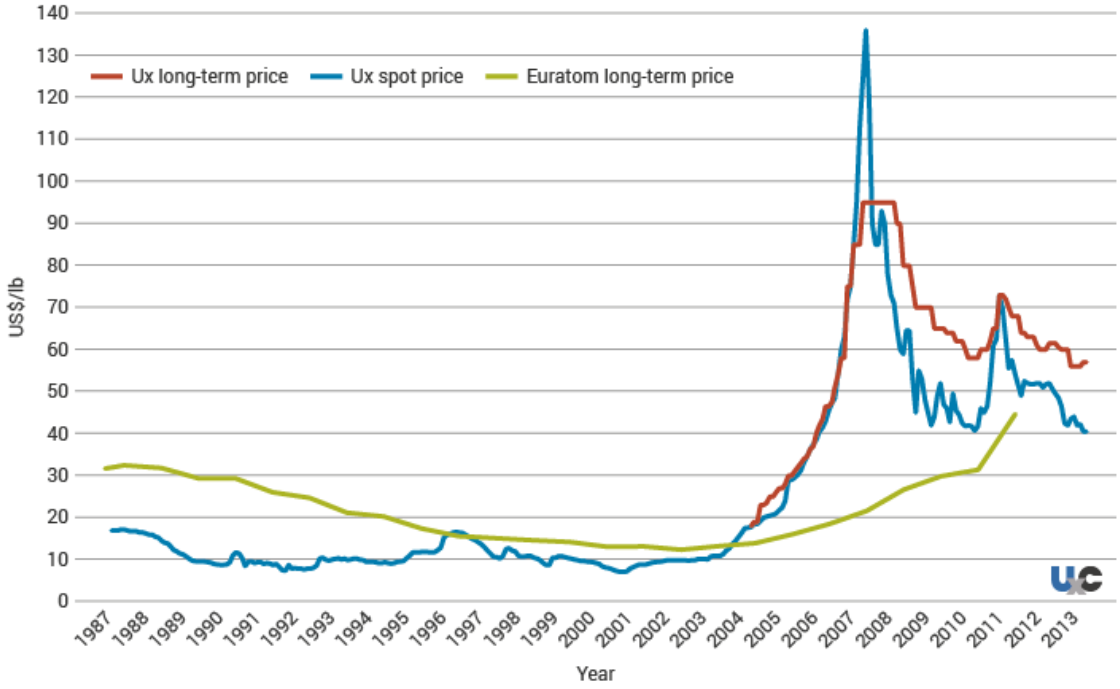
Information in this sub-section is abstracted from UxC LLC (UxC), 2018.

A total of 272 spot transactions were reported in the period from the second quarter to the third week of August, 2018, involving 56.4 Mlb U_3O_8 equivalent. A portion of the increase can be attributed to a new investment fund, Yellow Cake plc, entering the market and buying over 8 Mlb U_3O_8 equivalent in a single transaction. Cameco production and purchase announcements also contributed to volume activity.

Overall, spot uranium pricing increased during the first half of 2018 to over US$22/lb, further increasing to US$26/lb by August, 2018. This upward pricing resulted in three- and five-year forward prices lifting to the US$30–$35 range, and the term market responded with offers for base-escalated prices reflecting their first increase since 2017 (Figure 19-2).

With the addition of a larger reprocessed uranium contract award, the term volume for 2018 (to the third week of August) now totals close to 60 Mlb U_3O_8 equivalent. Utility buying over the third quarter of 2018 remained a mix of both mid-term and longer-term needs.

Figure 19-1: Uranium Pricing 1987–2013



Note: Figure from World Nuclear Association website, prepared by UxC. The Euratom long-term price is the average price of uranium delivered into the European Union that year under long term contracts. It is not the price at which long-term contracts were written in that year.

Figure 19-2: Monthly Spot Volume vs. Price

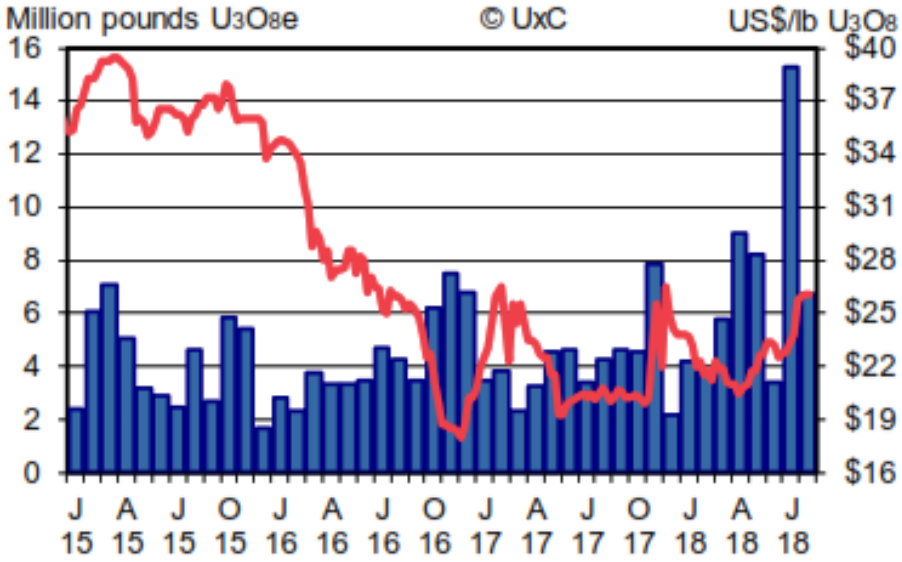


Figure prepared by UxC, 2018. Note: months cited are January, April, July and August.

Since the second quarter of 2018, six new contract awards were added to the database involving 40.5 Mlb U_3O_8 equivalent, bringing annual contracting volume to 56.6 Mlb U_3O_8 equivalent under 18 contract awards. Estimates for potential demand over the next 12 months (August 2018 to August 2019) total about 36.3 Mlb U_3O_8 equivalent. Figure 19-3 shows the utility long-term contract (LTC) volumes from 1990 to 2017, and the 2018 projection.

The estimate of uncovered requirements declined by a cumulative 132 Mlb U_3O_8 equivalent over the 2018–2035 period (Figure 19-4). Given the 2–3 year forward lead-time for term coverage, the uncovered demand curve indicates that term contracting could remain limited over the next 12 to 24 months (August 2019 to August 2020). For 2018, UxC expects world uranium production to decline by 19 Mlb to 136 Mlb U_3O_8 equivalent, down 12% from 155 Mlb U_3O_8 equivalent in 2017.

Combined with secondary supplies, aggregate supply in 2018 is expected to total 184 Mlb, which is 7 Mlb lower than the UxC requirements model (URM) base case demand prediction of 191 Mlb U_3O_8 equivalent.

19.1.3 Term Contracts

Information in this sub-section is abstracted from UxC, 2018.

Term contracts are an integral part of uranium market transactions, some of which use base-escalated pricing mechanisms, while others are tied to market-related terms that index spot indicators. While the long-term base price is generally set at a risk premium (which is the amount that a buyer is willing to pay to lock in the future prices) to the spot price, these two indicators have historically exhibited a very close relationship. The gap between these two prices is expected narrow over time as both prices are likely to react to changes in market fundamentals via different buying and selling activities.

Unlike supplies in the spot and mid-term markets, which are mainly driven by available inventories (including secondary supplies of uranium), long-term contracts are typically offered by uranium producers who can commit supplies for multiple years in the future. Thus, the long-term base price provides an indicator of future supply availability, and is linked to production costs. To the extent that producers can sell their excess uranium production directly into the spot market, spot prices are also partially linked to production costs. Therefore, while both spot and term price indicators serve as inputs in making initial production decisions, these prices are likely to place a cap on production costs.

Figure 19-3: Utility LTC Volume

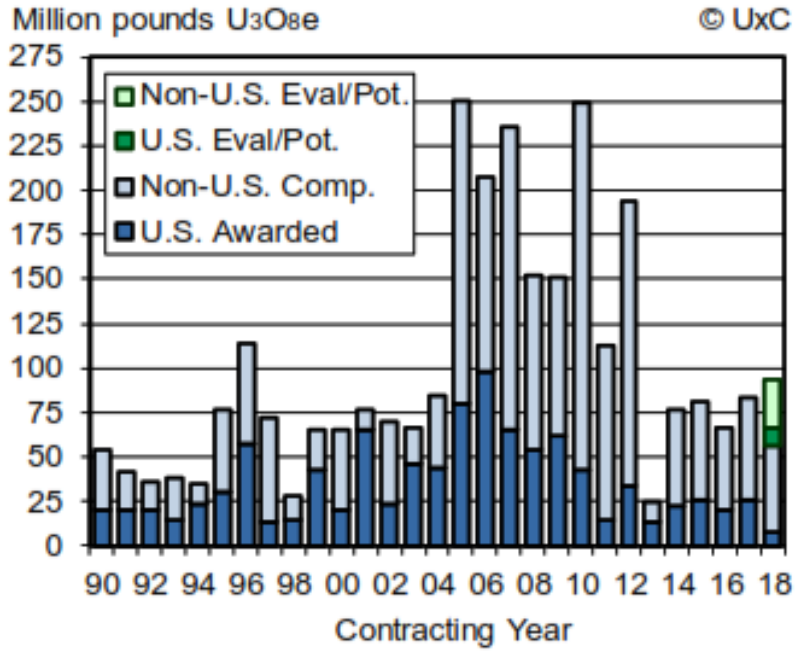


Figure prepared by UxC, 2018.

Figure 19-4: Uncovered Requirements

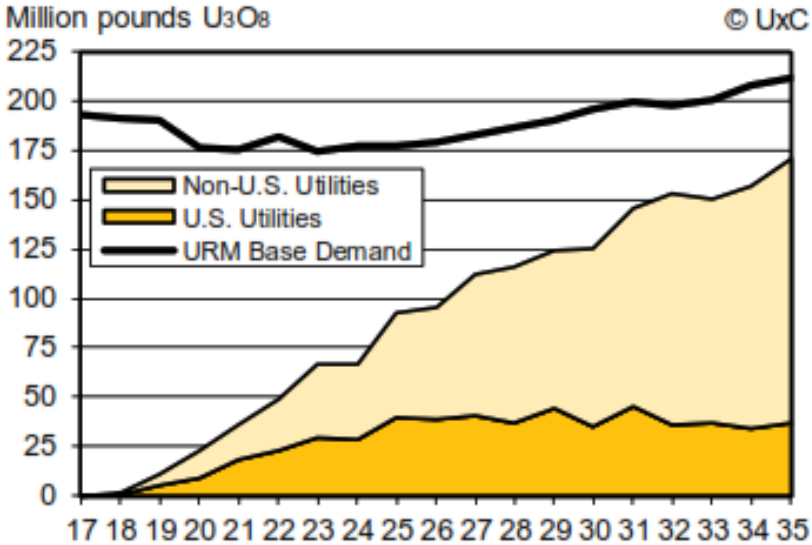


Figure prepared by UxC, 2018.

19.1.4 Market Outlook

Information in this sub-section is abstracted from UxC, 2018.

The three-month spot price variance (from August 2018) improves the third quarter to US$24–US$29 from US$20–US$25 in the second quarter of 2018 as lower-cost inventory supplies have cleared the market due to heavy investor purchasing and supplier/trader purchasing following a round of new production cuts/suspensions.

Although utility demand is not foreseen improving much in the three-month forward period due to sizable inventory positions, some U.S. utilities could seek out U.S.-origin material ahead of the Section 232 decision expected next year.

Intermediaries are expected to remain active buyers, but this group is becoming less aggressive on the sell side. The 12-month spot price range increases to US$24–US$31 from US$19–US$26 in the second quarter of 2018 for many of the same reasons in the three-month spot price variance.

Recent primary production cuts are serving to realign supply with demand. As more primary production is cut and suppliers elect to purchase available low-cost secondary supply to meet forward sales commitments, spot prices are expected to continue trending higher as successive tranches of low-cost supply are removed from the market

19.2 Commodity Price Projections

19.2.1 Overview

Uranium pricing used in the 2018 Pre-feasibility Study is derived from projections prepared by UxC.

The combination of term price forecasts and the projected spot prices provided UxC with the essential inputs for deriving an annualized forward price curve, which was used to quantify pricing information and develop risk aversion tactics for utility fuel managers and financial planners.

There are three scenarios for long-term (LT) contract base price projections: Mid LT Base, High LT Base, and Low LT Base. The two commonly used pricing approaches in long-term uranium contracts are base- escalated and market-related pricing. The term price projected using UxC's proprietary U-PRICE model was the base price of uranium in long-term contracts signed in any given year. The UxC 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 from contracts signed at different points in time. In addition, the average delivery price also includes prices under market price contracts.

The long-term contract base price projection scenarios can be summarized as:

- Low long-term base: Assumes further delays in Japan's reactor restarts and China scaling back its reactor construction program, a growing number of premature reactor closures around the world will place more downward pressure on the long-term base price. As uranium demand continues to decline at a faster pace than a reduction in total supply, the term price is projected to stay below the $50 mark until 2033
- Mid long-term base: The term price is projected to move higher in the near term. However, limited uncovered requirements and excess inventories will continue to be the two key factors that cap upward momentum of the term price during the next several years. The term price will remain below the $40 mark until 2023. In addition, the path of price recovery could be volatile as the market needs to absorb excess supply and elevated inventories. The outcomes for several pending trade cases, such as the Section 232 petition filed by Energy Fuels and Ur-Energy, bring additional uncertainty to the nuclear fuel market. Over the long run, as the market reaches a more balanced state, more stable economic factors such as exchange rates will contribute to sustained upward price momentum
- long-term base: The key assumption in this scenario is a more positive market outlook due to the acceleration of restarting Japan's remaining idled reactors, a more solid growth pace for China's nuclear power program, and more timely changes to state-level policies that support the U.S. nuclear power industry. In addition, steady economic growth in the U.S. and a more rapid recovery of other major economies could help stabilize key macroeconomic instruments such as exchange rates and interest rates. Absent interference from external factors, the long-term base price has a better chance to recover due to improved market fundamental
- Probability-weighted average of the low, mid and high long-term base price scenarios. This is based on an assumption that during a rising price period, the high long-term base price will have a better chance of occurring than the low long-term base price. Conversely, when price is declining, the probability of reaching a low long-term base price would be notably greater than the high long-term base price being reached.

Error! Reference source not found. is a graph showing the price forecasts for each scenario. Information to 2017 is actual, from 2018 the data are forecast.

19.2.2 Forecasts

Consistent with UxC Mid long-term base guidance, commodity price forecasts used in the financial model in Section 22 assume:

- Uranium price of US$50/lb U_3O_8 based on long-term forecasts
- 100% of uranium sold at long-term price of US$50/lb U_3O_8

The projected exchange rate is C$1.00 = US$0.75.

19.3 Contracts

NexGen is considering selling all production from the Project through 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.

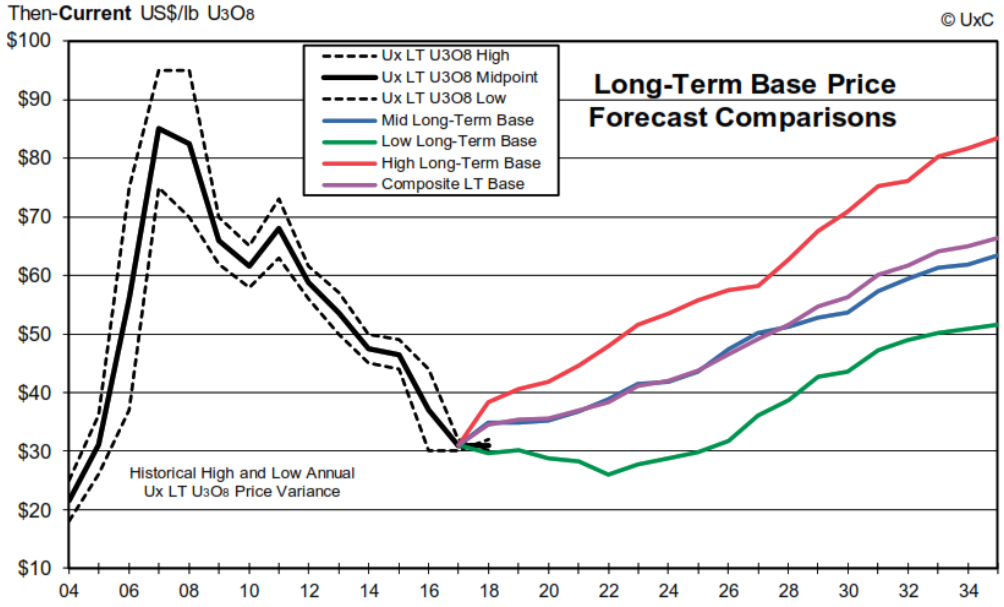
No contracts have currently been entered into for the Project.

19.4 Comments on Section 19

Uranium price assumptions are based on long-term forecasts. No contracts have been entered into for sales of uranium product from the Project.

The QP is of the opinion that the information on marketing and metal price forecasts is acceptable to support the financial analysis in Section 22.

Figure 19-5: Long-Term Base Price Projection Comparison, 2004–2035



Note: Figure prepared by UxC, 2018.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Baseline Studies

Environmental baseline studies have been undertaken to gather information on the current conditions for the biophysical, cultural and socioeconomic environment in the area of and relation to the Project.

Completed studies or studies that are underway are summarized in Table 20-1.

20.2 Waste Rock Storage Facilities

About 5.1 Mm^3 of clean waste rock will be generated over the course of the LOM. This material is considered non-mineralized and non-acid generating and therefore the storage area will not need to be lined. However, the runoff in contact with the stockpile will be diverted and controlled. The runoff from the waste rock pile will drain north and west and will be intercepted by a diversion ditch. It will then flow to either the east waste rock runoff pond or the west collection basin.

The waste rock will be stockpiled with 4:1 side slopes to minimize shaping work at the end of the Project. 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 is a layout plan showing the waste rock stockpile layout plan.

It is estimated that about 1% of the waste rock brought to surface will be mineralized but will not meet economic criteria. This material is considered special waste rock and must be stored in a contained area. This area will be dual-lined with high-density polyethylene (HDPE) and will have a leak detection system installed between the liners. Similar to the ore storage pad, runoff from the special waste rock will be contained within the dual-lined area and that area designed to hold runoff anticipated from a probable maximum precipitation (PMP) rain event. Water will be collected and transferred to the settling pond for treatment in the effluent treatment circuit.

20.3 Underground Tailings Management Facility

This facility is discussed in Section 16.5.

Table 20-1: Baseline Studies Initiated

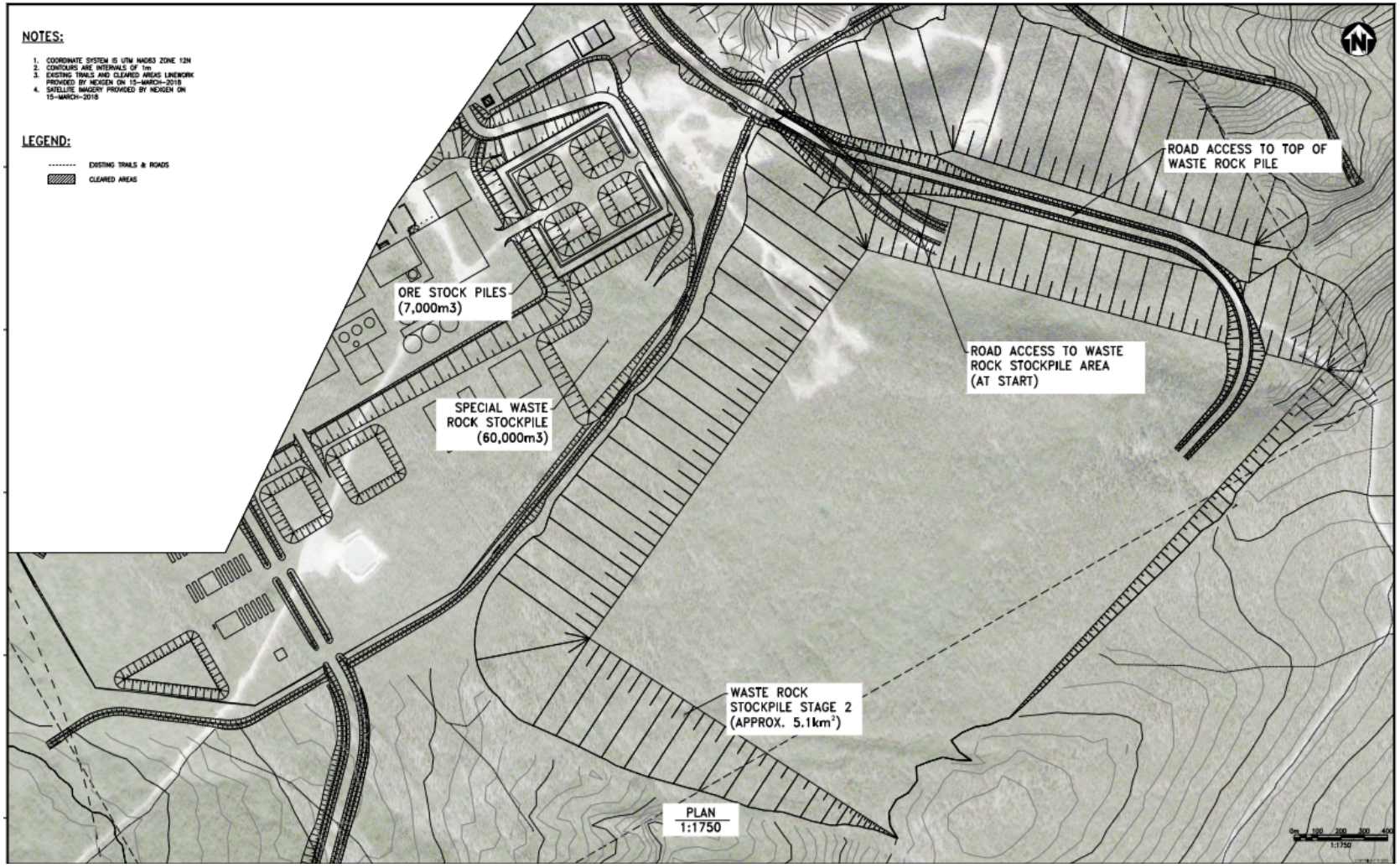
Study	Purpose	Comments
Air quality	Characterize climate conditions and meteorology in the Project vicinity; characterize existing air quality conditions and ambient background concentrations for selected constituents of potential concern; measure precipitation quantity to determine the conditions of rainfall in the area; measure existing ambient baseline radon levels at the Project site; and provide local and regional baseline data and a basis for long-term monitoring studies that can be integrated with other environmental components	The baseline program was developed to sufficiently robust to acquire enough data to satisfy the requirements outlined in the Saskatchewan Air Quality Modelling Guideline (AQMG) requirements. The meteorological program will provide a robust site-specific data set that will assist NexGen in understanding annual trends and climate extremes in the local area and provide for a multi-year climate baseline. The baseline air quality program was initiated in Q3 2018 and will continue, in whole or in part, throughout Project construction and operations.
Acoustic and light	Establish the existing outdoor acoustic environment at locations that may be sensitive to noise from the proposed development; and measure existing light levels within the local area and establish a representative baseline for light trespass and sky glow	The baseline noise and light field program was completed in Q3 2018. The results of the program are pending.
Geology and soils	Geology: characterize site specific, local and regional geology in the area surrounding Project that may influence or be influenced by the Project; provide geotechnical information and gather information for development of a detailed geological model of overburden and bedrock units Soils: obtain local and site-specific information on terrain; characterize existing soil quality and distribution across the Project area; determine baseline soil chemistry, and evaluate soil sensitivities near the Project site; provide information on potential sensitive terrain and/or soils that can be used in the project design to mitigate effects; and use this information to support the assessment of potential effects from the Project on terrain and soil	The site geological features were characterized through extensive drilling conducted from 2014–2018. A baseline geotechnical program was conducted in 2018 to provide further information on the geological characteristics within the Project vicinity. Baseline metal chemistry is an indicator of soil quality and the terrain and soils baseline program was conducted in 2018 that included the collection of a composite soil samples for metals, and radionuclides analysis. Sampling locations were co-located with vegetation chemistry sampling to provide integration between the two components and to meet expected requirements for future monitoring. The terrain and soils baseline program is was completed in Q4 2018. The results of the program are pending.

Study	Purpose	Comments
Hydrogeology	Provide information on hydrogeological flow conditions, including rates and direction, at both a local and regional scale; characterize baseline groundwater quality in the overburden and bedrock structures surrounding the Project; and collect adequate information for development of a hydrogeological model specific to the Project for use in planning, design and assessment activities	A baseline hydrogeological program was first established in 2017 with the installation of several monitoring wells and vibrating wire piezometers at the site. Following review of the initial hydrogeological data, additional monitoring wells were installed in 2018, including a deep-well for characterization of hydrogeology in the bedrock basement unit. Sampling is conducted on a quarterly basis and includes water level measurement and detailed chemical analysis for a broad range of parameters of interest specific to the uranium mining industry. The monitoring program is expected to provide a basis for ongoing long-term monitoring
Hydrology	Characterize the Project hydrological baseline conditions to support the assessment of potential environmental effects; and support Project engineering design throughout its various stages	Core activities for the baseline program are hydrometric monitoring, and water level, stream discharge, and sediment transport measurements. An initial hydrology field program was completed in the summer of 2018, consisting of establishing hydrometric monitoring stations in watercourses, existing hydrometric stations in waterbodies were visited, and continuous water level monitoring instrumentation was installed. A second hydrological baseline field trip was completed in Q3 2018. In addition to collecting information from the previously-established sites, additional studies included completion of stream channel surveys, a geodetic survey, geomorphological field study and a mixing study. Hydrological monitoring will be ongoing through the construction and operating periods.
Surface water quality	Identify any waterbodies that may be potentially influenced by the Project both at a local and regional scale; characterize existing water quality through detailed chemical analysis and field measurement for waterbodies identified for study; and provide adequate baseline data for use in modelling and assessment activities	Surface water quality is evaluated through direct measurement and the collection of representative samples for chemical analysis. Chemical analysis is conducted by an independent analytical laboratory and includes analysis for a wide range of chemical parameters of interest specific to the uranium mining industry. Water quality monitoring studies were initiated in May 2018. Stations were established on a number of lakes and streams in the vicinity of the Project, located at both a local and regional scale. Monitoring is conducted at each station on a quarterly basis

Study	Purpose	Comments
		providing representation of conditions throughout the year. Water quality monitoring will be ongoing through the construction, operation and decommissioning phases.
Aquatic resources	Characterize the existing aquatic environment in waterbodies that may influence or be influenced by the Project; establish baseline chemistry for key components of the aquatic environment; characterize fish habitat and fish communities in waterbodies that may be influenced by the Project; characterize supporting variables, such as sediment quality, plankton and benthic invertebrate communities, in the waterbodies identified; and provide information on various aquatic resources and components for modeling and assessment	Aquatic resource baseline studies were initiated in 2018 and conducted within a number of waterbodies within the local and regional study areas. These included: fish habitat assessment; fish community surveys; fish chemistry sampling; fish spawning surveys (spring and fall); sediment chemistry; benthic invertebrate community surveys; and phytoplankton/zooplankton community surveys
Vegetation	Characterize the presence and abundance of various plant species and community assemblages within the vicinity of the Project; establish baseline chemistry in select vegetation species in the Project area; determine the presence of any invasive or noxious plant species within the vicinity of the Project; identify any rare or endangered species which may occur in the vicinity of the Project; and provide information for development of ecosite and habitat maps	Baseline vegetation studies were conducted in 2018 at a local and regional scale. Information was collected through a combination of vegetation transects and establishment of vegetation plots. Baseline studies included mapping using a combination of satellite and aerial imagery and vegetation surveys.
Wildlife	Provide information on the presence and abundance of resident and migratory wildlife within the local and regional study areas; characterize wildlife habitat and use; establish the condition of select wildlife components through chemical analysis of collected samples; identify of any rare or endangered wildlife species present at an appropriate scale; and outline specific details for inclusion in modeling and assessment	Baseline wildlife studies were first conducted in 2016 with a comprehensive program completed throughout 2018 at both a local and regional scale.

Study	Purpose	Comments
Cultural studies	Determine the presence of any culturally or archaeologically significant sites or artifacts within the vicinity of the Project, and consider traditional Indigenous land use practices within the Project vicinity	Heritage resource surveys have been completed throughout the Project area to determine the presence of any sites of cultural or historic significance. Research into traditional land use practices has been initiated with further studies incorporating local and indigenous knowledge to be initiated through engagement with local Indigenous communities.
Socio-economic	Characterize current socio-economic and health conditions within communities identified as being potentially influenced by the Project; and the quality of various traditional foods collected and consumed from the region	Factors being considered include health, social and economic indicators. This considers information available through public records and community resources. Further phases will incorporate direct input from local and Indigenous communities to provide further detail for consideration in future planning, design and assessment stages.

Figure 20-1: Waste Rock Storage Facility Layout Plan



Note: Figure prepared by Wood, 2018

20.4 Water Management

For the 2018 Pre-feasibility Study design, water management infrastructure has been designed to maximize diversion of surface runoff water from the general site footprint and disturbed area. Water that interacts with the clean waste rock pile will be captured and controlled.

The storm water runoff pond (SWRP) is designed to hold runoff from the immediate plant-site and mine island footprint following a 24-hour 1:100 year storm event. The pond will have dual HDPE liners to provide dual containment and will include a leak detection system between the two liners. All ponds and pads containing mineralized or radiologically contaminated material have been designed to accommodate a 24-hour probable maximum precipitation (PMP) Additional runoff volumes from the general site footprint and disturbed area anticipated from a 24-hour PMP storm event will be captured by a constructed berm and retention area located between the site and Patterson Lake to the north. This secondary storage is considered as contingency containment and will only be required during to contain runoff in the unlikely event that rainfall or runoff exceed the 1:100 year storm event. The contingency containment area is intended only to prevent suspended solids entrained in the runoff from entering Patterson Lake and will not be lined, allowing for natural dissipation.

The storage areas for mineralized ore and mineralized/special waste rock are similarly planned to be dual-contained with HDPE liners and will be able to store volumes of water associated with a PMP storm event. These areas will also contain separate leak detection pipe under-drain systems. The clean waste rock stockpile and the undisturbed area uphill from site will be intercepted by a diversion ditch and will collect in runoff retention areas located to the east and west of the site. The east collection area will consist of a constructed pond with one HDPE liner layer. The west collection area will be an unlined area that will be able to hold very large volumes of runoff water.

Six water storage ponds are planned, and will include four 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 event. The feed settling pond will have a capacity of 16,000 m^3 with 1 m freeboard. About 1,100 m^3 of the feed settling pond capacity is reserved for a 1:100 year 24-hour storm event which comes from rain runoff collecting immediately surround 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 24-hour 100-year storm event as well as the anticipated volumes of water generated under routine and non-routine operating conditions.

20.5 Closure Plan

20.5.1 Regulatory Process

Prior to 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.

As part of the detailed decommissioning plan, site-specific end-state environmental and radiological objectives and criteria will be established and will be used to evaluate progress. Monitoring will be conducted throughout the decommissioning period and results will be compared to established objectives and criteria. Upon completion of decommissioning these monitoring results will be used to confirm that end-state objectives have been achieved and that the site has been returned to a safe, stable and improving condition.

Final closure of the site will be precluded by the completion of decommissioning activities and by the achievement of end-state objectives. An application to abandon the property will be made to Provincial and Federal authorities and the site will be transferred back to the Province through the Institutional Control Program in accordance with *The Reclaimed Industrial Sites Act* and *The Reclaimed Industrial Sites Regulations*.

20.5.2 Decommissioning Process

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 practical, surface and underground 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.

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 underground where they will be incorporated 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 similarly transferred underground and backfilled within the mine. These areas will then be subject to testing and radiological surveys to ensure conditions meet established criteria.

Roadways, storage pads, building foundations, ditches, berms and other earthworks components will undergo radiological survey prior to decommissioning. Material not meeting decommissioning criteria will be removed and backfilled as part of mine decommissioning with areas re-surveyed until criteria are achieved. These areas will then undergo contouring, scarification and revegetation with approved native vegetation species.

A portion of the stockpiled clean waste rock will be used to backfill mine shafts and as supplemental fill where required for earthworks. The remaining clean waste rock 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, to provide for the 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 within the second shaft along with any remaining surface infrastructure and fill material.

Underground Workings

Mine decommissioning will occur in parallel with surface decommissioning, with designated surface materials, transferred underground and backfilled into the lateral portions of the mine along with mine infrastructure, equipment and material. Backfilled material will be placed utilizing available space until all designated waste has been removed from surface. During this period, the mine will continue to be dewatered, ventilated and required infrastructure maintained.

Shafts will be decommissioned sequentially following completion of backfilling in the lateral portions of the mine. The lower portion of the shaft, 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, 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 up to decommissioning of the first shaft, at which time the water treatment system will be decommissioned and placed within the bottom portion of the second shaft during backfilling. All other openings to surface will be filled with a low conductivity, impermeable material and sealed at surface during decommissioning.

The UGTMF will be progressively decommissioned during operation and active decommissioning will not be required during the decommissioning phase. Facility performance and environmental monitoring criteria established during 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.

20.5.3 Closure Costs

NexGen has allocated closure costs as part of the capital cost estimate in Section 21.2.

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 26.

20.6 Permitting

20.6.1 Regulatory Framework

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 and Control Act* (NSCA), proponents wishing to carry out activities related to the preparation, construction, operation, decommissioning and abandonment of a uranium mine and mill in Canada must first obtain a licence from the federal nuclear regulator, the Canadian Nuclear Safety Commission (CNSC). Before the CNSC can make a licensing decision, proponents are required to undergo an environmental assessment (EA) of the proposed project under the Canadian Environmental Assessment Act 2012. In addition, a new uranium mine and mill project is subject to a number of other federal regulatory requirements and project approvals.

In Saskatchewan, new uranium mines and mills are subject to *The Environmental Assessment Act* that requires the completion of an EA and approval from the Minister of Environment before proceeding with a new development. Developments that are approved are also required to obtain approvals under the *Environmental Management and Protection Act, 2010*.

As the Project falls under both Federal and Provincial jurisdiction for EA which means the CNSC and Saskatchewan Ministry of Environment (MOE) – Environmental Assessment Branch (EA Branch) will each require an EA prior to project approval. The CNSC and MOE will coordinate the 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 and fulsome process is followed to limit environmental effects. Under this process, the CNSC acts as the responsible EA authority and acts as the lead agency overseeing the EA process, coordinating activities with the Provincial and other Federal agencies to ensure all regulatory considerations are taken into account.

Federal Environmental Assessment Process

Under the *Canadian Environmental Assessment Act 2012* (CEAA 2012) environmental assessments are required for designated projects, which are defined under the *Regulations Designating Physical Activities* (RDPA). The construction of a new uranium

mine and mill is considered a designated activity as defined by the RDPA with the CNSC designated as the responsible authority for the oversight of the EA; therefore, an EA under CEAA 2012 will be required for the Project.

Provincial Environmental Assessment Process

In Saskatchewan, EAs are governed under *The Environmental Assessment Act*, which requires that a proponent receives approval from the Minister of Environment before proceeding with a development as defined in *The Environmental Assessment Act*.

20.6.2 Licensing and Approval Process

Under Federal rules, 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. The regulations made under the NSCA 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 four general licensing phases:

- License to Prepare Site and Construct
- License to Operate
- License to Decommission
- License to Abandon.

Under the Provincial legislation, the Saskatchewan MOE 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* must be secured prior to construction and adhered to throughout the Project lifecycle.

The *Mineral Industry Environmental Protection Regulations (MIEPR)* governs the permitting and operation of mines and mills in Saskatchewan. Under MIEPRs, NexGen will be required to have an *Approval to Construct, Install, Alter or Extend a Pollutant Control Facility* prior to commencement of construction. In addition, an *Approval to Operate a Pollutant Control Facility* will be required prior to operations 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. A financial assurance will be required to cover the costs associated with executing the

decommissioning plan and this will require approval by both the CNSC and the Saskatchewan MOE.

20.6.3 Other Regulatory Requirements

Additional key Federal and Provincial regulatory approvals that the Project will need to consider are provided in Table 20-2 and Table 20-3, respectively.

20.6.4 Exploration Permits

NexGen requires the following permits in support of exploration drill programs:

- Aquatic habitat permit
- Forest protection permit
- Industrial permit
- Water use permit.

These permits were in place for the 2018 drill programs. NexGen has applied for permit renewals for the four permits in support of the planned 2019 drill program.

Table 20-2: Federal Acts and Regulations Potentially Applicable to the Project

Act	Regulations
CANADIAN ENVIRONMENTAL ASSESSMENT ACT, 2012*	Regulations designating physical activities*
	Prescribed information for the description of a designated project regulations*
	Cost recovery regulations*
NUCLEAR SAFETY AND CONTROL ACT	General nuclear safety and control regulations
	Uranium mines and mills regulations
	Radiation protection regulations
	Nuclear substance and radiation devices regulations
	Packaging and transport of nuclear substances regulations
	Nuclear security regulations
	Nuclear non-proliferation import and export control regulations
FISHERIES ACT*	Metal and diamond mining effluent regulations
	Deposit out of the normal course of events notification regulations
	Wastewater systems effluent regulations
CANADIAN ENVIRONMENTAL PROTECTION ACT	Environmental emergencies regulations
	Federal halocarbon regulations
PAN-CANADIAN FRAMEWORK ON CLEAN GROWTH AND CLIMATE CHANGE*	Greenhouse gas reporting program*
TRANSPORTATION OF DANGEROUS GOODS ACT	Transportation of dangerous goods regulations
AERONAUTICS ACT	Canadian Aviation Regulations
NAVIGATION PROTECTION ACT*	No specific regulations related to this act.
SPECIES AT RISK ACT	No specific regulations related to this act.
CANADIAN WILDLIFE ACT	Wildlife area regulations
MIGRATORY BIRDS CONSERVATION ACT	Migratory birds regulations
	Migratory birds sanctuary regulations
EXPLOSIVES ACT	Explosives regulations

Note: * denotes legislation that is currently under review or revision and is subject to change during the Project lifecycle

Table 20-3 Provincial Acts and Regulations Potentially Applicable to the Project

Act	Regulations
ENVIRONMENTAL ASSESSMENT ACT	No specific regulations related to this act.
ENVIRONMENTAL MANAGEMENT AND PROTECTION ACT, 2010	Environmental management and protection (general) regulations
	Mineral industry environmental protection regulations, 1996
	Environmental management and protection (Saskatchewan environmental code adoption) regulations
	Discharge and discovery reporting chapter
	Site assessment chapter
	Corrective action plan chapter
	Halocarbon control chapter
	Environmental code of practice on halons
	Industrial source (air quality) chapter
	Hazardous substances and waste dangerous goods regulations
	Municipal refuse management regulations
	Waterworks and sewage works regulations
WATER SECURITY AGENCY ACT	Water security agency regulations
	Withdrawal from allocation regulations
FISHERIES ACT (SASKATCHEWAN), 1994	Fisheries regulations
GROUNDWATER CONSERVATION ACT	Groundwater protection regulations
WILDLIFE ACT, 1998	Wildlife regulations
	Wild species at risk regulations
FOREST RESOURCES MANAGEMENT ACT	Forest resources management regulations
WILDFIRE ACT	Wildfire regulations
PROVINCIAL LANDS ACT, 2016	Saskatchewan wetland conservation corporation land regulations
	Crown resource land regulations
	Provincial lands regulations

Act	Regulations
HERITAGE PROPERTY ACT	Heritage property regulations
CROWN RESOURCES ACT	Crown resource land regulations, 2017
MINERAL RESOURCES ACT	Quarrying regulations
NATURAL RESOURCES ACT	Resource protection and development service regulations
PEST CONTROL ACT	Pest declaration regulations
	Pest control products amendment regulations, 2012
WEED CONTROL ACT	Weed control regulations
MANAGEMENT AND REDUCTION OF GREENHOUSE GASES ACT*	No specific regulations related to this act.
NORTHERN MUNICIPALITIES ACT, 2010	Northern municipalities regulations
SASKATCHEWAN EMPLOYMENT ACT	Occupational health and safety regulations, 1996
	Mines regulations, 2018
RADIATION HEALTH AND SAFETY ACT, 1985	Radiation health and safety regulations, 2005
BOILER AND PRESSURE VESSEL ACT	Boiler and pressure vessel regulations
TECHNICAL SAFETY AUTHORITY OF SASKATCHEWAN ACT	No specific regulations related to this act.
ELECTRICAL INSPECTION ACT	Electrical inspection regulations
GAS INSPECTION ACT	Gas inspection regulations
PUBLIC HEALTH ACT	Food safety regulations
	Plumbing and drainage regulations
	Public sewage works regulations
	Public accommodations regulations
PASSENGER AND FREIGHT ELEVATOR ACT	Passenger and freight elevator regulations, 2017
RECLAIMED INDUSTRIAL SITES ACT	Reclaimed industrial sites regulations
TREATY LAND ENTITLEMENT ACT	No specific regulations related to this act.

Note: *denotes legislation that is currently under review or revision and is subject to change during the Project lifecycle.

20.7 Considerations of Social and Community Impacts

NexGen has developed a plan for public and Indigenous engagement specific to the Project. The plan is intended to foster long-term positive relationships with the public and Indigenous communities, and demonstrates NexGen's commitment to the following:

- Establishing and maintaining meaningful relationships with public and Indigenous communities in the general vicinity of the Rook I Project
- Providing accurate and timely information to public and Indigenous leadership and community members about the Rook I Project
- Receiving and considering feedback, including concerns or suggestions, and considering this feedback in environmental and technical studies and in the planning and design of the Rook I Project.

The plan for public and Indigenous engagement identifies early and ongoing communication and involvement with the public and Indigenous communities as an important aspect of the Rook I Project. This Plan is also intended to achieve the following:

- Assist with the identification, documenting and screening of important and relevant issues
- Provide a basis upon which the expectations of the public and Indigenous communities can be effectively managed
- Demonstrate that NexGen is incorporating public and Indigenous input into the Project.

Early and ongoing engagement is an important factor in establishing dialogue and fostering understanding between the proponent, regulators, the public, Indigenous communities, and special interest groups. Throughout the EA and licensing processes, establishing and maintaining positive and respectful relationships and allowing opportunity for input is important. NexGen will maintain a transparent process whereby issues are understood, considered, addressed where possible and practical, and a defensible scientific analysis is completed whereby concerns can be referred during the EA, allowing the regulatory process to make informed decisions with respect to the Project.

20.8 Comments on Section 20

Baseline studies have commenced or are planned to support Project permitting. NexGen will need to comply with a number of Federal and Provincial acts during construction and operations. NexGen has developed a Project-specific plan for engaging the public and Indigenous communities on the Project.

21.0 CAPITAL AND OPERATING COSTS

21.1 Introduction

Capital and operating costs estimates were prepared by Wood and RPA, with contributions from NexGen.

21.2 Capital Cost Estimates

21.2.1 Basis of Estimate

The estimate meets the classification standard for a Class 4 estimate as defined by AACE International, and has an intended accuracy of ±30%. The estimate is reported in Q3 2018 Canadian dollars.

The capital cost estimate reflects a detailed bottom-up approach that is 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. Capital costs are divided among the areas of underground mining, processing, infrastructure, indirect expenses and contingency. Sustaining capital costs are related to underground mine equipment and development, process plant and infrastructure maintenance and mine closure.

21.2.2 Mine Capital Costs

Mining capital costs primarily comprise the following areas: shaft sinking, lateral mine development, and stationary mine infrastructure. Mine mobile equipment, typically another major cost area, is assumed to be purchased on a "lease to own" basis, with the costs incurred in the lease payments counted toward operating costs. RPA assumed that a shaft sinking and mine development contractor would be operating at the site from Year -3 to Year 1 (a total of four years). The mine contractor would be responsible for sinking both shafts concurrently, developing the underground drifts, including the internal ramp, footwall and hanging wall access drifts, other underground mine infrastructure, and the ventilation system. The contractor would also develop all internal vertical development (ventilation raises, ore and waste passes). The mining capital cost contribution is summarized in Table 21-1.

Table 21-1: Mine Capital Cost Estimate

Area	Units	Cost
Shaft sinking	C$ millions	107.4
Underground mine development	C$ millions	65.7
Underground mining equipment	C$ millions	-
Underground mine infrastructure and services	C$ millions	31.8
Ventilation and services	C$ millions	16.3
Underground personnel and material handling	C$ millions	19.7
UGTMF	C$ millions	12.6
Mine surface infrastructure	C$ millions	49.9
Total	***C$ millions***	***303.4***

Note: Totals may not sum due to rounding

21.2.3 Process Capital Costs

Process plant costs were divided between process plant directs and process plant construction, and are summarized in Table 21-2.

21.2.4 Infrastructure Capital Costs

Infrastructure costs will include provision for the LNG power plant, as well as allocations for site preparation, permanent camp, maintenance shop, fuel storage, administration and dry facility, water treatment systems, airstrip and site roads. Costs are summarized in Table 21-3.

21.2.5 Indirect Costs

Indirect costs include temporary construction facilities, construction services and supplies, engineering, procurement and construction management (EPCM) costs, construction equipment, freight, Owner's costs and contingency. Indirect costs are shown in Table 21-4.

21.2.6 Contingency

Contingencies were applied to each of the respective areas of the cost estimate, as summarized in Table 21-5.

Table 21-2: Process Capital Cost Estimate

	Description	Units	Cost
Direct	Ore handling and crushing	C$ millions	3.8
	Grinding	C$ millions	6.6
	Leaching	C$ millions	3.8
	Liquid/solids separation	C$ millions	22.5
	Solvent extraction	C$ millions	18.5
	Precipitation	C$ millions	9.5
	Tailings neutralization	C$ millions	4.6
	Product drying and packaging	C$ millions	4.7
	Reagents	C$ millions	4.4
	Acid plant	C$ millions	28.2
	Subtotal direct process capital costs	*C$ millions*	*106.6*
Construction	Earthworks and civil	C$ millions	0.7
	Concrete	C$ millions	13.0
	Structural steel	C$ millions	7.0
	Buildings architectural	C$ millions	7.0
	Building services	C$ millions	24.8
	Electrical	C$ millions	4.7
	Instrumentation	C$ millions	12.2
	Piping within process plant	C$ millions	31.5
	Insulation and protection	C$ millions	3.1
	Acid plant construction costs	C$ millions	28.2
	Subtotal process construction capital costs	*C$ millions*	*132.1*
	Total process capital costs	**C$ millions**	**238.7**

Note: Totals may not sum due to rounding

Table 21-3: Infrastructure Capital Cost Estimate

Description	Units	Cost
Propane storage facility	C$ millions	2.8
Diesel fuel storage facility	C$ millions	0.2
Gasoline fuel storage facility	C$ millions	0.1
Site preparation - stripping and grubbing	C$ millions	16.5
Site preparation - HDPE liners for pads	C$ millions	2.3
Site roads	C$ millions	3.4
Explosives magazine	C$ millions	0.2
Permanent camp	C$ millions	17.3
Maintenance shop	C$ millions	11.0
Administration and dry facility	C$ millions	9.7
Warehouse	C$ millions	0.0
Water treatment facilities	C$ millions	22.0
Site power grid (surface)	C$ millions	10.0
Power plant	C$ millions	25.8
Airstrip including apron and hangar	C$ millions	5.8
Fire protection	C$ millions	4.9
Total infrastructure capital costs	***C$ millions***	***131.8***

Note: Totals may not sum due to rounding

Table 21-4: Indirect Capital Cost Estimate

Description	Direct Cost (C$ millions)	Indirect (%)	Indirect Cost (C$ millions)
Infrastructure	131.8	61.7	81.3
Capital mine development	185.7	37.3	69.2
Underground mining mobile and fixed equipment	117.7	37.3	43.9
Capitalized pre-production operating costs	23.7	NA	NA
Processing	238.7	61.7	147.2
Total indirect capital costs	***697.5***	***49.0***	***341.6***

Note: Totals may not sum due to rounding

Table 21-5: Contingency Capital Cost Estimate

Description	Direct and Indirect Costs (C$ millions)	Contingency (%)	Contingency (C$ millions)
Infrastructure	213.0	20.0	42.6
Capital mine development	254.9	20.0	51.0
Underground mining mobile and fixed equipment	161.6	20.0	32.3
Capitalized pre-production operating costs	23.7	20.0	4.7
Processing	385.9	20.0	77.2
Total contingency capital costs	**1,039.1**	**20.0**	**207.8**

Note: Totals may not sum due to rounding

21.2.7 Sustaining Capital

Table 21-6 provides a summary of the projected sustaining and reclamation capital cost estimates for the Project.

Underground

Sustaining capital incorporates all capital expenditures after the pre-production period of Year -3, Year -2, and Year -1. Mine sustaining capital cost accounts for some areas of capital spending that were not completed until Year 1, as well as certain pieces of equipment that will be rebuilt over the LOM, and ongoing mine development. Sustaining capital for the mine infrastructure was also included, namely the replacement of the production hoist cables, as well as an amount meant to address replacement of fans, pumps, and other mine capital items on an ongoing basis. The mine has a relatively sustaining-short mine life, and therefore minimal major replacements of mobile or stationary equipment are planned.

The UGTMF will be constructed in phases, with the first phase being completed during the pre-production years. Subsequent expansions of the UGTMF will occur during the production period, on an as-needed basis.

Table 21-6: Sustaining and Reclamation Capital Cost Estimates

Description	Units	Total
Underground mining equipment	C$ millions	142.5
Underground mine development	C$ millions	51.4
Process plant	C$ millions	12.9
Infrastructure	C$ millions	7.1
Total sustaining capital costs	*C$ millions*	*213.9*
Reclamation and closure	C$ millions	48.0
Total sustaining and reclamation	**C$ millions**	**261.9**

Note: Totals may not sum due to rounding

Surface

Sustaining capital on surface will be used to replace worn-out equipment and to upgrade the facilities to improve operating efficiency. Sustaining capital was estimated for surface as a percentage of the replacement capital cost of the plant and surface infrastructure. For the 2018 Pre-feasibility Study, 0.5% of the replacement cost (less Owner's costs) was used and the sustaining capital was divided between the plant and surface infrastructure by the percentage of capital in each area. Sustaining capital starts in Year 2 at 50% of the estimated amount and ramps up to the full amount in Year 6. There is no sustaining capital in Year 9 as operations ramp down

Closure and Reclamation

NexGen prepared an estimate to decommission and reclaim the site. Reclamation costs of $48 million have been included in Year 9.

21.2.8 Capital Cost Summary

Table 21-7 outlines the estimated capital cost for designing, supplying, constructing and pre-commissioning the Project.

Table 21-7: Total Capital Cost Estimate

Description	Units	Cost
Underground mining	C$ millions	303.4
Processing	C$ millions	238.7
Infrastructure	C$ millions	131.8
Subtotal pre-production direct costs	*C$ millions*	*673.9*
Pre-production indirect costs	C$ millions	365.2
Subtotal direct and indirect	*C$ millions*	*1,039.1*
Contingency	C$ millions	207.8
Initial capital cost	***C$ millions***	***1,246.9***
Sustaining	C$ millions	213.9
Closure	C$ millions	48.0
Total	***C$ millions***	***1,508.9***

Note: Totals may not sum due to rounding

21.3 Operating Cost Estimates

21.3.1 Basis of Estimate

Operating cost estimates were developed to show annual costs for production. Unit costs are expressed as C$/tonne processed and C$/lb U_3O_8. Operating costs were allocated to one of mining, process or general and administration (G&A).

21.3.2 Mine Operating Costs

Underground mining takes place during Year -2 to Year 9 (note in Years -2 and -1 underground mining costs are capitalized). Underground mining begins with capital development in Year -2 and the capitalized development continues through Year 8. Underground mining costs are summarized in Table 21-8.

21.3.3 Process Operating Costs

Process operating costs are primarily composed of labour, power consumption, water and consumables. Consumables consist of reagents, grinding media, mill liners and LNG. An allowance was made for annual maintenance. Process costs are included in summary format in Table 21-9.

Table 21-8: Underground Mine Operating Cost Estimate

Description	LOM Cost (C$ millions)	Unit Cost (C$/t processed)	Unit Cost (C$/lb U_3O_8)
Labour	276.6	81.01	1.21
Mine consumables	73.6	21.55	0.32
Equipment operations & maintenance	107.1	31.38	0.47
Power consumption	67.8	19.85	0.30
In-fill drilling	12	3.51	0.05
Total	***537.1***	***157.31***	***2.35***

Note: Totals may not sum due to rounding

Table 21-9: Process Operating Cost Estimate

Description	LOM Cost (C$ millions)	Unit Cost (C$/t processed)	Unit Cost (C$/lb U_3O_8)
Labour	104.0	30.46	0.46
Power	69.5	20.35	0.30
LNG for calciner	4.8	1.42	0.02
Water	11.9	3.47	0.05
Reagents	315.8	92.51	1.38
Maintenance materials	55.1	16.14	0.24
Laboratory	1.0	0.30	0.00
Total	***562.1***	***164.65***	***2.46***

Note: Totals may not sum due to rounding

21.3.4 Infrastructure Operating Costs

G&A costs (Table 21-10) 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 made for reimbursable fees paid to the CNSC.

Table 21-10: Infrastructure Operating Cost Estimate

Description	LOM Cost (C$ millions)	Unit Cost (C$/t processed)	Unit Cost (C$/lb U_3O_8)
Labour	87	25.61	0.38
Camp costs	43	12.65	0.19
Flights and logistics	39	11.55	0.17
Miscellaneous	46	13.56	0.20
Equipment maintenance and fuel	5.9	1.72	0.03
Power	6.9	2.03	0.03
Total	**229.1**	**67.11**	**1.00**

Note: Totals may not sum due to rounding

21.3.5 Operating Cost Summary

LOM operating costs are estimated to be C$1,328.3 million. LOM operating costs are summarized in Table 21-11.

21.4 Comments on Section 21

Total capital costs, including initial, sustaining and reclamation costs, total C$1,508.9 million.

LOM operating costs are estimated to be C$1,328.3 million.

Table 21-11: Operating Cost Estimate Summary

Description	LOM Cost (C$ millions)	Average Annual (C$ millions)	Unit Cost (C$/t processed)	Unit Cost (C$/lb U_3O_8)
Mining	537.1	59.7	157.31	2.35
Processing	562.1	62.5	164.65	2.46
General and administration	229.1	25.5	67.11	1.00
Total	***1,328.3***	***147.6***	***389.07***	***5.81***

Note: Totals may not sum due to rounding

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 here. Forward-looking statements in this Report include, but are not limited to, statements with respect to future uranium prices, the estimation of Mineral Resources and Mineral Reserves, the estimated mine production and uranium recovered, the estimated capital and operating costs, and the estimated cash flows generated from the planned mine production. Actual results may be affected by:

- Differences in estimated initial capital costs and development time from what has been assumed in the 2018 Pre-feasibility Study
- 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 amount 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, 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.

The production schedules and financial analysis annualized cash flow table are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project assumptions as discussed in this Report and may result in changes to the calendar timelines presented and the information and statements contained in this Report.

No development approval has been forthcoming from the NexGen Board and statutory permits, including environmental permits, are required to be granted prior to mine commencement.

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 initial 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 flow projections. Cash flows are taken to occur at the end 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 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 2018 Pre-feasibility Study 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.

22.3 Financial Model Parameters

22.3.1 Mineral Resource, Mineral Reserve, and Mine Life

The Mineral Resource discussed in Section 14 was converted to the Mineral Reserve outlined in Section 15. The estimated Mineral Reserve will support a nine-year production life, using the mine plan as provided in Section 16.

22.3.2 Metallurgical Recoveries

The basis for the process recoveries is included in Section 13, and the process design is outlined in Section 17.

22.3.3 Commodity Prices

The commodity price basis is discussed in Section 19.

22.3.4 Capital and Operating Costs

The capital and operating cost estimates are detailed in Section 21.

22.3.5 Key Assumptions

Economic criteria that were used in the cash flow model include:

- Long-term price of uranium of US$50/lb U_3O_8 based on long-term forecasts
- 100% of uranium sold at long-term price of US$50/lb U_3O_8
- The recovery and sale of by-products is excluded from the cash flow model
- Exchange rate of C$1.00 = US$0.75
- Life of mine processing of 3,414 kt grading 3.09% U_3O_8
- Nominal 379 kt of material processed per year during steady state operations
- Mine life of nine years
- Overall process recovery of 97.6% including a ramp-up of recovery in Year 1
- Total recovered yellowcake of 228.4 Mlb
- Transportation costs of C$740/t of yellowcake with a presumed destination of Port Hope, Ontario
- Royalties calculated in accordance with *"Guideline: Uranium Royalty System, Government of Saskatchewan, June 2014"*
- Unit operating costs of C$389/t of processed material, or C$5.81/lb U_3O_8
- Pre-production capital costs of C$1,247 million, spread over three years
- Sustaining capital costs (including reclamation) of C$262 million, spread over the mine life.

22.4 Taxes

Taxes and depreciation for the Project were modelled based on input from NexGen, as well as a review 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. Under current Canadian tax codes, pre-production mine development costs are counted towards CEE; however, this is being phased out. Consequently, all pre-production capital was allocated to either CDE or CCA. Up to 30% of the CDE balance can be applied in any given year. All mining equipment and structures that are considered depreciable fall under Class 41 of Canadian tax codes, which can be depreciated at 25% annually.

In Saskatchewan, multiple government royalties exist for uranium projects. Royalties generally fall into two categories: revenue royalties and profit royalties. An explanation of the various royalties is provided below:

- Resource Surcharge of 3% of net revenue (where net revenue is defined as gross revenue less transportation costs directly related to the transporting of 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 on the first C$22.00 profit per kilogram of yellowcake, followed by 15% royalty on profits exceeding C$22.00 per kilogram.

In the tiered profit royalty, the basic royalty and resource surcharge are not deductible for calculating profit royalties. Profits for the purposes of royalties are calculated by taking the net revenue, subtracting the full value of operating costs, capital costs, and exploration expenditures. Revenue royalties were included in the "pre-tax" cash flow results, while profit royalties are considered a tax, and are included in "post-tax" results.

The royalties and carried interest that are applicable on certain mineral concessions have not been applied, as the Arrow deposit is not situated within those concessions.

Federal and Provincial taxes were applied at a rate of 15% and 12%, respectively.

Table 22-1 provides a summary of the taxes and royalties that are anticipated to be paid to the Provincial and Federal governments.

22.4.1 Closure Costs and Salvage Value

Reclamation costs were included with the capital cost estimate.

22.4.2 Financing

The base case economic analysis assumes 100% equity financing and 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 NPV8% is C$6,055.2 million, the IRR is 73.6%, and the assumed payback period is one year. On a post-tax basis, the NPV8% is C$3,661.1 million, the IRR is 56.8% and the assumed payback period is 1.2 years.

A summary of the LOM cashflow is provided in Table 22-2. Table 22-3 summarizes the economic results of the 2018 Pre-feasibility Study, with the NPV8% base case highlighted. A cashflow table on an annualized basis is provided in Table 22-4.

Table 22-1: Taxation and Royalty Considerations

	Description	Units	Value
Provincial payments	Saskatchewan resource surcharge	C$ millions	454.6
	Basic revenue royalty	C$ millions	644.0
	Profit royalty <C$22.00/kg	C$ millions	209.9
	Profit royalty >C$22.00/kg	C$ millions	1,517.4
	Provincial taxes	C$ millions	1,135.2
	Total provincial payments	*C$ millions*	*3,961.1*
Federal payments	Federal taxes	C$ millions	1,419.0
Total government royalties and taxes		***C$ millions***	***5,380.1***

Table 22-2: LOM Cashflow Forecast Summary Table

Description	Units	Value
Gross revenue	C$ millions	15,228.7
Less: transportation	C$ millions	(76.7)
Net smelter return	C$ millions	15,152.0
Less: provincial revenue royalties	C$ millions	(1,098.5)
Net revenue	C$ millions	14,053.5
Less: total operating costs	C$ millions	(1,328.3)
Operating cash flow	C$ millions	12,725.2
Less: capital costs	C$ millions	(1,508.9)
Pre-tax cash flow	C$ millions	11,216.3
Less: provincial profit royalties	C$ millions	(1,727.4)
Less: taxes	C$ millions	(2,554.2)
Post tax cash flow	***C$ millions***	**6,934.8**

Table 22-3: 2018 Pre-feasibility Study Forecast Economic Results (base case is highlighted)

Description	Units	Value
Pre-Tax		
Net present value at 8%	**C$ millions**	**6,055.2**
Net present value at 10%	C$ millions	5,232.7
Net present value at 12%	C$ millions	4,534.2
Internal rate of return	**%**	**73.6%**
Payback period	**years**	**1.0**
After-Tax		
Net present value at 8%	**C$ millions**	**3,661.1**
Net present value at 10%	C$ millions	3,140.3
Net present value at 12%	C$ millions	2,698.5
Internal rate of return	**%**	**56.8%**
Payback period	**years**	**1.2**

Table 22-4: Annualized Cashflow Forecast

	Units	PV	LOM	-3	-2	-1	1	2	3	4	5	6	7	8	9
Prices															
Uranium	C$/lb		67	66.67	66.67	66.67	66.67	66.67	66.67	66.67	66.67	66.67	66.67	66.67	66.67
Production															
Tonnes Milled	t		3,414,100	0	0	0	424,800	431,000	453,000	350,000	353,000	387,000	456,300	456,300	102,700
Uranium Production	lbs U_3O_8 x 1,000		228,430	0	0	0	25,075	29,927	29,949	29,966	29,990	29,952	27,954	19,362	6,255
Revenue															
Uranium	C$ millions		15,228.7	—	—	—	1,671.7	1,995.1	1,996.6	1,997.8	1,999.4	1,996.8	1,863.6	1,290.8	417.0
Total	C$ millions	8,671.9	15,228.7	—	—	—	1,671.7	1,995.1	1,996.6	1,997.8	1,999.3	1,996.8	1,863.6	1,290.8	417.0
Charges															
Transportation	C$ millions		76.7	—	—	—	8.4	10.0	10.1	10.1	10.1	10.1	9.4	6.5	2.1
Total															
Net Smelter Return	C$ millions		***15,152.0***	***—***	***—***	***—***	***1,663.2***	***1,985.1***	***1,986.5***	***1,987.7***	***1,989.3***	***1,986.7***	***1,854.2***	***1,284.3***	***414.9***
Gross Revenue Royalties	C$ millions		1,098.5	—	—	—	120.6	143.9	144.0	144.1	144.2	144.0	134.4	93.1	30.1
Net Revenue	C$ millions		***14,053.5***	***—***	***—***	***—***	***1,542.7***	***1,841.2***	***1,842.5***	***1,843.6***	***1,845.1***	***1,842.7***	***1,719.8***	***1,191.2***	***384.8***
Mine	C$ millions	302.2	537.0	—	—	—	63.4	69.7	70.3	59.4	58.5	59.0	61.5	57.2	38.1
Processing	C$ millions	311.7	562.1	—	—	—	55.3	64.1	68.9	66.1	65.9	68.2	70.8	66.2	36.6
G&A	C$ millions	126.7	229.1	—	—	—	25.8	25.8	25.8	25.8	25.8	25.8	25.8	25.8	23.0
Total onsite operating cost	C$ millions		1,328.3	—	—	—	144.4	159.5	165.0	151.3	150.2	153.0	158.1	149.2	97.7
Gross Revenue Royalties	C$ millions		1,098.5	—	—	—	120.6	143.9	144.0	144.1	144.2	144.0	134.4	93.1	30.1
Profit Royalties	C$ millions		1,727.4	—	—	—	7.0	255.0	255.5	257.1	259.2	257.9	239.3	159.1	37.3
Total SK Royalties	C$ millions	1,578.2	2,825.9	—	—	—	127.6	398.9	399.5	401.3	403.4	401.9	373.7	252.2	67.3
Operating cashflow	C$ millions	7,262.1	12,725.2	—	—	—	1,398.3	1,681.7	1,677.5	1,692.3	1,694.9	1,689.7	1,561.7	1,042.0	287.2
Taxes	C$ millions	1,441.4	2,554.2	—	—	—	225.5	318.0	333.1	348.2	357.8	363.6	339.3	224.5	44.2
Pre-Tax Cashflow	C$ millions		11,216.3	(177.8)	(549.9)	(519.2)	1,295.5	1,655.8	1,658.8	1,669.9	1,683.7	1,674.7	1,553.7	1,032.0	239.2
Cumulative Pre-Tax Cashflow	C$ millions			(177.8)	(727.8)	(1,246.9)	48.6	1,704.4	3,363.2	5,033.1	6,716.8	8,391.5	9,945.2	10,977.1	11,216.3
Taxes															
Less SK Profit Tax	C$ millions	952.6	1,727.4	—	—	—	7.0	255.0	255.5	257.1	259.2	257.9	239.3	159.1	37.3

	Units	PV	LOM	-3	-2	-1	1	2	3	4	5	6	7	8	9
Operating cash flow less SK profit tax	C$ millions		10,997.8	—	—	—	1,391.3	1,426.6	1,422.0	1,435.1	1,435.7	1,431.9	1,322.4	882.9	249.9
Less Deductions	C$ millions		1,700.1	14.8	39.1	108.4	556.1	248.9	188.3	145.6	110.4	85.1	65.7	51.3	86.4
Taxable Earnings	C$ millions		9,297.7	(14.8)	(39.1)	(108.4)	835.2	1,177.7	1,233.8	1,289.6	1,325.3	1,346.7	1,256.7	831.6	163.5
Federal Corporate Income Tax (15%)	C$ millions		1,419.0	—	—	—	125.3	176.7	185.1	193.4	198.8	202.0	188.5	124.7	24.5
Provincial Corporate Income Tax (12%)	C$ millions		1,135.0	—	—	—	100.2	141.3	148.1	154.7	159.0	161.6	150.8	99.8	19.6
Net Profit	C$ millions		6,743.5	(14.8)	(39.1)	(108.4)	609.7	859.7	900.6	941.4	967.4	983.1	917.4	607.0	119.4
After-Tax Cash Flow	C$ millions		6,934.8	(177.8)	(549.9)	(519.2)	1,063.1	1,082.8	1,070.2	1,064.5	1,066.6	1,053.3	975.1	648.4	157.7
Cumulative After-Tax Cash Flow	C$ millions			(177.8)	(727.8)	(1,246.9)	(183.8)	898.9	1,969.1	3,033.7	4,100.3	5,153.6	6,128.7	6,777.0	6,934.8
Income tax payable	C$ millions		2,554.2	—	—	—	225.5	318.0	333.1	348.2	357.8	363.6	339.3	224.5	44.2
Initial Capital	C$ millions	1,048.3	1,246.9	177.8	549.9	519.2									
Working Capital	C$ millions	-	-												
Sustaining Capital	C$ millions	139.5	213.9	—	—	—	102.7	25.9	18.7	22.4	11.2	15.0	8.0	10.0	0.0
Reclamation	C$ millions	19.0	48.0	—	—	—	—	—	—	—	—	—	—	—	48.0
Total Capital Costs	C$ millions	1,206.9	1,508.9	177.8	549.9	519.2	102.7	25.9	18.7	22.4	11.2	15.0	8.0	10.0	48.0
Discount factor				0.93	0.86	0.79	0.74	0.68	0.63	0.58	0.54	0.50	0.46	0.43	0.40
Pre Tax															
Cash Flow	C$ millions	6,055.2	11,216.3	(177.8)	(549.9)	(519.2)	1,295.5	1,655.8	1,658.8	1,669.9	1,683.7	1,674.7	1,553.7	1,032.0	239.2
Cumulative cash flow	C$ millions			(177.8)	(727.8)	(1,246.9)	48.6	1,704.4	3,363.2	5,033.1	6,716.8	8,391.5	9,945.2	10,977.1	11,216.3
NPV 8%	C$ millions		6,055.2												
Payback period	years		1.0				1.0								
IRR before tax	**%**		73.6												
After Tax															
Cash Flow	C$ millions	3,661.1	6,934.8	(177.8)	(549.9)	(519.2)	1,063.1	1,082.8	1,070.2	1,064.5	1,066.6	1,053.3	975.1	648.4	157.7
Cumulative cash flow	C$ millions			(177.8)	(727.8)	(1,246.9)	(183.8)	898.9	1,969.1	3,033.7	4,100.3	5,153.6	6,128.7	6,777.0	6,934.8
NPV 8%	C$ millions		3,661.1												
Payback period	years		1.2				1.0	0.2							
IRR after tax	%		56.8												

22.6 Sensitivity Analysis

The cash flow model was tested for sensitivity to variances in head grade, process recovery, uranium price, overall operating costs, overall capital costs and to Canadian to United States dollar exchange rate.

Figure 22-1 illustrates the results of the sensitivity analysis. The anticipated Project cash flow is most sensitive to the price of uranium, head grade and process recovery. Yellowcake is primarily traded in United States dollars, whereas capital and operating costs for the Project are primarily priced in Canadian dollars. Therefore, the Canadian to United States dollar exchange rate significantly influences Project economics.

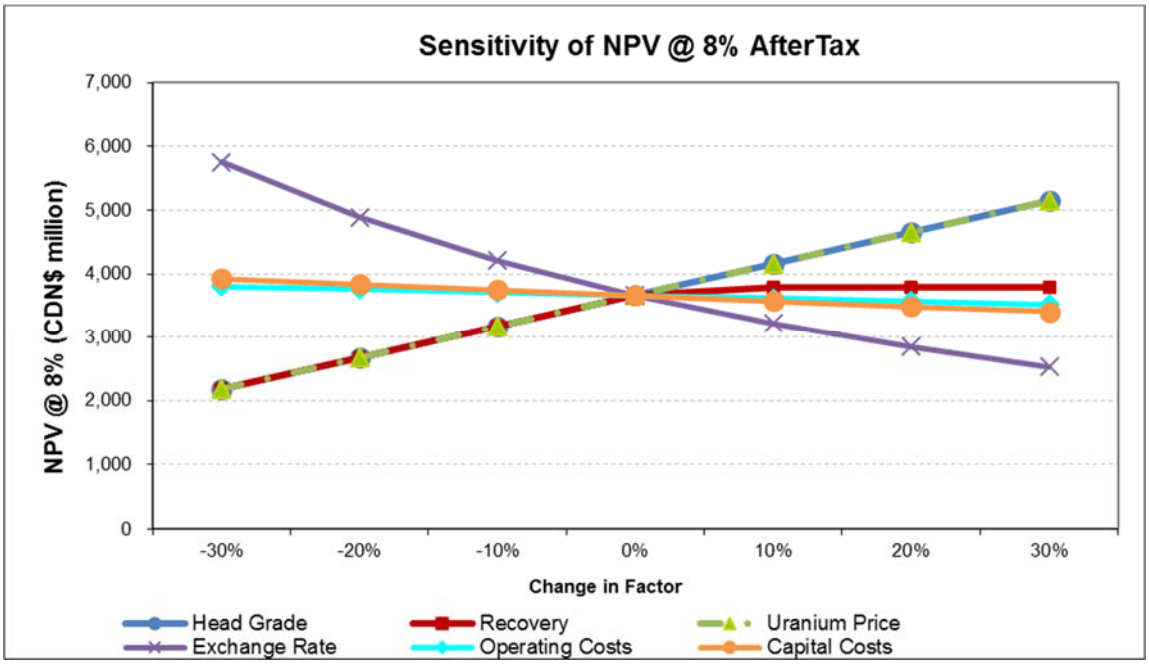
An extended sensitivity analysis was undertaken to evaluate the Project sensitivity to fluctuations in the uranium price. The results of the extended sensitivity analysis are displayed in Figure 22-2. This graph shows that the Project is robust to price changes, with negative economics only seen when the commodity price reaches US$10/lb U_3O_8.

22.7 Comments on Section 22

Under the assumptions presented in this Report, the Project shows positive economics.

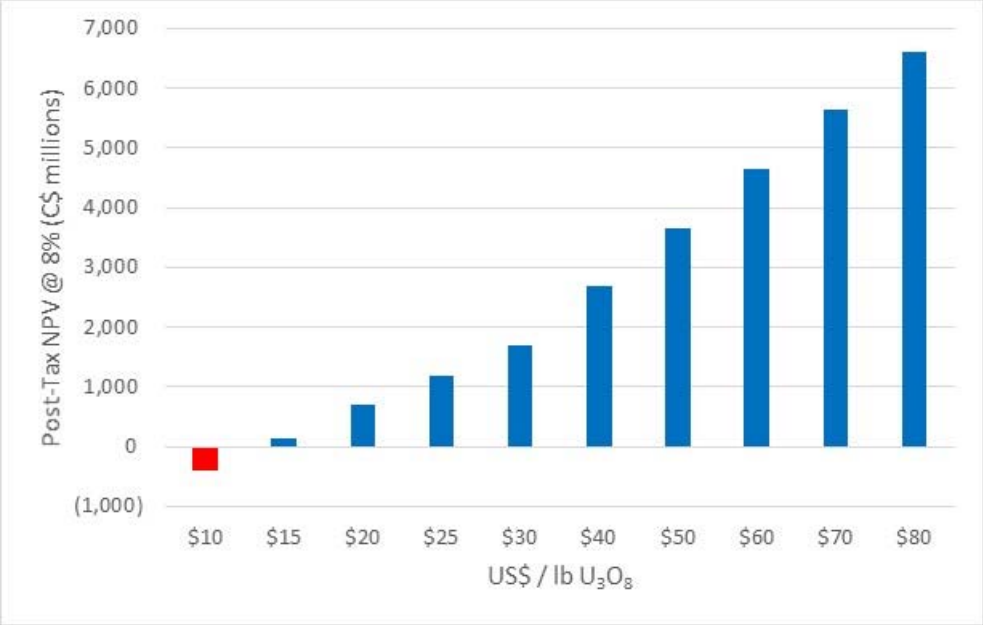
The anticipated Project cash flow is most sensitive to the price of uranium, head grade and process recovery. The Canadian to United States dollar exchange rate also exerts significant influence over the projected Project economics.

Figure 22-1: Sensitivity Analysis



Note: Figure prepared by Wood, 2018.

Figure 22-2: Project U Price Sensitivity



Note: Figure prepared by Wood, 2018.

23.0 ADJACENT PROPERTIES

This section is not relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Risks (NexGen/Wood)

A risk evaluation was completed NexGen and Wood at the Project level, reviewing issues identified during the PEA. The goal was to update the risk and identify any new mitigations that were either in place or that should be considered. Additionally, the group identified new risks that were not identified in the PEA report. A total of 10 significant risks were identified, five ranked as medium, and five as low. Following proposed mitigation measures, the risks were reclassified, with nine classified as low, and one as very low (Table 24-1).

24.2 Project Risks (RPA)

RPA independently conducted a risk analysis and identified risks associated with the technical and cost assumptions used. RPA classified these risks as low/moderate/high and commented on risk mitigation in the development and operating plan. In all cases, the level of risk referred to RPA's subjective assessment as to how the identified risk could affect the achievement of the Project objectives.

Identified risks and potential mitigation strategies are provided in Table 24-2. Three risks were considered to have a moderate ranking and a moderate chance of occurrence, two were considered moderate risks with a low probability of occurrence, one had a low risk but moderate probability of occurrence, and one was considered both low risk and low probability.

Table 24-1: Project Risk Evaluation (NexGen/Wood)

Risk Description	Classification	Existing / Proposed Strategies / Actions	Post Action
There is a risk that resource tonnes and grade estimates are less than expected	Low	Infill drilling is required in areas classified as Inferred. There is upside potential to increase resources in all directions.	Low
There is a risk the shaft sinking will encounter adverse ground conditions	Low	Pilot hole program underway. Conducting geotechnical and hydrogeological assessment around planned shaft locations. Temporary freeze wall and hydrostatic liner for first 150m to assist ground stability.	Very Low
There is a risk of poor ground conditions within the altered rock which will affect mining conditions	Low	Geotechnical drilling and analysis in progress to further refine ground support requirements.	Low
There is a risk that uranium recovery of the process plant is less than expected.	Low	Test work supports recovery assumption. Additional test work completed to allow optimization of flowsheet.	Low
There is a risk that placement of material in the Underground Processed Waste Management Facility will require more volume than budgeted in the PFS	Medium	Phase 2 of paste testing to be performed.	Low
There is a risk that the project will encounter environmental permitting delays	Medium	Project submissions and supporting studies Clearly define regulatory permitting timelines.	Low
There is a risk of radiation exposure to the workers	Medium	Continue radiological studies and design for radiation safety in FS and detailed design.	Low
There is uncertainty around the temporary artificial ground freezing required for shaft sinking the first 150 m	Medium	Shaft pilot hole work started. Estimate and design of the temporary ground freezing complete for PFS. Early engagement of shaft sinking contractor	Low
There is uncertainty around shaft sinking	Medium	Early engagement of shaft sinking contractor PFS design advancement, more robust assessment	Low
There is a risk that the cost of key materials and supplies will increase	Low	PFS design advancement, better understanding of operating consumables. Acid plant development.	Low

Table 24-2: Project Risk Evaluation (RPA)

Project Element	Issue	Risk Level	Risk Probability	Mitigation
Unknown mineralization	The shafts or lateral development could encounter unknown mineralization of varying grades. This could cause significant delays and cost overruns. In a worst-case scenario the continued intersection of mineralized material could cause substantial delays.	Moderate	Moderate	Conduct a sterilization diamond drill program that tests for mineralization in the areas of the shafts and lateral waste development.
Mineral Resources	The mine plan is based on Indicated Mineral Resources, with a drill hole spacing of approximately 25 m. Vein continuity between diamond drill holes could be less than what is predicted in the model.	Moderate	Low	It is recommended that parts of the deposit be drilled to the frequency of Measured Mineral Resources, to ensure that the vein continuity supports mine planning.
Temporary ground freezing	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 150 m down prior to the shaft pre-sink commencing.	Low	Low	Pilot hole drilling could be used to understand certain characteristics of the overburden, including salinity, temperature, and hydraulic conductivity. Once this data is obtained, temporary ground freezing requirements can be better defined.
Procurement	Lead times for certain shaft equipment are highly variable, and can often stretch several months. The current estimate assumes a "middle of the road" case in terms of lead times for certain critical items. If the industry experiences an upswing, the lead times could be extended, causing an overall delay to the project.	Moderate	Low	Ongoing lead-time awareness of certain key pieces equipment should be prioritized. Additionally, the used equipment market should be continually monitored to determine if there is availability for equipment to be refurbished.

Project Element	Issue	Risk Level	Risk Probability	Mitigation
Pre-Shaft Excavation Grouting Program	Currently, minimal cost and scheduling allowances have been made for a grouting program. The purpose of the grouting program would be to prevent the inflow of water into the shaft. There is a risk that grouting could be required, causing delays to the overall schedule. Further, grouting after water has already been encountered has a lower likelihood of success compared to grouting prior to ground excavation.	Moderate	Moderate	The pilot hole will assist in determining the necessity of a grouting program. Detailed packer testing and core logging should be undertaken on the shaft pilot hole.
UGTMF	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.
	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 performed a review of the opportunities that should be explored in future studies. These include:

- Assess the potential to use a mechanical solid/liquid separation step versus thickeners to reduce water consumption. This also provides an opportunity to reduce effluent treatment costs and environmental emissions
- Investigate the use of renewable energy such as wind/solar power as an addition to baseload power generation
- Evaluate the use of waste heat from the proposed acid plant to generate power, thereby reducing the number of generators required in the power plant
- Examine the use of scanners in ore sorting, so as to reduce the amount of waste brought to surface
- Review whether water consumption can be reduced by enhanced process water recycling
- Assess if the slurry to be pumped to the UGTMF can be made more dense, so as to reduce the number of storage cavities required, and minimize the amount of clean waste that would need surface storage
- Investigate the use of the most technologically advanced remote-controlled and autonomous equipment to improve Project economics, and to help lower radiological exposures
- Review whether VOD requirements can be lowered as a result of the remote-controlled/autonomous investigation, so as to reduce heating costs and airflow requirements.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

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

Mineral tenure is registered in the name of NexGen Energy Ltd. All claims are in good standing until at least 2035, and the claim that hosts the Arrow deposit, S-113927, is in good standing until 2039.

NexGen currently holds no surface rights in the Project area.

The Mineral Resources and Mineral Reserves do not occur within claims covered by the 2% NSR or 10% production carried interest and therefore the Arrow deposit is free of royalties.

25.3 Geology and Mineralization

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

The geological setting, mineralization style, and structural and stratigraphic controls are sufficiently well understood to provide useful guides to exploration and Mineral Resource estimation.

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

Exploration completed to date has resulted in delineation of the Arrow deposit and a number of exploration targets.

The mineralization in the Arrow deposit is sub-vertical and true width is estimated to be from 30% to 50% of reported core lengths based on currently-available information.

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.

RPA considers the QA/QC protocols in place for 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.

No limitations were placed on RPA's data verification process. RPA considers the resource database reliable and appropriate to support a Mineral Resource estimate.

A major drilling program is planned to commence in late December 2018, and will comprise infill drilling in the Arrow deposit area and evaluation of regional targets.

25.5 Metallurgical Testwork

The metallurgical test program included a bench test program, a pilot plant and paste backfill testing.

The average recovery estimate used in the 2018 Pre-feasibility Study was determined from the pilot testing.

No major deleterious elements or elemental concentrations have been identified to date.

25.6 Mineral Resource Estimates

The Mineral Resource estimate is reported using the 2014 CIM Definition Standards. The effective date of the Mineral Resource estimate is May 25, 2018. Estimated block model grades are based on chemical assays only. The Mineral Resources were estimated by NexGen and audited by RPA.

In RPA's opinion, the estimation methodology is consistent with standard industry practice and the Indicated and Inferred Mineral Resource estimates for Arrow and South Arrow are considered to be reasonable and acceptable.

Factors that may affect the resource estimate include: 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 reasonable prospects for eventual economic extraction; assumptions as to social, permitting and environmental conditions; and additional infill or step out drilling or results obtained from the planned 2019 drill program.

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 that are not discussed in this Report.

25.7 Mineral Reserve Estimates

The Mineral Reserve estimate is reported using the 2014 CIM Definition Standards. The effective date of the Mineral Reserve estimate is May 25, 2018.

The vertical extent of the Mineral Reserves extends from approximately 355 m below surface to 655 m below surface, and are limited to the A2 and A3 veins within the Arrow deposit. Mineral Reserves are reported using a cutoff of 0.25% U_3O_8, assumptions of transverse stope and longitudinal retreat stope mining methods, an average mining rate of 1,100 t/d, dilution in external stopes of 31%, fill dilution of 13%, and an overall mining recovery of 95% for longhole mining and ore development.

Factors that may affect the Mineral Reserve estimate include: 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 can be permitted; assumptions as to social, permitting and environmental conditions; and additional infill or step out drilling or results obtained from the planned 2019 drill program.

25.8 Mine Plan

The Arrow deposit is planned to be accessed using two shafts. The first shaft will be used as a production shaft, and for transportation of personnel and materials into the mine, the second shaft will be used as an exhaust ventilation shaft and an emergency escapeway. The ventilation system is designed as a predominately negative or "pull" system.

Eleven level spacings at Rook I mine are planned at 30 m intervals, sill to sill. In addition to the lateral development, there will be an internal ramp system that will connect all mining levels. 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 vein, for a total of four separate production areas. Transverse longhole mining will be done using primary and secondary stoping methods to avoid

leaving pillars. The order in which stopes are extracted is largely driven by the head grade, with the overarching goal of achieving annual production of 30 Mlbs U_3O_8.

The 2018 Pre-feasibility Study proposes that an underground tailings management facility (UGTMF) will be used to store tailings and gypsum. A portion of the tailings will be used to generate paste backfill for use in the underground 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 underground and active reclamation of the mine.

25.9 Recovery Plan

The planned process plant will use conventional technology for the uranium industry, consisting of grinding, leaching, liquid–solid separation via counter current decantation, solvent extraction, yellowcake precipitation, yellowcake packaging, and a paste tailings backfill plant.

25.10 Infrastructure

The infrastructure contemplated for the Project is conventional to the uranium industry 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 site.

25.11 Environmental, Permitting and Social Considerations

25.11.1 Environmental Considerations

Environmental baseline studies have been undertaken or are underway.

25.11.2 Waste Rock Management Facility

Facility design is applicable for LOM storage requirements. Both lined and unlined facility areas are envisaged, for storage of mineralized material that will not meet economic criteria and non-mineralized waste rock respectively.

25.11.3 Water Management

Water management structures include a storm water runoff pond, dike, and six process ponds.

For the 2018 Pre-feasibility Study design, surface water runoff from the general site footprint, the waste rock pile, and surrounding undisturbed areas is considered to be non-contact water, but is still planned to be captured and contained.

25.11.4 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.

25.11.5 Permitting Considerations

There are several Federal and Provincial regulatory approvals required for a new uranium mine and mill development.

The CNSC and MOE will coordinate the EA requirements for each jurisdiction. The Project will also have to consider regulation compliance requirements under approximately 12 separate Federal acts, and about 27 Provincial acts.

Permits are in place for the exploration work programs for 2018, and applications for permit renewals have been made in support of the planned 2019 drilling.

25.11.6 Social Considerations

NexGen has developed a Public and Indigenous Engagement Plan that is specific to Project.

25.12 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 Arrow mine 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.

25.13 Capital Cost Estimates

The estimate meets the classification standard for a Class 4 estimate as defined by AACE International, and has an intended accuracy of ±30%. The estimate is reported in Q3 2018 Canadian dollars.

Total capital costs, including initial, sustaining and reclamation costs, total C$1,508.9 million.

25.14 Operating Cost Estimates

Operating cost estimates were developed to show annual costs for production. Unit costs are expressed as C$/tonne processed and C$/lb U_3O_8.

LOM operating costs are estimated to be C$1,328.3 million.

25.15 Economic Analysis

The study is based on an assumption of processing of 3,433 kt grading 3.09% U_3O_8 over a nine-year mine life to produce 228.4 Mlb of recovered yellowcake, using a metallurgical recovery forecast of 97.6%.

On a pre-tax basis, the NPV8% is C$6,055.2 million, the IRR is 73.6%, and the assumed payback period is one year. On a post-tax basis, the NPV8% is C$3,661.1 million, the IRR is 56.8% and the assumed payback period is 1.2 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.

25.16 Risks and Opportunities

Two separate risk evaluations were undertaken, one by NexGen/Wood, and one by RPA.

- The main NexGen/Wood risks related to: assumptions as to volumes needed in the UGTMF; environmental permitting delays; worker radiation exposure; shaft sinking uncertainties primarily related to temporary artificial freezing down 150 m; and increases in costs to key materials and supplies when compared to the 2018 Pre-feasibility Study estimates
- The main RPA risks related to: if shafts or lateral development encounter unknown mineralization of varying grades, this could cause delays, grouting may be required to address shaft water inflows, and the UGTMF may not be viewed favourably by regulatory bodies, as it is a new concept for uranium mines; however, it facilitates permanent storage of process tailings underground and active reclamation of the mine.

NexGen performed an opportunity analysis. Opportunities that were recognized included: water useage reduction and recycling, reduction in the number of generator sets needed in the power plant through use of renewable energy or waste heat from the acid plant; using leading-edge remote and autonomous equipment to improve project economics, lower radiological exposures, optimize VOD and heating needs; use of ore-sorting technology to reduce waste sent to surface; and slurry thickening to reduce the number of required storage cavities and the amount of clean waste needed to be stored on surface.

25.17 Conclusions

Under the assumptions presented in this Report, the Project shows positive economics.

26.0 RECOMMENDATIONS

26.1 Introduction

The recommendations proposed are presented as a two-phase work program. Portions of the second recommended work phase are dependent on information generated in the first phase. The Phase 1 recommendations are projected to require a budget of $45 million to complete. The Phase 2 recommendations are estimated at $6.5 million.

26.2 Phase 1

The Phase 1 recommendations consist of a 125,000 m drill program.

Infill drilling is planned in the high-grade portion of the Arrow deposit, with the intent of infilling the current drill spacing to the drill support spacing identified for classification of Measured Mineral Resources. The program will consist of 71,000 m, and is designed to provide support for potentially upgrading Mineral Resources that are currently classified as Indicated to Inferred.

Additional infill drilling, totalling 54,000 m, is planned to infill areas where Inferred Mineral Resources have been estimated, to tighten the drill spacing to that used to classify Indicated Mineral Resources. The program is designed to provide support for potentially upgrading Mineral Resources that are currently classified as Indicated to Inferred.

A portion of the planned drilling is aimed at providing further characterization of the geotechnical and hydrological/hydrogeological setting in the planned underground operations and UGTMF areas. About 12,500 m of drilling has been allocated to this investigation.

The drill program is estimated to cost $45 million, based on an all-in drilling cost of $360/m assuming completion of the entire planned 125,000 m program.

The drill program commenced on 5 December 2018.

26.3 Phase 2

The following work program is proposed for the second work phase.

26.3.1 Geotechnical Studies

The following studies are proposed:

- Using the information generated from the first work phase targeting geotechnical and hydrological/hydrogeological rock mass conditions in the planned underground operations and UGTMF areas, confirm ground support requirements for the underground mine and optimum shaft locations, and placement of the UGTMF
- Confirm the suitability and design requirements for the selected sites for the WRF and ore stockpile, processing facilities, and infrastructure and conduct a site-specific hazard assessment for these locations.

This program is estimated at $1.0 million.

26.3.2 Hydrological and Hydrogeological Studies

The following studies are proposed:

- Using the information generated from the first work phase targeting geotechnical and hydrological/hydrogeological considerations, complete hydrogeological studies in order to understand the impact of groundwater on the underground mine, mine dewatering requirements, ground support requirements and to understand the long-term influence of backfilling underground on the environment
- Complete hydrological studies in order to fully understand the water flow rates and to assess the water management and water treatment requirements for the site.

This program is estimated at $1.5 million.

26.3.3 Environmental Studies

The following studies are proposed:

- Complete environmental baseline studies including terrestrial and aquatic flora and fauna surveys, soils, sediment, and hydrogeology and testing such as geochemical and water analyses in order to support, with a high degree of confidence, that the activities associated with the mine development can be carried out in a manner that will not degrade the environment.

This program is estimated at $4.0 million.

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