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Exhibit 96.1

ANDERSON URANIUM PROJECT
INITIAL ASSESSMENT
US SEC Subpart 1300 Regulation S-K Report
Yavapai County, Arizona, USA

PREPARED FOR:
URANIUM ENERGY CORPORATION

AUTHORED BY:
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JULY 1, 2022

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1.0 EXECUTIVE SUMMARY

The Anderson Project is located in Yavapai County, west-central Arizona, approximately 75 miles northwest of Phoenix and 43 miles northwest of Wickenburg (latitude 34°18'29" N and longitude 113°16'32" W, datum WGS84) (Figure 4-1). The general area is situated along the northeast margin of the Date Creek Basin. The Anderson Project is located on the south side of the Santa Maria River, approximately 13 miles west of State Highway 93. The Anderson Project occupies part or all of Sections 1 and 3, 9 through 16, 21 through 27, and 34 of Township 11 North, Range 10 West and portions of Sections 18, 19, and 30 of Township 11 North, Range 9 West of the Gila and Salt River Base Meridian. The planned mining method for the Project is by a combination of open-pit and underground conventional mining.

The Anderson Project covers 8,268 acres (12.9 square miles) and is comprised of 386 contiguous, unpatented lode mining and placer claims and one Arizona State land section. It is located in western Yavapai County, approximately 75 miles northwest of Phoenix. The northern section of the Anderson Project area holds the open-pit resource, and the adjacent southern section holds the underground resource.

The Anderson Project is located along the northeast margin of the Date Creek Basin of the Basin and Range Province of the western United States. Uranium mineralization at the Anderson Project is strata bound and occurs exclusively in the sequence of Miocene-age lacustrine lakebed sediments. The lacustrine sediments unconformably overlie the andesitic volcanic unit over most of the Anderson Project.

The area was discovered by radiometric surveys at outcrop in the 1950s, and 10,758 tons of material containing 33,230 pounds of uranium at a grade of 0.15 %U_3O_8 was mined and shipped to AEC buying stations prior to 1960.

Subsequently the area was explored extensively by,

- Getty Oil Company, 1967-1968, limited drilling;
- Urangesellschaft U.S.A Inc., 1973-1982, 319 rotary and 33 core drill holes;
- Minerals Exploration Company, 1974-1980, 970 rotary and 84 cores holes; and
- Concentric Energy Corp., 2006, 24 RC drill holes and 1 core hole.

Within the Anderson Project area, drill data from 1,175 drill holes were used in the current mineral resource estimate, including hole location and radiometric equivalent data in 0.5 foot downhole increments. Uranium Energy Corp has not completed any drilling on the Anderson Project.

1.1 Interpretations and Conclusions

This Initial Assessment for the Anderson Project has been prepared in accordance with the regulations set forth in S-K 1300 (Part 229 of the 1933 Securities Act). Its objective is to disclose the mineral resources at the Anderson Project.

Based on the density of drilling, continuity of geology and mineralization, testing, and data verification the mineral resource estimates meet the criteria for indicated mineral resources as summarized herein.

Estimated indicated mineral resources are summarized in Table 1-1 at a 0.02% eU3O8 grade cutoff and a 0.1 ft% GT cutoff. Mineral resources were estimated separately for each mineralized zone. The total contained mineralized material was first estimated. Then reasonable prospects for economic extraction were applied. Indicated Mineral Resources are reported within the 2014 PEA Conceptual Mining envelopes as having reasonable prospects for economic extraction and represent an 18% reduction from the estimate of total mineralized material.

Mineral resources are not mineral reserves and do not have demonstrated economic viability in accordance with CIM standards. However, considerations of reasonable prospects for eventual economic extraction were applied to the mineral resource calculations herein.

Table 1-1: Indicated Mineral Resources

Mineral Resource Estimates (0.1% Sum GT Cutoff)	**Tons (millions)**	**Average Sum Thickness (ft)**	**Average Grade (%eU3O8)**	**Pounds eU3O8 (millions)**
Resource Zone A				
Reasonably Extractable Indicated Resource	0.862	3.8	0.111	1.907
Resource Zone B				
Reasonably Extractable Indicated Resource	7.347	9.5	0.108	15.816
Resource Zone C				
Reasonably Extractable Indicated Resource	6.211	10.4	0.094	11.730
Resource Zone D				
Reasonably Extractable Indicated Resource	0.760	3.2	0.093	1.421
Resource Zone E				
Reasonably Extractable Indicated Resource	0.911	7.6	0.060	1.095
Resource Zone F				
Reasonably Extractable Indicated Resource	0.084	4.6	0.051	0.086
ALL ZONES GRAND TOTALS				
Extractable Indicated Resource	**16.175**	**8.2**	**0.099**	**32.055**

Notes:
1. Mineral Resources are not mineral reserves and do not have demonstrated economic viability.
2. Economic factors have been applied to the estimates in consideration of reasonable prospects for economic extraction.
3. Tools may not sum due to rounding.

1.2 Recommendations

The authors have provided recommendations and costs estimates in Section 23 relative to exploration, environmental studies, project design and feasibility, and permitting for reference. At this time, only the drilling exploration program is recommended as the other work items are based on the successful completion of this work item.

The recommended drilling and assaying will attempt to confirm historic results and upgrade the classification of resources in some areas. Chemical assays will also be used to confirm historic results and determine the propriety of the disequilibrium correction applied to current eU_3O_8 grades.

The following work items related to additional exploration are recommended for the Anderson Project:

Table 1-2: Exploration Budget

Item	Cost (USD)
Permitting and reclamation	$100,000
5 diamond drill holes (300 ft average, 1,500 ft total)	$200,000
45 RC and rotary holes (600 ft average 27,000 ft total)	$800,000
Site supervision including geological services and surveying	$200,000
Geophysical Logging 50 holes	$100,000
Assay of core and RC chips (2,000 samples by ICP-MS)	$150,000
Metallurgical heap leach testing	$200,000
Resource model update, reporting and preparation of PEA	$200,000
Road maintenance	$50,000
Total	**$2,000,000**

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1.3 Risks

Neither a Pre-feasibility nor a Feasibility study to estimate detailed capital and operational expenditures to the Anderson Project has been completed. Since these studies have not been completed for the Anderson Project, there has not been a formal demonstration of economic and technical capability.

Technical risks related to the Anderson Project exist, as large-scale conventional mining methods have not been demonstrated in the area. However, these risks are considered moderate as conventional mining and recovery methods are proven in similar sandstone hosted environments.

Risks related to permitting and licensing and public acceptance of the Anderson Project exist.

Mineral tenor is based on the 1872 mining law which establish mining claims in the US. There is a political risk that the changes in the 1872 act may affect the mineral tenor.

Other risk factors are typical for similar mining projects, including, without limitation:

- risks associated with mineral resource estimates, including the risk of errors in assumptions or methodologies;
- geological, technical and processing problems, including unanticipated metallurgical difficulties, less than expected recoveries, ground water control and other factors relating to the ISR mining methodology;
- risks associated with labor costs, labor disturbances, and unavailability of skilled labor;
- risks associated with the availability and/or fluctuations in the costs of raw materials and consumables used in the production processes;
- risks associated with environmental compliance and permitting, including those created by changes in environmental legislation and regulation, and delays in obtaining permits and licenses that could impact expected mineral extraction and recovery levels and costs; and
- Estimation of costs and uranium price for the purposes of constraining the mineral resource estimate, based on reasonable prospects for economic extraction.

Readers are cautioned that any estimate of forward cost or commodity price is by its nature forward-looking. It would be unreasonable to rely on any such forward-looking statements and information as creating any legal rights. The statements and information are not guarantees and may involve known and unknown risks and uncertainties, and actual results are likely to differ (and may differ materially) and objectives and strategies may differ or change from those expressed or implied in the forward-looking statements or information as a result of various factors. Such risks and uncertainties include risks generally encountered in the exploration, development, operation, and closure of mineral properties and processing facilities. Forward-looking statements are subject to a variety of known and unknown risks and uncertainties.

2.0 INTRODUCTION

2.1 Registrant

This initial Assessment (IA) was prepared for Uranium Energy Corp (UEC) on the Anderson Project (the Project), located in Arizona, USA. UEC is incorporated in the State of Nevada, with principal offices located at 500 North Shoreline Boulevard, Suite 800N, Corpus Christi, Texas, 78401, and at 1030 West Georgia Street, Suite 1830, Vancouver, British Columbia, Canada, V6E 2Y3.

This Technical Report Summary (TRS) was prepared for UEC by BRS Engineering (BRS) under the supervision of Douglas Beahm, PE, PG and co-authored by Clyde Yancey, PG, Vice President of Exploration, UEC.

Figure 2-1: Project Location Map

2.2 Terms of Reference

In May of 2011, UEC entered into a merger agreement with Concentric (the Merger). According to the terms of the agreement, UEC Concentric Merge Corp., a wholly-owned subsidiary of UEC, was the surviving corporation of the Merger and was vested with all of Concentric's assets and property including the Project.

The Project covers 8,268 acres (12.9 square miles) and is comprised of 386 contiguous, unpatented lode mining and placer claims and one Arizona State land section. It is located in western Yavapai County approximately 75 miles northwest of Phoenix. The northern section of the Project area holds the open-pit resource, and the adjacent southern section holds the underground resource.

2.3 Information Sources and References

The information and data presented in this IA was gathered from various sources described herein. UEC possesses the original drill data, from which a drill hole database has been developed and verified. The uranium mineral resource estimate was based on a total of approximately 655,000 feet of drilling from 1,464 holes, including 1,054 holes drilled by Minerals Exploration Company (MinEx), 385 holes by Urangesellschaft U.S.A. Inc. (Urangesellschaft) and 25 holes drilled by Concentric Energy Corp. (Concentric) as of 15 April 2012, the effective date for this estimate. UEC has not completed drilling on the Project.

Units of measurement unless otherwise indicated are feet (ft), miles, acres and pounds (lbs). Uranium production is expressed as pounds U_3O_8, Radiometric equivalent uranium grade is expressed as $\%eU_3O_8$. Unless otherwise indicated, all references to dollars ($) refer to United States currency.

2.4 Inspection on the Property by Each Qualified Person

Mr. Beahm was onsite the Project site on the 17th and 18th of December 2013. During this period, Mr. Beahm inspected uranium mineralization in outcrop and mineralized stockpiles from past mining operations and collected samples of mineralized material for metallurgical testing, which was subsequently completed by Resource Development Inc. (RDI) of Wheat Ridge, Colorado, under the direction of Dr. Terrence McNulty.

In addition, Mr. Beahm examined several drill sites and mineral claim monuments at the site and examined available core and drill cuttings stored at UEC's facility in Wickenburg, Arizona.

Mr. Yancey was onsite on the 24th of February 2020. During this time, Mr. Yancey performed a reconnaissance of the Project area and visited the UEC office facility in Wickenburg, Arizona.

2.4.1 QP Qualifications

Mr. Beahm is the independent Qualified Person (QP) and co-author of this TRS. Mr Beahm is responsible for the preparation of Sections 4, 6, 7, 9, 11 and contributed to portions of Sections 1, 2 and 22-25 of this IA, which includes the mineral estimates herein. Mr. Beahm is a QP under the S-K 1300 standards responsible for the content of this IA and a Professional Engineer, Professional Geologist, and a SME Registered Member with 48 years of professional experience.

Mr. Yancey is the Vice President of Exploration for UEC, a QP and co-author of this TRS. Mr. Yancey responsible for the preparation of section 3 and 5 and contributed to portions of 1, 2 and 22-25 of this IA. Mr. Yancey is a QP under the SK-1300 standards responsible for the content of this IA and a Professional Geologist with 44 years of professional experience.

2.5 Previous Technical Report Summaries

UEC has not previously filed an IA on the Project under SK-1300 standards.

Previous technical reports prepared under Canada's NI 43-101 and CIM guidance include:

- "Mineral Resources of the Anderson Uranium Property Yavapai County, Arizona, USA" prepared for Global Uranium Corporation by Agapito Associates, Inc., July 1, 2010.

- "Technical Report and PEA of the Anderson Uranium Project, Yavapai County, Arizona, USA" prepared of UEC by BRS Inc., T. P. McNulty and Associates, BD Resource Consulting Inc., and SIM Geological Inc., July 31, 2014.

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3.0 PROPERTY DESCRIPTION

3.1 Property Description and Location

The Project is located in Yavapai County, west-central Arizona, approximately 75 miles northwest of Phoenix and 43 miles northwest of Wickenburg (latitude 34°18'29" N and longitude 113°16'32" W, datum WGS84) (Figure 4-1). The general area is situated along the northeast margin of the Date Creek Basin. The Project is located on the south side of the Santa Maria River, approximately 13 miles west of State Highway 93. The Project occupies part or all of Sections 1 and 3, 9 through 16, 21 through 27, and 34 of Township 11 North, Range 10 West and portions of Sections 18, 19, and 30 of Township 11 North, Range 9 West of the Gila and Salt River Base Meridian.

3.2 Mineral Rights

The Project comprises the majority of the claim positions historically held by MinEx and Urangesellschaft in the 1970s. In 1995, Hanson Exploration, Inc. (Hanson) of Phoenix, Arizona consolidated these claim positions. By 1998, Hanson dropped the claims and Concentric restaked them in 2001. Concentric's claim holdings consisted of 370 contiguous, unpatented, lode mining claims and 9 placer claims that superimpose part of the lode claim block. On 15 April 2010, Global Uranium Corporation (Global) entered into an option and joint venture agreement to acquire the Project property. Under the terms of the agreement, Global would earn a 70% in the Project property over a six-year period by paying $80,000 on signing and issuing 11.3 million shares incrementally over the six-year period. Global had to spend at least $2 million on the Project property before the fifth year of the agreement. After completing their commitment to acquire a 70% interest, Global had the option to acquire the remaining 30% by issuing an additional 2.7 million shares.

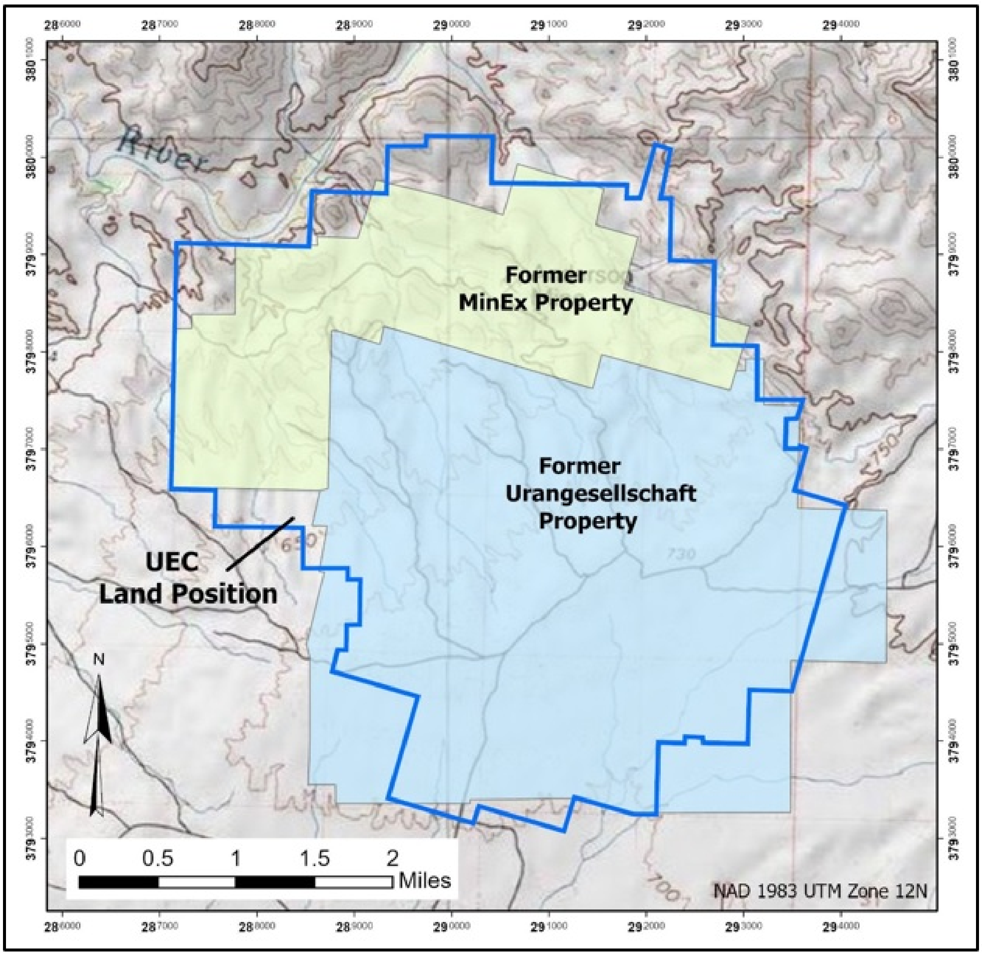
In May of 2011, UEC entered into a merger agreement with Concentric. According to the terms of the agreement, UEC Concentric Merge Corp., a wholly-owned subsidiary of UEC, was the surviving corporation of the Merger and was vested with all of Concentric's assets and property.

UEC staked an additional 89 lode mining claims in June and July of 2011, and acquired mineral leases on Arizona State Sections 16 (exploration permit 08-115674) and 36 (exploration permit 08-115675) of Township 11 North, Range 10 West, Gila and Salt River Base Meridian (Figure 4-2). Subsequently, in 2018, UEC dropped 73 claims and State Section 36 in order to streamline their land position. The entire claim block and the State mineral lease comprise an area of approximately 8,268 acres (12.9 square miles).

Unpatented mining claims, either lode or placer, are located under the authority of the Mining Law of 1872 (the Mining Law) on Federal lands administered by the Bureau of Land Management (BLM). Under the Mining Law, the locator has the right to explore, develop and mine minerals on unpatented mining claims without paying production royalties to the Federal Government. Claim maintenance fees of $165 per claim are due on September 1 of each year.

Arizona State mineral leases are held with an exploration permit. There is a $500 annual fee for the exploration permit, plus $1 per acre rental for the first five years. For the first two years, there is also a minimal exploration expenditure requirement of $10 per acre per year. For years three through five, there is a $20 per acre minimum.

Figure 3-1: Anderson Project Area and Mineral Tenor.



3.3 Surface Rights

All of the Project mining claims are on public lands administered by the BLM. Arizona State sections are administered by the Arizona State Land Department; UEC has surface rights on these lands as outlined in their exploration permits. Under BLM regulations, surface use for the development and exploitation of mineral resources is permitted, subject to environmental permitting requirements, including the use of the surface for development of access and infrastructure, mine waste and mineral tailings disposal.

3.4 Mineral Exploration Permit Requirements

Exploration and mining activities for the mining claims of the Project are administrated by the BLM, Kingman Field Office. Exploration drilling and associated activities require an exploration permit, and a reclamation bond must be posted. Exploration and mining activities on Arizona State land are administrated by the Arizona State Land Office. This Project was drilled as recently as 2006, and it is not expected that any of these requirements will have an effect on the ability to conduct exploration activities. UEC has exploration permits on the two State sections. In order to conduct the recommended program for BLM ground, as outlined in Section 26, UEC needs to submit a plan of operations, a minimal impact exploration permit and a special use permit. There are no royalties.

3.5 Environmental Liabilities

The Authors are not aware of significant environmental liabilities on the property. However, it is important to note that 195 acres in the northern part of the Project area were classified as “disturbed” by the BLM. The disturbed area is a result of minor production dozer cuts from surface mining in the 1950s.

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4.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Physiography

The Project is located in the northeast portion of the Date Creek Basin. The basin consists of low undulating terrain, centrally dissected by Date Creek Wash. The site lies along the south bank of the Santa Maria River, which runs along the northern edge of the basin.

The majority of the Project drains northward to the Santa Maria River. Headward erosion of these tributaries has produced a series of sub-parallel gullies and ridges trending north to northwest. The southern quarter of the Project drains southward to Date Creek Wash. Both the Santa Maria River and Date Creek Wash are generally intermittent. Significant flows can occur in the Santa Maria River during winter and spring run-off months. Elevations above sea level are between 1,700 ft and 2,400 ft. Maximum local topographic relief at the site is approximately 700 ft.

Vegetation on the property is typical of the Sonoran Desert of central Arizona and consists predominately of Joshua trees, palo verde bushes, saguaro, cholla, ocotillo, creosote bushes and desert grasses. Fauna includes jackrabbits, rattlesnakes, roadrunners, desert tortoise, various lizards and, less commonly, mule deer, wild burros and mules (Hertzke, 1997).

The alluvial valley of the Santa Maria River varies substantially in width and depth to bedrock. The volume of alluvium, and particularly the depth of the material, influences the proportion of surface flow to underflow in the river valley. The groundwater in the alluvium consists of underflow that is forced toward the surface as the depth of the alluvium decreases (MinEx, 1978b).

4.2 Accessibility and Local Resources

The Project is accessed by paved, all-weather gravel and dirt roads. The property is reached by taking the Alamo Lake turnoff, located approximately 21 miles northwest of Wickenburg on Arizona State Highway 93 (Joshua Tree Parkway), then driving 0.25 miles north of mile marker 179, and then following the Alamo Road for 5.8 miles to the Pipeline Ranch Road turnoff. The road passes through the Pipeline Ranch, located in the bottom of Date Creek Wash and continues for approximately 6.3 miles to FR 7581. The Project property boundary is located 1.4 miles north on FR 7581. There are alternate dirt roads, including a 15-mile primitive road from Highway 93 over Aso Pass (2,900 ft elevation) (Figure 4-1).

4.3 Climate

The climate is arid, with hot summers and mild winters. Annual rainfall averages 10 to 12 inches with rain showers from January through March and thunderstorms during the summer months. Snowfall is rare. On average, temperatures range between a low of 31°F during winter months and a high of 104°F during summer months. Temperature extremes of 10°F in winter and 120°F in summer have been recorded. The climate is favorable for year-round mining operations and requires no special operational or infrastructure provisions that relate to weather.

4.4 Infrastructure and Local Resources

The Project area is undeveloped, with the exception of various access and drill roads and various water wells previously constructed. No utilities exist on or adjacent to the Project area. A transmission power line runs northwest-southeast along Highway 93, approximately 8 miles to the east; however, direct access to the power line may be obstructed by the Arrastra Mountain Wilderness and Tres Alamos Wilderness located between the power line and the Project. The construction of a power line would require routing along one of the existing road corridors, a distance of 16.2 miles to the Project boundary.

Various water wells exist on and near the Project that can support large-scale mining operations. There is plenty of usable land space to locate processing plants, heap leach pads, tailings storage areas, waste disposal areas and other infrastructure development associated with large-scale mining. The Project includes most of a 195-acre area designated by the BLM as "disturbed" resulting from surface mining in the 1950s. It may be possible to expedite the permitting process for future metallurgical exploration and mining activities, including waste disposal, within the disturbed area.

The nearest town is Congress (population 1,700) located 32 road miles to the east. The nearest major housing, supply center and rail terminal is in Wickenburg (population 6,363) located approximately 43 miles from the Project by road. Phoenix (population 1.45 million), approximately 100 miles to the southeast by road, is the nearest major industrial and commercial airline terminal. Kingman (population 24,000) is located approximately 110 miles to the northwest by road. UEC's surface rights encompass 15.4 square miles; this is sufficient for the surface structures associated with any proposed mining operation.

Arizona currently has a number of operating open pit mines and, historically, has had a number of operating underground mines. As a result, personnel with the required skill set exist in the State.

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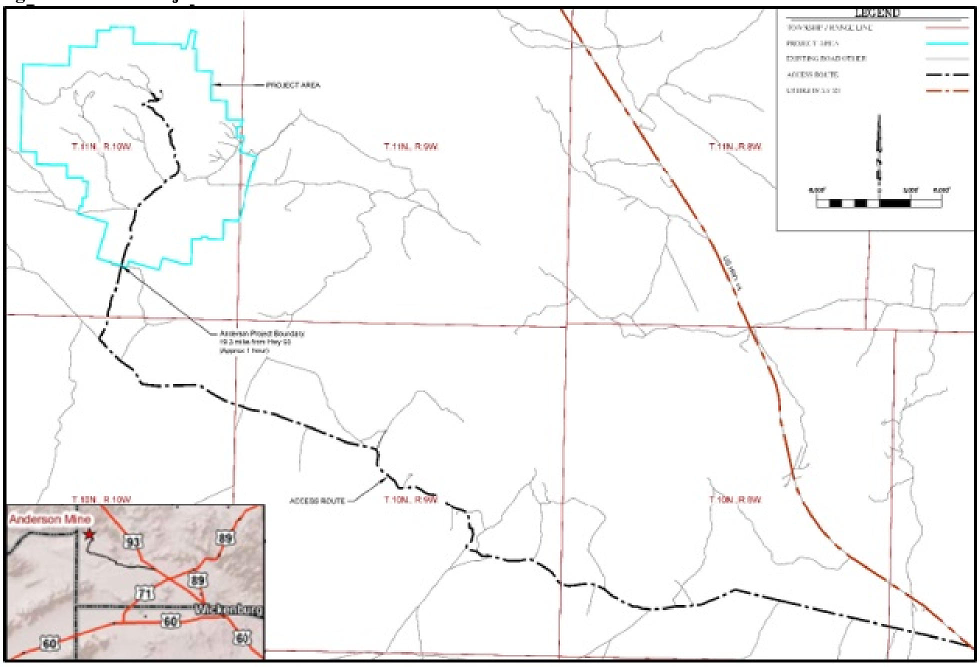
Figure 4-1: Project Access



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5.0 HISTORY

5.1 Prior Ownership and Exploration Activities

In January 1955, T.R. Anderson of Sacramento, California detected anomalous radioactivity in the vicinity of the Project using an airborne scintillometer. After a ground check revealed uranium oxide in outcrop, numerous claims were staked. The "Anderson Mine," as the operation was known at the time, was drilled and mined by Mr. Anderson (MinEx, 1978a). Work between 1955 and 1959 resulted in 10,758 tons that averaged 0.15% U3O8, and 33,230 pounds U3O8 (Table 6.1) were shipped to Tuba City, Arizona for custom milling (Gilbride et al, 2010). In 1959, production stopped when the Atomic Energy Commission (AEC) ended the purchasing program.

Table 5-1: Historical Production at the Anderson Mine (Agapito, 2010)

Year	Tons	Grade (%U3O8)	Pounds (U3O8)
1955	9	0.56	101
1956	31	0.21	130
1957	3,614	0.19	14,043
1958	725	0.27	3,928
1959	6,379	0.12	15,028
Totals	10,758	0.15	33,230

During 1967 and 1968, Getty Oil Company (Getty) secured an option on claims in the northern portion of the Project. Some drilling and downhole gamma logging was conducted during the option period, but this failed to locate a sizeable uranium deposit. In 1968, Getty dropped their option.

In 1974, the increasing price of uranium created a renewed interest in the vicinity of the Project. Following a field check and an evaluation of the 1968 Getty drill data, MinEx optioned the northern portion of the current Project (MinEx, 1978a).

In 1975, MinEx purchased the northern portion of the current Project after a 53-hole, 5,800 m (19,000 ft) drilling program on 250 m centers confirmed a much greater uranium resource potential than had been interpreted from the 1968 Getty gamma log data. Further exploration work, consisting of a 180-hole, 22,555 m (74,000 ft) drill and core program on 120 m centers was conducted from November 1975 through February 1976 to further delineate the uranium resources (MinEx, 1978a). By 1980, MinEx had completed a total of 1,054 holes by rotary and core drilling.

In 1977, the Palmerita Ranch, located 11 km west of the deposit along the Santa Maria River, was acquired by MinEx to provide a water source for the operations in the event that closer sources proved inadequate. Based on favorable economics, indicated in a Preliminary Feasibility Study completed by Morrison-Knudsen Company, Inc. in December 1977, a detailed Final Feasibility Study was undertaken early in 1978 to evaluate the MinEx holdings on the northern portion of the current Project (MinEx, 1978a-c).

In 1973, Urangesellschaft expressed an interest in the former Anderson Property. Urangesellschaft located a claim block, “Date Creek Project,” on the down-dip extension of the mineralization immediately to the south of MinEx’s claims. In 1973 to 1982, subsequent drilling programs delineated mineralization from a total of 352 drill holes with 122,744 m (402,773 ft) of rotary and core drilling (Hertzke, 1997).

Depressed uranium prices stalled exploration activities until 1995 when Hanson consolidated portions of the former MinEx and Urangesellschaft claims under single ownership. Hanson dropped the claims by 1998. In 2001, Concentric restaked the claims and controlled ownership until May, 2011. In 2006, Concentric drilled 24 reverse-circulation holes and one core hole on the MinEx portion of the Project to confirm the reproducibility and authenticity of the historical MinEx exploration database. Concentric had planned a similar confirmation drilling campaign on the former Urangesellschaft portion of the Project for the 2007 field season, but the drill program was never done. UEC has not conducted any drilling activity to date.

5.1.1 Historical Mineral Resource Estimates

Historical mineral resources estimates were completed by,

- MinEx (1978);
- Urangellschaft (1979);
- Agapito (2010); and
- On behalf of UEC (BRS et al, 2014).

UEC considered the previous mineral resource estimates as historical. These historical mineral resource estimates were prepared before the implementation of the SEC’s S-K or Canada’s current NI 43-101 standards, and do not necessarily use the categories for mineral reserve and mineral resource reporting as defined by those standards. The reader should not rely on these historical reserve estimates, as they are superseded by the mineral resource estimate presented in Section 11.0 of this TRS.

6.0 GEOLOGICAL SETTING, MINERALIZATION, AND DEPOSIT

6.1 Regional Geology

The Project is located along the northeast margin of the Date Creek Basin of the Basin and Range Province of the western United States. The Date Creek Basin is one of hundreds of Paleogene basins throughout western Arizona, southeastern California, Nevada and western Utah. Paleogene lacustrine and fluvial sediments, and Quaternary gravels have filled these basins to depths of several thousand meters (Urangesellschaft, 1979a). The approximate location of the Basin boundaries is shown in Figure 6-1.

The basin is surrounded by dissected mountain ranges containing Precambrian metamorphic rocks and granites. Surrounding mountain ranges include the Black Mountains to the north and northeast, and the Rawhide, Buckskin and McCracken Mountains to the west. To the south and southeast, the basin is bordered by a low drainage divide imposed by the Harcuvar and the Black Mountains (MinEx, 1978). Margins of the basin are filled with early Paleogene volcanic flows and volcaniclastic sediments. The basin itself is filled with Oligocene to Miocene lacustrine and deltaic sediments covered by a thick mantle of Quaternary valley fill (Urangesellschaft, 1979a).

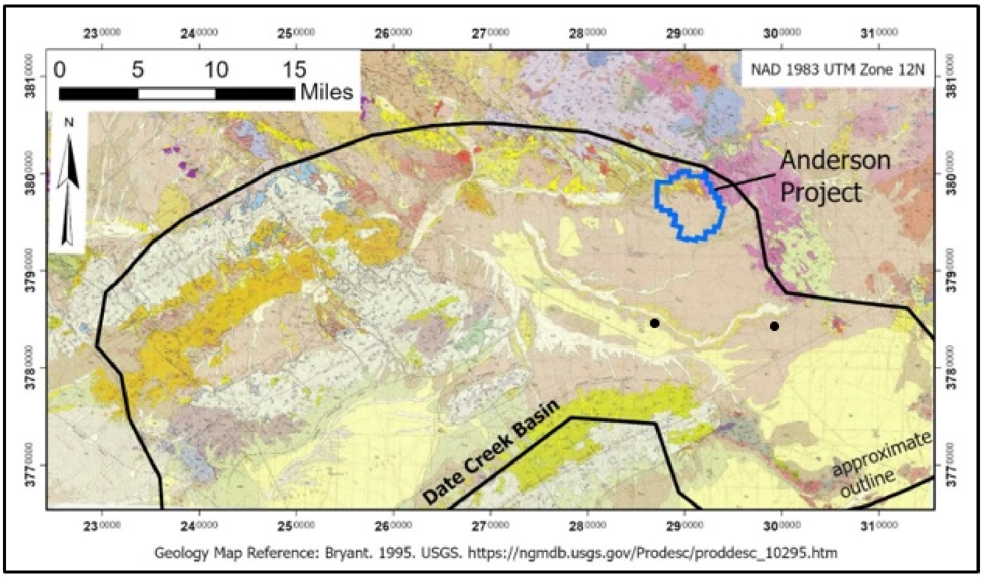
The Date Creek Basin was an area of active volcanism during Paleogene time. A thick series of volcanic flows and associated sediments of volcanic ash and clastics were deposited on the pre-existing surface. During a quiescent period, the Date Creek Basin was covered by a shallow lake or swamp in which a thick sequence of fine-grained sediments was deposited. Interbedded coarse sediments, volcanic basalt flows and conglomerates overlay the lake-bed sediments. This sequence of stratified volcanic and sedimentary rocks is 3,000 to 5,000 ft thick in the central portion of the Date Creek Basin (Hertzke, 1997).

The regional stratigraphic sequence was summarized, from oldest to youngest by MinEx (1978b), as follows:

- Precambrian or Jurassic granitic basement complex
- Lacustrine clastic and volcanic members of the Palaeocene-Eocene Artillery Peak Formation
- Arrastra Volcanic Complex, including dacitic intrusions, andesitic flows and volcaniclastic members of Paleogene age
- Chapin Wash Formation, Anderson Mine lacustrine sediments of Miocene age
- Conglomeratic-sandstone unit, possibly equivalent to upper Chapin Wash Formation
- Miocene basalt
- Pliocene-Pleistocene conglomerate
- Quaternary alluvium

Uranium Energy Corp

Figure 6-1: Regional Geologic Setting of the Date Creek Basin



Legend: Geologic Map Units

Qal	young alluvial deposits (Holocene)
Qtc	talus and colluvium (Holocene)
Qu	young and old alluvial deposits, young and old pediment gravel, talus, and colluvium (Holocene)
Qop	old pediment deposits (Pleistocene)
Tbf	basin fill deposits (Pliocene and Miocene)
Tby	younger basalt (Miocene)
Tbo	older basalt (Miocene)
Tsc	sandstone and conglomerate (Miocene)
Tlr	lacustrine rocks (Miocene)
Trmc	volcaniclastic and epiclastic rocks (Miocene)
Trmf	rhyolite, trachydacite, and dacite (Miocene)
Ta	trachyandesite (Miocene and Oligocene)
	high-angle fault

6.2 Structure

The Date Creek Basin has been on the margin of several regional deformations. The basin was located on the northwestern margin of Mazatzal Land and the southeastern margin of the Cordilleran Geosyncline and was subsequently deformed by the Laramide Orogeny. The Date Creek Basin is presently located on the margin of the Basin and Range Province and exhibits structural deformation typical of the province. Basin and Range deformation is the dominant expression evident at the Project property today. Structural trends of this deformation comprise a dominant northwest-southeast trend of parallel to sub-parallel hinged block faults and a less dominant west-northwest, east-southeast fault system. Many of these faults exhibit recurrent movements (MinEx, 1978).

Three major faults cross the Project: the East Boundary Fault System; Fault 1878; and the West Boundary Fault System (MinEx, 1978b). Faults trend predominantly N30ºW to N55ºW and dip steeply (approximately 80º) to the southwest.

Another set of faults trending more westerly (N65ºW) are present in the south-central portion of the Project. A fault set trending northeast-southwest has been speculated by Urangesellschaft and others, but has not been observed in the field (MinEx, 1978b). Many of the north-westerly surface water drainage tributaries are developed partially along fault traces (Figure 7-2).

Minor faults and shear zones occur throughout the Project. These probably represent fractures with slight offset of strata during differential compaction of the underlying sediments or local adjustment to major faulting.

The largest fold in the area is a broad, gentle, northwest-trending syncline in the south-eastern quarter of Section 9, T11N, R10W. Dips reach a maximum of 13º, except where modified by shearing. Many smaller folds with amplitudes of several feet are present in the lacustrine strata (MinEx, 1978b).

Fault displacements range from a few centimetres to more than 100 m. Fault movement is generally of normal displacement resulting in stair-stepped fault blocks. Local faults also have a tendency to hinge. Minor faulting across the mineralized area is often difficult to discern from variations in sedimentary dips. The lacustrine sediments dip south to south-westerly from 2º to 5º, to a maximum of 15º (Urangesellschaft, 1979a). Much of this dip is attributed to recurrent faulting during deposition.

6.3 Stratigraphy

Nine stratigraphic units were identified on the Project (MinEx, 1978b). Listed from oldest to youngest, they are as follows:

- Crystalline Intrusive Rocks: coarse-grained to pegmatitic Precambrian granite
- Felsic to Intermediate Volcanic: flows, breccias, tuffs and minor intrusive
- Felsic to Intermediate Volcaniclastic: ash flows, tuffaceous beds and arkosic sandstone
- Andesitic Volcanic: porphyritic andesitic flows with a paleosurface and locally reddish-brown paleosols

- Lacustrine Sedimentary rocks: micaceous siltstones and mudstone, calcareous siltstones and silty limestone, thin beds of carbonaceous siltstone and lignitic material and host of uranium mineralization, averaging about 60 to 100 m thick
- Lower Sandstone Conglomerate: arkosic sandstones and conglomerate, averaging about 60 to 100 m thick
- Basaltic Flows and Dikes: amygdular basalt, averaging about 20 m thick
- Upper Conglomerate: cobble and boulder conglomerate, partly indurate and locally calcite cemented, averaging about zero to 60 m thick
- Quaternary Alluvium: unconsolidated sand and gravel, caliche-formed where calcite-cemented

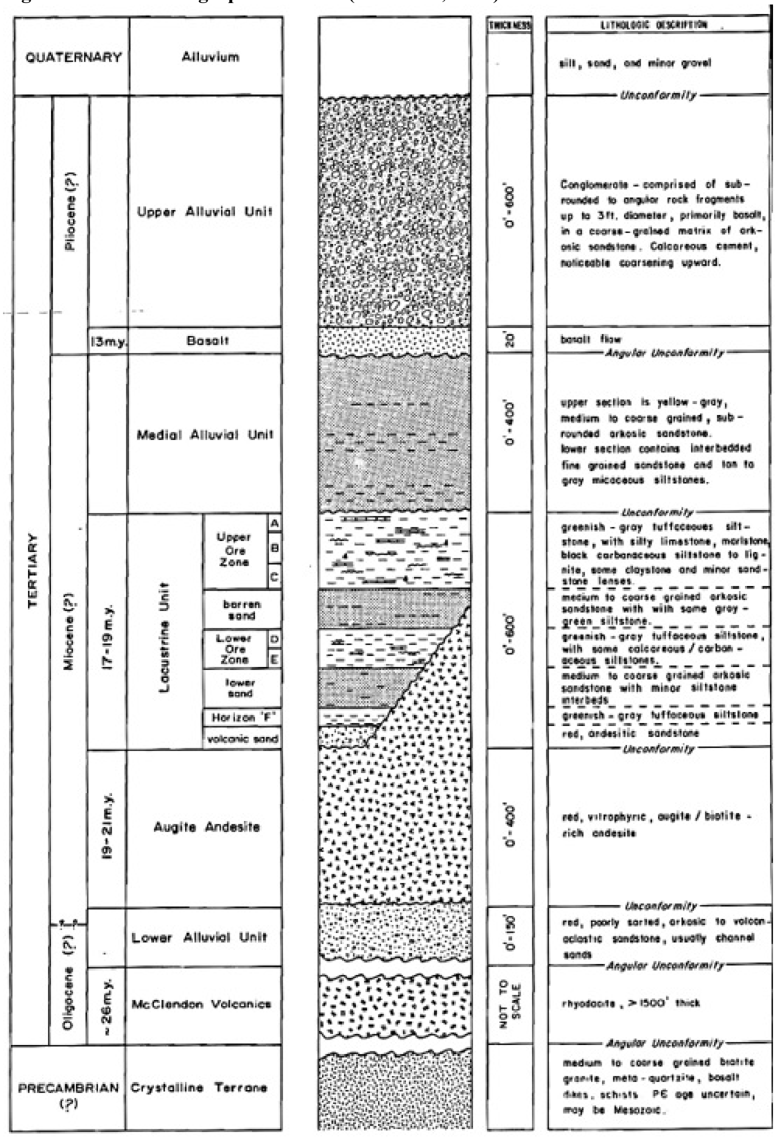
A representative, stratigraphic column of the Project is shown in Figure 6-2.

Uranium mineralization at the Project occurs exclusively in the sequence of Miocene-age lacustrine lakebed sediments. The lacustrine sediments unconformably overlie the andesitic volcanic unit over most of the Project. However, to the east of the Project, they overlie the felsic to intermediate volcanic unit.

Evidence suggests that deposition of the lacustrine sediments occurred in a restricted basin less than 5 km wide by 10 to 12 km long on the northern edge of an old Paleogene lake (Chapman, Wood & Griswold Inc., 1977). Moving southward, these sediments inter-tongue with siltstones and sandstones. The lakebed sediments represent time-transgressive facies deposited within a narrow, probably shallow, basinal feature. This type of depositional environment exhibits complex relationships between individual facies, lensing out, vertical and horizontal gradation, and interfingering (MinEx, 1978, b-c).

The lake sediments include green siltstones and mudstones, white calcareous siltstones, and silty limestone or calcareous tuffaceous material. Much of this material is silicified to varying extents and was derived in part from volcanic ashes and tuffs common throughout the lakebeds. Also present in the lacustrine sequence are zones of carbonaceous siltstone and lignitic material. Along the boundary between the former MinEx and Urangesellschaft properties, drill holes encounter the basal arkosic sandstone. To the south and southwest, lakebeds interfinger with and eventually are replaced by a thick, medium to coarse-grained arkosic sandstone unit (MinEx, 1978, b-c).

Figure 6-2: Stratigraphic Column (Schneider, 1979)



Uranium Energy Corp

BRS ENGINEERING

6.4 Mineralization

Uranium mineralization in outcrops and the pit floor at the old Anderson mine was reported by the US Bureau of Mines in Salt Lake City as tyuyamunite ($Ca(UO_2)_2(VO_4)_2 \cdot 5\text{-}8H_2O$). Carnotite ($K(UO_2)_2(VO_4)_2 \cdot 3H_2O$) and a rarer silicate mineral, weeksite ($K_2(UO_2)_2(Si_2O_5)_3 \cdot 4H_2O$), were also reported in outcrop samples. Carnotite mineralization occurs as fine coatings and coarse fibrous fillings along fractures and bedding planes, and has been noted in shallow drill holes and surface exposures as shown in the subsequent picture.

Pictured above, secondary uranium mineralization coating fracture planes.

The uranium mineralization found at depth on the former Urangesellschaft property was reported by Hazen Research, Inc. of Golden, Colorado, (Hazen Research) to be poorly crystallized, very fine-grained, amorphous uranium with silica. This could be in the form of either coffinite ($U(SiO_4)_{1-x}(OH)_{4x}$) or uraninite (UO_2) in a primary or unoxidized state (Hertzke, 1997). Mineralogical studies performed by Hazen Research (1978a, 1978b, 1978c and 1979) on Urangesellschaft core found that mineralization was associated, for the most part, with organic-rich fractions of the samples. Specifically, the uraniferous material occurs as stringers, irregular masses and disseminations in carbonaceous veinlets with uranium up to 54% as measured by microprobe analysis. X-ray diffraction identified the mineral as coffinite. It is possible that an amorphous, ill-defined uranium silicate with a variable U:Si ratio is precipitated and, under favorable conditions, develops into an identifiable crystalline form (coffinite).

Of special note is the detection of high-grade, low-reflecting uraniferous material occurring with carbonaceous material in the siltstone. Similar assemblages in unoxidized mineralization have also been reported for the former MinEx property (Hertzke, 1997).

Urangesellschaft (1979a) distinguished seven mineralized zones, identified as Horizons A, B, C, D, E, F and G, with the youngest (uppermost) being Horizon A and the oldest (deepest) being Horizon G. The majority of uranium occurs in Horizons A, B and C within the property. A conglomeratic sandstone unit interbeds with these units, but does not contain uranium mineralization; it is referred to as the Barren Sandstone Unit and it lies between Horizon C and Horizon D. Consequently, Horizons A through C have been called the Upper Lakebed Sequence and Horizons D through G have been called the Lower Lakebed Sequence.

Grades of mineralization range from 0.025% U3O8 to normal highs of 0.3% to 0.5% U3O8 with intercepts on occasion of 1.0% to 2.0% U3O8. Secondary enrichment of the syngenetic mineralization is observed along faults and at outcrops (Hertzke, 1997).

Uranium mineralization hosted by the basal San Miguel Formation of the UPC is interpreted to represent a variety of the roll-front-type mineralization by the early workers of Anschutz. Sandstone-type deposits are characteristically sedimentary formations of clastic-detrital origin, containing reducing environments. These deposits are usually tabular in shape and may occur in continental sandstones, deltaic or shallow marine environments. Typically, roll-front-type uranium deposits have, in the direction of the flow of mineralizing solutions, a barren (oxidized) interior zone surrounded by a (reduced) mineralized zone. Between the barren zone and the mineralized zone is an altered zone. The overall shape of the roll-front is similar to a crescent with extended tails at each end, which also outlines the barren interior zone, and uranium is deposited at the interface between the oxidized zone and the reduced zone. Ground water flow direction is usually a good guide in detecting roll-front-type deposits in sandstones. Figure 6.3 shows the Project drill hole locations and cross sections shown on Figures 6.4 and 6.5.

The style of mineralization within the sandstones at the Project includes some characteristics of the roll-front-type mineralization, as in the Powder River Basin of Wyoming in the United States. It is likely that the style of mineralization is a variety of the roll-front-type uranium mineralization (Wilson, 2008).

6.5 Local Geology and Drilling

Figure 6.3 is a drill hole and cross section index map. The Figure shows the surface location of the existing drill holes and reference locations for representative subsurface cross section shown in Figure 6.4. Five local mineralized geologic horizons have been interpreted from the drill data, Horizons A-G, with A being the upper-most horizon.

Figure 6-3: Drill Hole and Cross Section Index Map

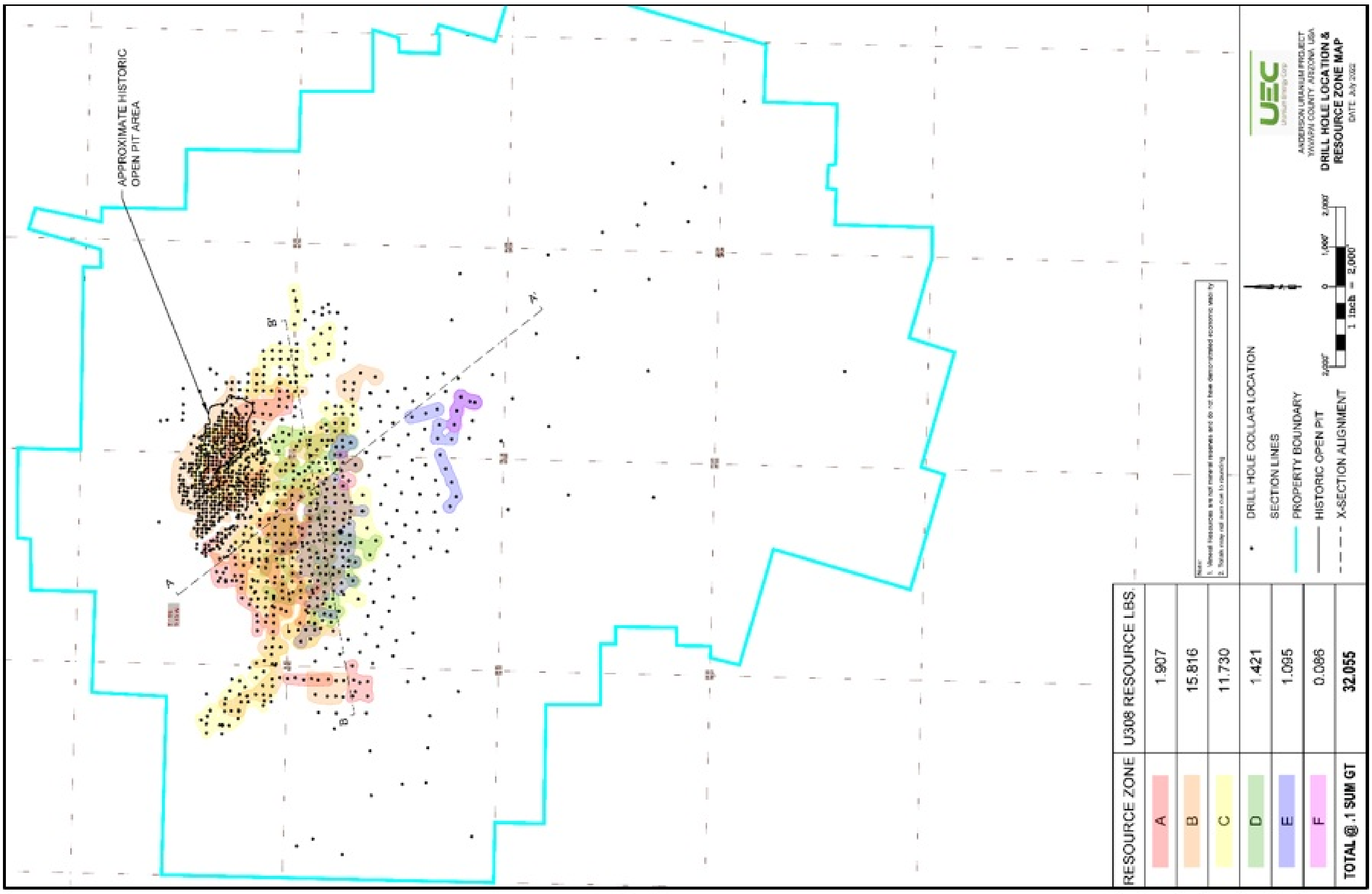
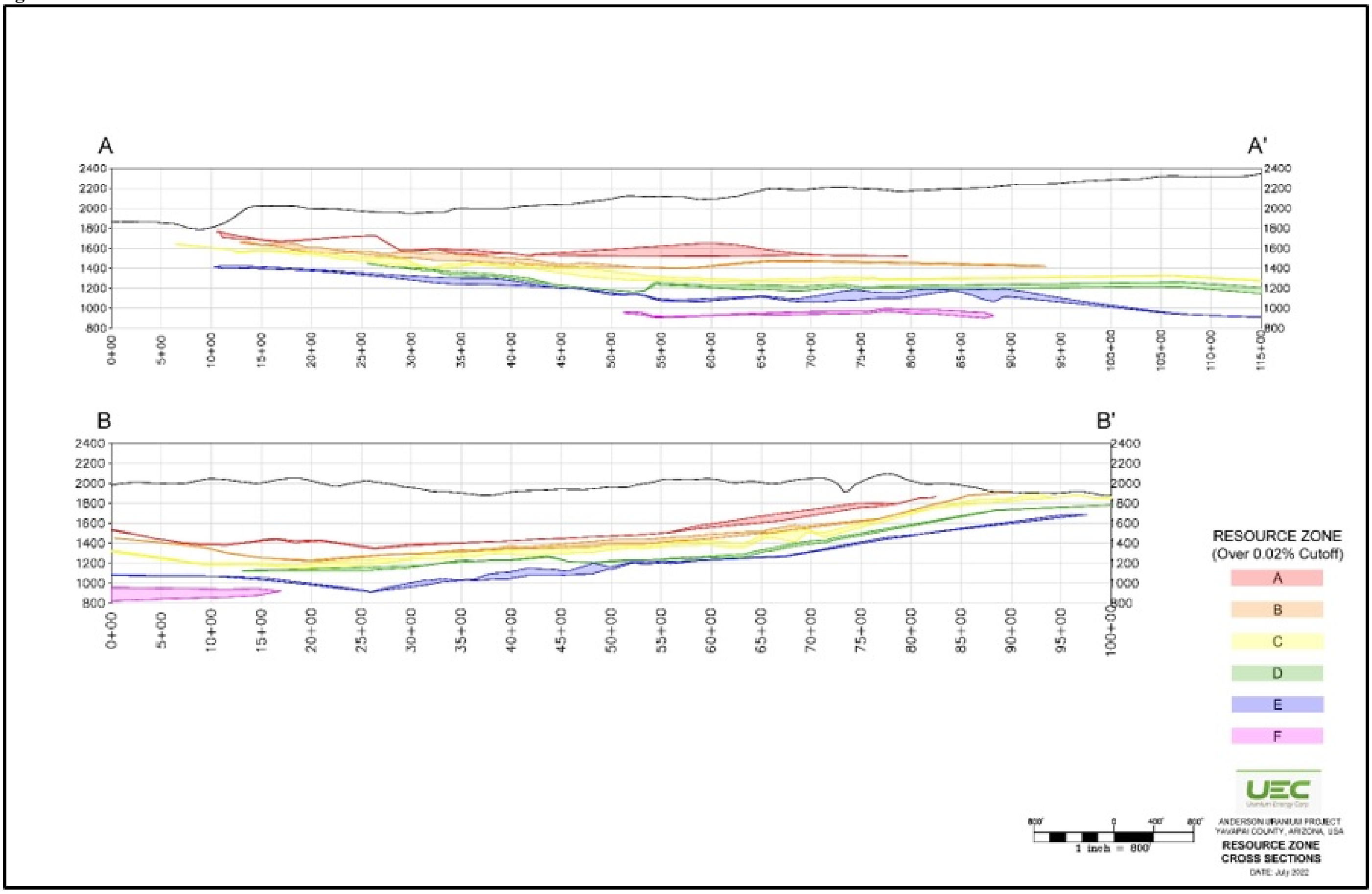


Figure 6-4: Cross Sections



7.0 EXPLORATION

7.1 Exploration

Since acquiring the Project, UEC has performed no exploration on the property and has relied entirely on legacy data for the updated mineral resource estimates and Project planning.

7.1.1 Previous Exploration

The subsequent table summarizes the phases of the historical exploration on the Project (Agapito, 2010). As described in Section 5: History, historical production occurred on the property near outcrop in the 1950s.

Company	Period	Exploration Activities
Mining Group Led by Mr. T. R. Anderson	1955−1959	Aerial scintillometer surveying, ground prospecting, and outcrop mining.
Getty	1967−1968	Limited exploration drilling.
Urangesellschaft	1973−1982	Exploration drilling: 352 total holes with 319 rotary holes and 33 core holes over a 610 ha area.
MinEx	1974−1980	Exploration drilling: 970 rotary holes and 84 core holes over a 425 ha area.
Concentric	2006	Confirmation drilling: 24 RC holes and one RC core hole.

7.1.2 Qualified Person's Interpretation of the Exploration Information

The QP considers the exploration completed to date on the Project to be consistent with industry standards and adequate to support mineral resource estimation.

7.2 Drilling On Property

7.2.1 Overview

Historical drilling on the Project for which the data is available and has been obtained by UEC totals 1,431-boreholes, including 105 core holes with chemical assay. The total reported footage drilled is approximately 702,330 ft. In addition, chemical assays are reported from a total of 3,577 samples.

Drill hole locations are shown on Figure 6.3, along with subsurface cross sections shown in Figure 6.4.

UEC has drilled no exploration holes.

Drilling included rotary percussion and diamond core drilling.

7.2.2 Drill Holes Excluded from Mineral Resource Estimation

Approximately 260 drill holes were dropped due to missing or conflicting data.

7.2.3 Drill Data Used in the Current Mineral Resource Estimation

The current drill hole database within the limits of the mineral resource model consists of 1,175 drill holes. These drillholes contained 9,279 unique intercepts within defined mineralized horizons.

The uranium quantities and grades are reported as equivalent U3O8 (eU3O8), as measured by downhole gamma logging.

7.2.4 Qualified Person's Interpretation of the Exploration Information

The QP considers the exploration completed to date on the Project to be consistent with industry standards and adequate to support mineral resource estimation.

7.2.5 Drill Hole Logging Procedure

All holes were logged for gamma ray, resistivity, and self-potential (SP) curves plotted by depth. The resistivity and SP curves provide bed boundaries and were mainly used for correlation. The resistivity curve, calibrated in ohms, is a measure of the formation water resistivity. Generally, sandstones show a deflection to the right or a greater resistivity than shale. The spontaneous, or SP curve, indicates the natural potential differences, in millivolts, between an electrode at the surface and an electrode in the probe within the drill hole fluid that is pulled up past different beds. This potential depends on a number of factors, but it generally indicates the permeable zones, or sandstones, as a deflection left from the shale baseline.

The gamma ray (radiometric) log is used to interpret the amount of equivalent U3O8 in a zone by measuring the gamma radiation of radioactive uranium decay products. Measurement units are in counts of gamma radiation per second (cps); these are converted to an equivalent percentage of uranium by weight (% eU3O8). The scintillation probes can delineate anomalous uranium mineralization to approximately 7 to 15 parts per million (ppm) eU3O8.

The logs comprised a hard-copy continuous-line plot of the values for gamma radiation, resistivity and SP. Resistivity and SP were logged only in select holes. Vertical deviation surveys were also completed in some holes, although survey data are not readily available.

In 1977, Century Geophysical Corporation (Century Geophysical) introduced computerized logging to the Urangesellschaft exploration program, which allowed equivalent uranium grades to be calculated at the time of logging.

Field data were produced in four forms:

- the regular gamma electric log;
- the digital printout of gamma data in cps;
- a grade analyses printout (% eU3O8); and
- a magnetic cartridge tape containing all data contained in the electric log.

Downhole measurements were reported for 0.5 ft intervals.

7.2.6 Gamma Probe Calibration

The principal quality control element that affects the accuracy of radiometric sampling data is the calibration of the logging probe. Records indicate that all logging was conducted with probes that were calibrated by Century Geophysical and other contractors. With only minor exceptions, probe calibration parameters, including k-factor and dead-time factor, were reported with each gamma log.

Probe calibration was audited by Chapman, Wood & Griswold Inc. (1977) on the MinEx project. Prior to the audit, two different series of radiometric probes were used to log MinEx holes AM-1 through AM-515. One series of probes was shielded (three probes) and the other unshielded (one probe). The audit evaluated calibration data from 24 June 1976 and 07 June 1977 and tested one probe as part of the audit at the Grand Junction, Colorado, N-3 test pit maintained by Bendix Field Engineering Corporation for the US Department of Energy (DOE). The audit did not identify any deficiencies within the calibration practices of the exploration program and determined the tested probe to be accurate (Chapman, Wood & Griswold, 1977).

7.2.7 Collar Surveys

The collar locations were provided by Concentric using aerial image interpretation, remote sensing and global navigation satellite system measurements. The first phase of data validation used the database coordinate locations; each drill hole location was reviewed in relation to the high-resolution orthoimage in ESRI ArcMap. The second phase of data validation involved locating and measuring drill hole locations in the Project area using a Trimble GeoXH mapping-grade GPS unit.

7.2.8 Downhole Surveys

Down-hole surveying was not done. The formation is flat lying, so there would be no preferential direction of downhole drift due to the formation. It is the authors' opinion that the downhole drift would be a function of the setup of the drilling equipment.

7.2.9 Radiometric Database

No digital gamma logs were available from the MinEx and Urangesellschaft exploration programs. A total of 1,049 paper strip logs from the former MinEx property (including 2 repeats) and 336 from the former Urangesellschaft property (8 holes missing logs) were available and used by Agapito 2010. Agapito scanned all strip logs to digital raster images using a high resolution flatbed scanner, producing black and white 200- or 400-dpi TIFF files.

Didger, version 3.0 (Golden Software, Inc., 2006), a commercially available digitizing program, was used. The digitizing procedure involved registering multiple calibration points at known positions around the on-screen gamma log. Calibration points normally included the corners and midpoints of the gamma versus depth chart. Positions X and Y were manually entered for each calibration point. X-values were entered in inches; these corresponded to a recorded scaling factor in cps/in. Y-values were entered in feet of depth below the hole collar. The calibration points allowed the software to automatically adjust for misalignment and distortion in the scanned logs. A total of 57 MinEx logs and 16 Urangesellschaft logs were eliminated from the dataset due to illegibility, excessive distortion, incomplete information or other reasons.

Once calibrated, the gamma trace was manually digitized on-screen by sampling points along the trace with the cursor. Points were sampled with sufficient resolution to represent the highs (peaks) and lows (valleys) of the trace. Between points, the digitized trace was treated as linear. Intermediate points were sampled where additional definition was required. The density of points was adjusted based on the variability of the trace. Care was taken to include enough points to capture the "peak" of spikes and the "tails" of spikes to avoid over- or under-representation of the gamma signature. A typical digitized log included 50 to 1,500 points per hole, depending on the variability of the log and its length.

The majority of logs contained "reruns," where the original trace exceeded the range of the log strip at high gamma intercepts and required replotting at smaller scales. Reruns were superimposed on the original trace in the logs. A rerun required that the entire calibration and digitizing process be repeated separately for each alternative X-scale used. Typically, two to four reruns were encountered per hole.

Digitizing coordinates were temporarily stored in ASCII (.txt) files and later compiled into a single, dedicated MS Office Excel® spreadsheet per drill hole. The X-coordinate, expressed in inches, was converted to cps by multiplying the X-coordinate by the recorded cps/in scaling factor. A rerun was converted to cps by multiplying the X-coordinate by its respective scaling factor and manually adding the rerun to the original trace listing, which ultimately produced a continuous listing of cps versus depth.

Finally, all header information associated with the log was entered into a standardized header in the Excel® log file and combined with the gamma trace listing. Key header information included drill hole collar coordinates, hole diameter, water factor, k-factor and dead-time factor. Blank or illegible log entries were left blank in the Excel® file.

Digitizing was completed by Concentric personnel. Agapito and SRK personnel were responsible for error checking and quality assurance (Agapito, 2010).

Instead of attempting a line-by-line translation of the original GAMLOG FORTRAN code, Agapito used the algorithm described by Scott (1962) to develop an independent code within Excel®, with spreadsheet rows representing depth intervals and columns representing successive iterations of the composite grades for each interval.

UEC and SRK reviewed the Excel® code and did not identify any errors in logic; they found that the macro appeared to perform as intended. Agapito tested their macro against five example logs included with the original GAMLOG program to illustrate various input and output options: each example was processed using the new procedure. Agapito, 2010 concluded the comparison of grades confirmed close agreement between the two calculation methods.

The depths versus cps data from the log file are interpolated to create an array of equivalent cps values at 15 cm (0.5 ft) intervals. In effect, the algorithm integrates the area under the counts curve for each interval. The interpolation step was not required for those holes that were digitized at 15 cm intervals.

The equivalent counts data are adjusted for dead-time and water factor and used to determine an initial grade estimate for each interval. This estimate represents the grade of an infinitely thick layer of mineralization that would produce the observed count rate. Provision is included to accommodate multiple water factors, corresponding to changes in hole diameter with depth.

7.2.10 QP Statements Concerning Radiometric Drill Data

The authors reviewed the methodology of the drilling and downhole logging procedures employed during the past drilling programs and the database derived from those programs to be reliable for the purposes of this IA.

7.3 Core Data

Historical core holes were drilled alongside approximately 7% of the MinEx rotary holes and 9% of the Urangesellschaft rotary holes to verify the geophysical logs. Chemical assaying of the core measured the concentration of uranium and other constituents, including metals, such as vanadium, and trace elements, such as lithium and manganese. MinEx and Urangesellschaft both conducted chemical assays on cores recovered from exploration core holes. Approximately 3,125 individual chemical assays were completed by MinEx from 72 core holes. Approximately 2,471 individual chemical assays were completed by Urangesellschaft from 33 core holes. Tables containing historical assay values are provided in MinEx and Urangesellschaft historical reports. The original assay certificates were not reviewed for this TRS because they are not available. Agapito (2010) compiled all the historical assay data and imported the data in Gemcom to use in resource estimation and validation. A total of 3,577 historical assays were reported: 1,610 from MinEx drill holes and 1,967 from Urangesellschaft drill holes.

Two types of quantitative chemical assays were performed: fluorometric and colorimetric (wet) assays. Both methods reported percent uranium by weight. Chemical assays were performed by Hazen Research, which is certified under the United States Environmental Protection Agency (USEPA). Additional assays were completed at Chemical & Geochemical Laboratories in Casper, Wyoming and check assays were completed at Skyline Assayers and Laboratories, Inc. in Tucson, Arizona, both accredited commercial laboratories. Both the historic core drilling and sample assay program employed standard quality control and assurance procedures.

The authors reviewed the methodology of the core sampling and assay procedures employed during the past drilling programs and determined that the database derived from those programs is reliable for the purposes of this IA. It is, however, recommended that additional core data be collected as part of future mine design and/or feasibility studies for evaluation of disequilibrium conditions and for metallurgical testing.

7.4 Hydrogeology

Hydrologic investigations are limited to historic studies. Groundwater in the immediate vicinity of the Project property was investigated jointly by Urangesellschaft and MinEx. Hydrologic studies were conducted by the Water Development Corporation of Tucson, Arizona (WDC 1978, 1979).

Within the immediate vicinity of the Project property, two aquifers contain most of the groundwater: the Barren Sandstone Unit and the Lower Sandstone Conglomerate Unit. Water occurs near the base of the Lower Sandstone Conglomerate Unit at the contact with the lacustrine sediments. A pump test conducted by MinEx in this zone yielded an average flow rate of 57 gallons per minute (gpm). The initial and final water depths in this test were 56.2 and 78.6 ft, respectively, yielding a total drawdown of 22.4 ft and a specific capacity of 2.5 gpm/ft of drawdown (MinEx 1978b).

The deeper Barren Sandstone Unit is generally quite permeable and contains an artesian system. Groundwater in this unit is expected to produce 50 to 100 gpm. The artesian head stands at 300 to 450 ft above the level at which the aquifer is encountered. Urangesellschaft's drill water supply well produced water from the Barren Sandstone Unit at an estimated 80 to 100 gpm (Urangesellschaft 1979a). Specific capacity of the unit is approximately 1.4 gpm/ft of drawdown, based on well pump measurements conducted by Urangesellschaft (MinEx 1978b).

Considerable faulting and fracturing in the vicinity of the Project property has resulted in sufficient movement of water between units, such that it is not appropriate to treat the units as independent, particularly to the north on the former MinEx property where faults cut the Lower Sandstone Conglomerate Unit. To the south on the former Urangesellschaft property, the Lower Sandstone Conglomerate Unit occurs well above the water table.

The lacustrine sequence hosting the uranium mineralization is thought to be saturated but, due to poor permeability, water does not move readily through the unit. Faults in the lacustrine sequence in the deep areas to the south are considered too tight to allow significant water movement (Urangesellschaft 1979a).

The authors reviewed the methodology of hydrological studies employed during the past operators and consider these to be reliable for the purposes of this IA. It is, however, recommended that appropriate hydrological studies be completed as part of future mine design and/or feasibility studies.

7.5 Geotechnical Testing

Geotechnical testing is limited to historic studies. Urangesellschaft commissioned a geotechnical study through Dames & Moore in the spring of 1979 to evaluate highwall design associated with a downdip extension of the proposed MinEx open pit onto the former Urangesellschaft property (Dames & Moore 1979).

The study concluded that highwall stability could be achieved under drained conditions. In an undrained condition, a uniform slope was determined not to be capable of achieving an acceptable safety factor, even at a relatively shallow slope angle of 35º. Under drained conditions, a uniform slope between 35º and 40º was determined to achieve an acceptable safety factor.

The study recommended a benched slope design consistent with the design recommendation for MinEx (Dames & Moore 1977), consisting of a shallower slope of 35º to 40º in the lower, weak rock units and a steeper slope of 50º in the more competent overlying rock units, with a 100-ft-wide catch bench located between the slopes.

The authors reviewed the methodology of geotechnical studies employed during the past operators and consider these to be reliable for the purposes of this IA. It is however, recommended that appropriate geotechnical studies be completed as part of future mine design and/or feasibility studies.

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8.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

8.1 Sampling Methods

As discussed in Section 7, the primary sampling method is the collection of radiometric equivalent data via downhole radiometric logging. The majority of the drilling was completed in the 1970s and early 1980s, and consists of analog geophysical logs. The geophysical logs were scanned and digitized using standard industry methods to create a digital database for which the downhole counts per second (CPS) were converted to equivalent uranium grades %eU_3O_8 using industry accepted algorithms.

Core data is available from past drilling programs, but original laboratory certificates are generally not available.

8.2 Security

The original drill logs and remaining samples are stored in a facility located in Wickenburg, Arizona. Available samples are from post-2010 drilling programs. Samples from drilling programs prior to that time were not preserved.

8.3 Density Determinations

Bulk density sample data, referred here as specific gravity (SG) data, is limited to samples from 48 drill holes in the Project area. The majority of the SG data is from Resource Zone B and C, with little or no data available from the other domains.

Bulk density data was collected from core intervals within mineralized intervals. The SG sample average is 18.9 cf/st (1.70 t/m3) and ranges from 32.0 to 12.8 cf/st (1.00 t/m3 to 2.51 t/m3). Historically, an average value of 20.5 cf/st (1.56 t/m3) was used to calculate resource tonnage for the Project deposit. Comments in previous technical reports suggest that the SG value represented near-surface rocks and that the density increases to as high as 16.9 cf/st (1.9 t/m3) with depth. Due to the limited sample data and the fact that more potentially important economic resources tend to occur nearer to the surface, a constant value reflecting the sample average of 18.9 cf/st (1.70 t/m3) was used to calculate tonnages from the resource model. This assumption is considered reasonable and somewhat conservative based on available information. The author was not able to verify this data, as the samples were not preserved. It is recommended that additional density sampling be conducted.

8.4 Downhole Geophysical Logging

As discussed in Section 7, geophysical logging was routinely conducted for every drill hole completed on the Project by all operators. Natural gamma logs provide an indirect measurement or radiometric equivalent measurement of uranium content by logging gamma radiation in counts per second at one-tenth foot intervals. Counts per second are then converted to eU308.

Conversion of the downhole CPS to equivalent uranium content expressed as %eU3O8 followed industry standard procedures as described in Section 7 of this TRS.

8.4.1 Disequilibrium

The great majority of the data available for estimation of mineral resources is radiometric equivalent data from geophysical logging data. Radiometric equilibrium conditions may affect the grade and spatial location of uranium in the mineralization. Generally, an equilibrium ratio (Chemical U3O8 [c] to Radiometric eU3O8 [e]) is assumed to be 1, i.e., equilibrium is assumed. Equilibrium occurs when the relationship of uranium with its naturally occurring radioactive daughter products is in balance. Oxygenated groundwater moving through a deposit can disperse uranium down the groundwater gradient, leaving most of the daughter products in place.

Disequilibrium studies were completed by past operators, including Urangesellschaft and MinEx. Both studies concluded that remobilization of uranium had occurred, especially in the vicinity of faults.

In 1978, MinEx completed a disequilibrium study based on approximately 3,125 colorimetric and fluorimetric assays on core from 72 core holes distributed across the MinEx property (Lucht 1978). Average disequilibrium factors were calculated as the ratio of total uranium pounds chemical to total pounds gamma in each hole for a given polygonal area of influence. No attempt was made to correlate factors with individual radiometric beds. Disequilibrium factors varied over a wide range from 0.10 to 1.80, with low values generally associated with uranium depletion along faults. The average disequilibrium factor across the entire historical MinEx property was determined to be 0.999 (Agapito, 2010).

Urangesellschaft conducted a comparable disequilibrium study on the same 1.0-ft core samples from 33 core holes used in the 1979 bulk density study (Urangesellschaft 1979a). The program consisted of approximately 2,174 chemical assays. The disequilibrium factor per hole was calculated as the ratio of the total composite grade-tonnage (GT) for the flourimetric log to the GT for the associated gamma log. Factors were determined for four cutoff grades: 0.02%, 0.03%, 0.04%, and 0.05% U3O8. Results show the same trend toward increasing disequilibrium factors at higher grades. The average disequilibrium factor for the Urangesellschaft underground mining resource ranged from 0.992 at the 0.02% cutoff to 1.120 at the 0.05% cutoff (Agapito, 2010).

Although the historic studies show that a positive adjustment could be made for disequilibrium, the studies have not been verified. Thus, as a conservative measure, no adjustment in the equivalent uranium grades is recommended by the author.

8.4.2 Downhole Probe QC

The principal quality control element that affects the accuracy of radiometric sampling data is the calibration of the logging probe. Records indicate that all logging was conducted with probes that were calibrated by Century Geophysical and other contractors. With only minor exceptions, probe calibration parameters, including k-factor and dead-time factor, were reported with each gamma log.

Probe calibration was audited by Chapman, Wood & Griswold Inc. (1977) on the MinEx project. Prior to the audit, two different series of radiometric probes were used to log MinEx holes AM-1 through AM-515. One series of probes was shielded (three probes) and the other unshielded (one probe). The audit evaluated calibration data from 24 June 1976 and 07 June 1977, and tested one probe as part of the audit at the Grand Junction, Colorado, N-3 test pit maintained by Bendix Field Engineering Corporation for the DOE. The audit did not identify any deficiencies within the calibration practices of the exploration program, and determined the tested probe to be accurate (Chapman, Wood & Griswold, 1977).

8.5 Database

As discussed in Section 7, the database for the Project was developed using standard industry methods. The author reviewed the database, which included locating, validating and compiling drill hole locations and downhole mineralized interval data within the Project area. From the overall dataset grade, thickness and X, Y and Z locations were derived for all mineralized intercepts. These mineralized intercepts were then screened by 3-dimensional location and geologically interpreted into six stratigraphically discrete and laterally continuous Resource Zones.

8.6 Authors' Opinion on Sample Preparation, Security and Analytical Procedures

In the opinion of the authors:

- sample collection, preparation, analysis and security for drill programs are in line with industry-standard methods for similar uranium projects;
- drill programs included downhole gamma and calibration procedures are in line with uranium industry standard operating procedures; and
- digital database construction and security are adequate. The physical database is properly organized.

The authors are of the opinion that the quality of the uranium analytical data is sufficiently reliable to support mineral resource estimation, without limitations on mineral resource confidence categories.

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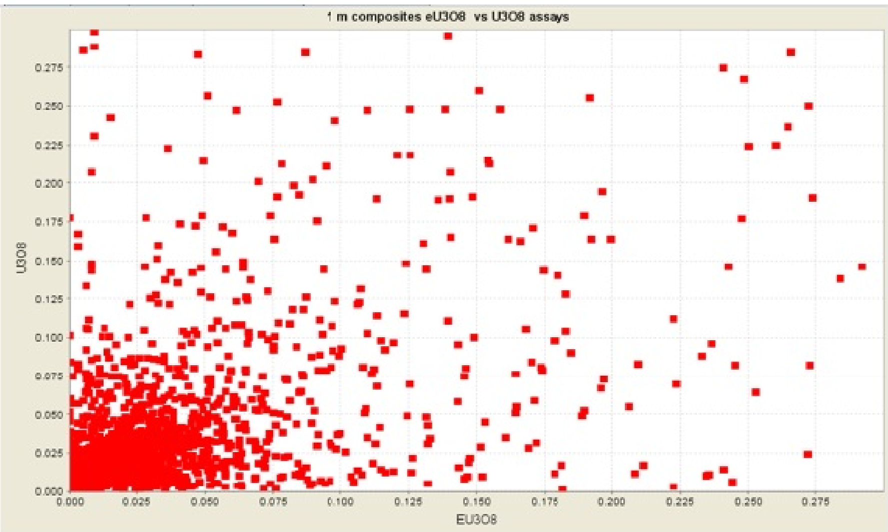
9.0 DATA VERIFICATION

9.1 Drill Data

As previously discussed in Section 7, industry standard methods were utilized at the time of data collection. Geophysical logs for historic drillholes were analyzed and evaluated for completeness and sufficiently quality checked in the process of developing the drillhole database for the resource modeling. Original geophysical logs for pre-2010 drilling were scanned and digitized. Downhole gamma measurements were converted to gamma equivalent values in %eU3O8 using standard methods (Agapito, 2010). Post-2010 geophysical logs and drill data were provided in digital format and converted to gamma equivalent values using the same method as applied to the pre-2010 data.

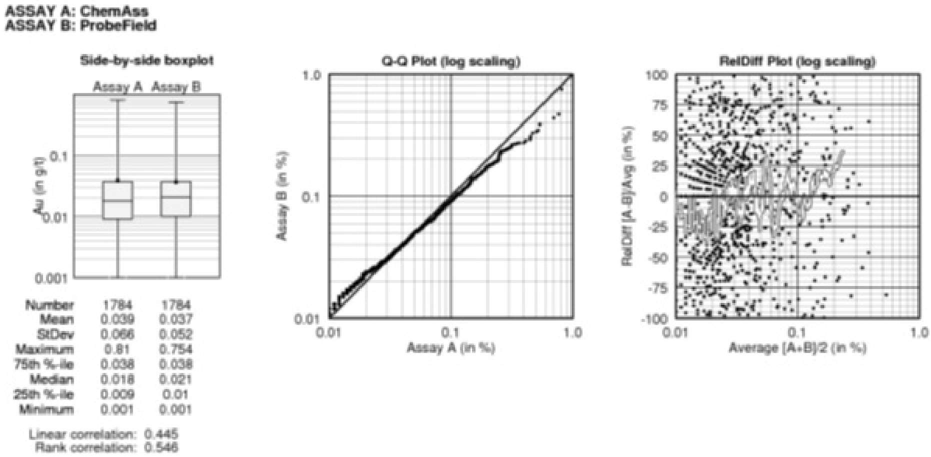
To confirm that the conversion of the cps to eU3O8 was done correctly, Agapito (2010) compared the converted eU3O8 data with the chemical assays for the 109 holes containing chemical assay data. The correlation between the assayed U3O8 and the eU3O8 values showed a wide scatter and the correlation is not very clear on the subsequent figure.

Figure 9-1: Scatter Plot of eU3O8 and U3O8 for Core Holes (Agapito, 2010)



A comparison of the Q-Q plot of the same data, however, demonstrates that for quartiles greater than 0.07%, the eU3O8 value is slightly less than the corresponding U3O8 assay value, which indicates that the eU3O8 values may be slightly underestimated or conservatively estimated (Figure 9-2).

Figure 9-2: Q-Q Plot Comparison



9.2 Drill Hole Locations

UEC validated the collar locations provided by Concentric using aerial image interpretation, remote sensing and global navigation satellite system measurements. The first phase of data validation used the database coordinate locations: each drill hole location was reviewed in relation to the high-resolution orthoimagery in ESRI ArcMap. Drill pads and areas of disturbance are easily identified on the orthoimagery. The location was deemed to be valid based on the following criteria: if the location of the drill hole matched a drill pad, the location and the elevation from the DEM were accepted; if the location of a drill hole did not match an area of disturbance in the image, the drill hole was flagged and rechecked in the field. Of the 1,336 drill hole locations originally provided in the database, approximately 20% (269 drill holes) were flagged for field-checking. An additional 27 areas were identified on the orthoimagery that showed disturbance, but were not correlated with a drill hole location.

The second phase of data validation involved locating and measuring drill hole locations in the Project area using a Trimble GeoXH mapping-grade GPS unit. The following drill hole classifications were used:

- located, if the drill hole was found and measured;
- location probable, if the original location was accepted after relocation in the field; and
- wrong location, if the drill hole could not be relocated or the site was obviously not suitable for drilling, such as on a steep hillside.

14 drill holes were found in the field that were ultimately not included in the database; they were classified as “unknowns”. One of these unknowns was identified from a Minex drill hole map. The drill hole database was updated with measured horizontal coordinates from the field work. Collar elevations were taken from the Cooper Aerial DEM, unless it was determined that subsequent earth works covered the drill hole; in this case, the original collar elevation was used.

9.3 Qualified Person's Opinion on Data Adequacy

The historic and more recent exploration data and the overall data adequacy is deemed to be reasonably sufficient by the authors for applying QA/QC techniques and verifying the legitimacy of the data incorporated into this IA.

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10.0 MINERAL PROCESSING AND METALLURGICAL TESTING

10.1 Introduction

Preliminary laboratory scale metallurgical testing programs have been completed for the Project. No definitive bench or pilot study has been completed. A summary of the testing follows.

Between 1977 and 1979, MinEx and Urangesellschaft conducted mineral processing and metallurgical test work. The historical mineral processing and metallurgical test work has not been verified. Testing for both companies was completed by Hazen Research. Testing focused on acid and alkaline leaching, with minor testing on heap leaching. Research had advanced to a relatively mature stage, culminating in a conventional acid-leach mill process flowsheet for the MinEx Final Feasibility Study (MinEx, 1978).

Prior to mid-1978, MinEx and Urangesellschaft conducted independent investigations on mineralized material from their respective properties. By late-1978, MinEx and Urangesellschaft began negotiations on a joint venture. From that point, metallurgical research was conducted cooperatively, with Urangesellschaft modifying its testing program to demonstrate the amenability of the Date Creek mineralization to MinEx's process flowsheet. It was concluded that the MinEx acid-leach process, although viable, was costly due to the high reagent consumption and, considering the wide chemical variability of the mineralized material across both properties (Urangesellschaft, 1980), it was not the optimal process. MinEx and Urangesellschaft outlined additional research to address outstanding deficiencies.

10.2 Acid Leaching

The host rock consists mostly of carbonaceous shale, siltstone and limestone, with both relatively hard and soft layers. The deposit consists mainly of two mineralized horizons, referred to as the Middle Zone and the Lower Zone. These are highly variable with respect to both carbonate and uranium content. The CO3 content varies from 0.03% to 32% and U3O8 ranges from 0.03% to 0.5%, with no apparent correlation between the two (MinEx, 1978).

MinEx and Urangesellschaft performed leach amenability tests on representative core samples from mineralized horizons. This was complicated by the variability in Corg (lignite) and CO3 (limestone) contents. Uranium extraction through acid leaching varied from 80% to 95%, with most samples exceeding 90%. Acid consumption ranged from 50 kg/ t (110 lbs/ton) to over 490 kg/t (1,080 lbs/ton) of processed material (average 270 kg/t) (Urangesellschaft, 1980). Analysis of leach solutions showed Na2CO3 consumptions of 17 to 37 kg/t. Roasting of the mineralized material lowered acid consumptions by approximately 6% to 21%. Testing also showed increased uranium solubility with decreasing particle size through minus 28-mesh. Thickening characteristics for high-lime composites were generally poor (6 to 14 ft2/tpd), but improved with roasting (3 to 9 ft2/tpd) (Hertzke, 1997).

Metallurgical analyses by Hazen Research established near-optimal, acid-leach conditions for the northern portion of the Project property, capable of 90% uranium extraction (Hertzke, 1997). The metallurgical programs concluded that substantial quantities of sulfuric acid would be required to sustain extraction rates above 90%. Typical mineralization was estimated to contain a minimum of 19% to 20% calcium carbonate (CaCO3), corresponding to a minimum acid consumption of 180 kg (400 lbs) of sulfuric acid (H2SO4) per ton of ore. Leachability was further complicated by carbonate contents, which varied from 0% to over 50% CaCO3 (Urangesellschaft, 1980).

Using acid-leach parameters employed by MinEx, comparative results were obtained (Hertzke, 1997):

- Average uranium solubility of 86.2% for Urangesellschaft mineralization versus 88.5% for MinEx ore
- Average acid consumption of 250 kg (551 lbs/ton) for Urangesellschaft mineralization versus 188 kg (415 lbs/ton) for MinEx ore

Further investigations to optimize leaching were briefly evaluated. One test involved the use of Caro's acid instead of sulfuric acid. This resulted in higher initial uranium dissolution rates, suggesting the possibility of significantly shorter leaching times. Another study involved preliminary flotation tests to remove significant portions of the acid-consuming minerals by calcite flotation prior to acid leaching. Results proved inconclusive and suggested that further innovation and study are required to optimize the acid-leach process (Urangesellschaft, 1980).

10.3 Alkaline Leaching

Alkaline leach tests on bulk samples from the MinEx deposit showed uranium extractions of 96%. Lower zone extractions ranged from 94% to 95% (Hazen, 1977). Alkaline leaching tests conducted by Urangesellschaft resulted in early-term uranium extraction rates of 81% to 94%. Final-stage uranium dissolutions ranged from 84% to 97% (Urangesellschaft, 1980). Settling rates for four of five composites tested by Urangesellschaft were relatively good with unit area requirements of 1.6 to 3.7 ft2/tpd.

Studies determined that the feasibility of alkaline leaching was limited by the high organic carbon content, primarily in the form of lignite (Corg), which poses a potential for significant problems during uranium solvent extraction by inhibiting the precipitation of sodium diuranate. Typical Date Creek mineralized material is estimated to contain 2.6% Corg on average. Actual values varied widely, ranging from 0% to over 15% Corg (Urangesellschaft, 1980).

The program concluded that additional research was required to improve the understanding of the relationship between U3O8 grade and associated CO3 and Corg, and that future test work should be focused on reducing the carbonate and/or lignite content of the mineralization before leaching. Preliminary research by MinEx determined that activated carbon or surfactants, Aerosol C-61 and Arquad T-2C-50, are effective in removing the alkaline soluble organics, but the amounts needed are not likely to be economically viable. Roasting was determined to be effective in eliminating the organics, but proved detrimental to uranium solubilization at temperatures above 350ºC (Hazen, 1977).

10.4 Heap Leaching

Limited testing of acid-heap leaching was done by Urangesellschaft. Mineralized material was leached using a recirculating solution of 5 g/l sulfuric acid. The column flow rate was favorable, and almost 70% of the uranium was solubilized during the initial days. The flow rate and uranium content of the effluent decreased drastically, and after 45 days of leaching, an overall solubilization of 79% was achieved (Urangesellschaft, 1978).

The historical metallurgical test work is limited and may not be representative of all types and styles of mineralization on the Project property. However, there is nothing in the test work to suggest that economic extraction of the mineralization will not be possible. More extensive testing will be required in order to determine this.

10.5 2013/2014 Metallurgical Testing

The current metallurgical scoping test results are provided in the report “Scoping Metallurgical Testing of Anderson Mine Samples” dated May 30, 2014, RDI, Appendix A.

On December 18, 2013, Mr. Beahm and Mr. McNulty were on site along with UEC personnel. During this time, four samples of approximately 50 lbs each were taken from discrete mineralized stockpiles across the site. Mr. Beahm cataloged the samples and delivered them personally to the RDI laboratory in Wheat Ridge, Colorado, for analysis and testing. Upon delivery of the samples, RDI screened the samples by size fractions (i.e. +2 in, 2X1 in, 1X½ in, ½ inX10 mesh, 10X48 mesh and minus 48 mesh), assayed the samples through a commercial lab using Xray Fluorescence (XRF) for uranium, vanadium, and carbonates and completed Loss of Ignition testing for organic material. Three of the four composites showed a distinct concentration of uranium values in the size fractions less than 0.5 inches. The sample composite assays ranged from 474 to 976 ppm uranium, 388.4 to 515.7 ppm vanadium, and the combined CaO and MgO content of the samples was less than 3.5%, indicating relatively low acid consumption under leaching.

The metallurgical testing was then completed in a phased approach. The initial scope was to investigate conventional milling, heap leaching and/or the combination of size separation and attrition scrubbing to mechanically concentrate the mineralized material prior to leaching. For these scenarios, the testing was to first focus on acid lixiviant. If the initial testing indicted high acid consumption or low recoveries, then alkaline lixiviant would be tested. As the testing proceeded, it was clear that the leaching parameters for an acid lixiviant were acceptable and alkaline lixiviant was not tested.

Limited testing of mechanical upgrading with including sizing and attrition scrubbing was inconclusive, although a concentration of uranium values in the fines was achieved. As the leach tests had demonstrated high recoveries with favorable acid consumption, acid leach was used in further testing. Mechanical upgrading would be appropriate if agitated leaching conventional mill) or vat leaching would be considered for the concentrate. However, in either case, heap leach recovery would likely also be needed for processing the oversize material. Further testing relating to mechanical upgrading of the mineralized material is recommended, particularly if regulators view this process to be a mining process rather than “milling”, which would require NRC licensing.

Initial leach tests were completed using standard “bottle roll” methods. Bottle roll leach tests provide reliable predictions of maximum uranium/vanadium extractions and maximum sulfuric acid consumption as functions of particle size and pH. Tests demonstrated that leaching extractions are somewhat higher at a fine grind (200-mesh) than at a fine crush (6-mesh), but not dramatically so. The respective 24-hour uranium extractions at pH 1.0 were 96.1% and 90.3%, indicating that coarser feed will leach satisfactorily if given enough time. The tests did clearly show that a high free acid concentration is necessary to satisfactorily extract uranium and to at least extract 25% of the vanadium. Bottle roll sulfuric acid consumption was lower (48.4 lb/ton) with the coarser feed than with finely ground feed (55.9 lb/ton), which is consistent with increased rock particle surface area at the finer grind. Leaching at different sodium chlorate (oxidant) concentrations indicated that a modest dosage of 2.0 lb/ton is adequate. However, future test work may reveal that less will be sufficient.

Once the leach chemistry parameters were established from the bottle roll tests, bucket leach test were conducted to simulate heap leaching at a bench scale. Bucket leach simulation of heap leaching is primarily useful in showing whether mineralization will degrade physically during heap leaching, generating fines and impairing solution percolation. For one of the four composites, the fines (minus 400-mesh) increased significantly from 23% to 33% of the total weight. This effect can be mitigated during operation by agglomerating the heap feed with dilute sulfuric acid at an addition rate of about half of the total acid consumption, as indicated by bottle rolls.

Acid consumptions in the bucket tests were very high at 1125-180 lb/ton, but it reflects aggressive exposure of various rock forming minerals to strong acid. Acid is then consumed during alteration or decomposition of those minerals. With respect to anticipated recovery and acid consumption, the bottle roll tests are the most conclusive and were applied in the PEA.

Note that acid consumption from the current heap leach testing is considerably lower than was reported for metallurgical testing in the 1970s. There are two major factors which contribute to this difference in acid consumption. First, the historical mine plans were based on maximum resource recovery, at a low cut-off and with bulk mining, which would add dilution. This would have resulted in mining more of the calcareous material as well as lignitic materials. The current mining plan focuses on only three of the mineralized horizons and only in the areas where the economic factors are most favorable. As such, the historical metallurgical testing reflected somewhat different mineralized material than the current testing program. Second, mineralization was not contemporaneous with the formation of the host rock, but occurred later. Mineralization at the site was observed as a filling interstitial voids and coating factures. While the heap leach process includes sizing the mineralized material by crushing, it is not ground as finely as would be the practice in conventional milling. Coarser sizing results in fewer reactive sites to consume acid. Current bottle roll test demonstrated the effect of sizing on acid consumption, with the finer grind (200 mesh) consuming more acid than the fine crush (6-mesh). Also, in conventional agitated leaching, the host rock is sufficiently exposed to the acid to fully react with the acid. This was demonstrated by the current bucket tests, which showed acid consumption similar to the historic testing when the host rock was left in contact with the acid for a long period of time. In summary, current testing indicates that heap leaching can recover uranium at reasonable acid consumption levels, without reacting fully with the host rock.

Conclusions from the current metallurgical testing include:

- Uranium extraction of over 90% was obtained at a pH of 1.0.
- Static bucket test confirm that uranium can be recovered via heap leaching.
- Vanadium extraction will be low at 25-40%.
- Acid consumption was reasonable at +/- 50 lbs per ton.
- Some degradation of the samples during the bucket test was observed, indicating that acid leaching may reduce heap permeability.

There is a risk that the samples taken from existing stockpiles may not be representative of the mineralized material, which may be treated throughout the life of the Project. The stockpile did originate from the C horizon, which is the primary focus of the conceptual mine plans, as described in Section 16. However, the stockpiles have been exposed at the surface and have likely oxidized to some extent relative to the condition of freshly mined material.

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11.0 MINERAL RESOURCE ESTIMATES

11.1 Introduction

The mineral resource estimation described herein utilizes geological interpretation methodologies, which have been employed by the authors for similar projects while working at operating mines within similarly hosted uranium mineralization. The primary method utilized in estimating the uranium mineral resources is the GT contour method, which is the CIM method recommended for sandstone-hosted deposits, such as those within the Project. The resource estimate was generated using drill hole sample results and the interpretation of a geologic model that relates and constrains the spatial distribution of U_3O_8. Mineral resources in the lacustrine sediment hosted deposits within each of six Resource Zones of the Project area were estimated using the same GT contour modeling methodology. Mineral resource estimates are presented for each Resource Zone separately.

The Project is within a well-known mining district. The Anderson mine was first discovered in 1955 due to anomalous surface radioactivity and the discovery of uranium oxide in outcrop. The Anderson mine is a Brownfield site with historic open pit mine production and extensive exploration, which resulted in the disturbance of approximately 195 acres of lands. These lands are federally administered by the BLM. The disturbances pre-date environmental regulatory regulations and standards and, as a result, there is no current obligation for reclamation of the site. In some states, such disturbances would be addressed by the state Abandoned Mine Lands program, which is funded through the Office of Surface Mining and based on a tax on current coal mine production. The Arizona program has not addressed this site and is not likely to do so. Both the BLM and the United States Geological Service have and/or are considering the development of AML-like programs, however, it is not known whether the historic mine related disturbances at the Anderson mine would be addressed. Although some local opposition is expected, the author is not aware of any factors, including environmental, permitting, taxation, socio-economic, marketing, political or other factors that would materially affect the mineral resource estimate herein.

11.2 Modeling Data Preparation and Interpretation

Historical drilling on the Project for which the data is available and has been obtained by UEC total of 1,431-boreholes, including 105 core holes with chemical assay. The total reported footage drilled is approximately 702,330 ft. In addition, chemical assays are reported from a total of 3,577 samples.

Data preparation included locating, validating and compiling drill hole locations and downhole mineralized interval data within the Project area. Data verification is discussed in detail in Section 9 of this TRS. From the overall dataset grade, thickness and X, Y and Z locations were derived for all mineralized intercepts. These mineralized intercepts were then screened by 3-dimensional location and geologically interpreted into six stratigraphically discrete and laterally continuous Resource Zones.

Approximately 260 drill holes were dropped due to missing or conflicting data. The modeled drill hole database consists of 1,175 drill holes. These drillholes contained 9,279 unique intercepts, which were categorized in the following manner:

Table 11-1: Drilling Intercept Data by Resource Zone

Resource Zone	No. Boreholes Intercepting Zone	Total No. Intercepts	No. Intercepts meeting 0.02% eU3O8 cutoff
A	232	657	310
B	811	3625	1822
C	619	3350	1548
D	212	720	426
E	198	761	488
F	47	166	103

The following criteria were used to build databases for the six Resource Zones referred to as A, B, C, D, E and F.

- Individual mineralized intervals were identified in each drill hole using characteristics of shape and position of natural gamma radiation from electric logs and chemical assay of core holes, where available. Cutoff criteria were applied to screen out intervals below 0.02% eU_3O_8.
- Mineralized Bedding Thickness Cutoffs and compositing of intercept data into sum grade x thickness values for each Resource Zone varied dependent upon the intended mining extraction method.
    - Surface Mining extraction methods applied to Zones A and B are assumed to allow for recovery of mineralized beds under 0.5 ft in thickness. As such, an idealized sum GT compositing method was applied to Zones A and B, with no minimum intercept thickness.
    - Highwall mining and underground mining extraction methods require a minimum mining thickness of 3 ft. This compositing criterion was applied to intercepts within Resource Zones C, D, E and F. Intercepts below 3 ft in thickness were recomposited to a minimum of 3 ft using 0.005% interstitial background grades. Intercepts overlapping within 3 ft were composited together. The resulting composited intercepts with average grades lower than the 0.02% eU_3O_8 were then screened out. Only intercepts with average grades greater than 0.02% were used to produce a sum GT product to model resource.

Following application of screening and compositing criteria, a single composited thickness, grade and GT value for each Resource Zone in each drill hole was calculated. Digital database records thus consist of X-Y-Z coordinates and composited interval data (thickness, grade and sum GT values) for each of the six modeled Resource Zones.

The QP considers the data used for the mineral resource estimate to be adequately prepared and is satisfied that the digital data are adequate for 2-D mineral resource estimation.

11.3 Geological Model

The Project mineralization is of syngenetic origin and similar in style to deposits found in Argentina and Lake Maitland, Australia. Most or all of the lakebeds on the property exhibit some uranium mineralization. The highest grades and most continuous mineralization are confined to the carbonaceous siltstones and lignitic materials. Occasional mineralization also exists in the basal sandstone of the lacustrine sediments and in the Lower Sandstone Conglomerate Unit. Carbonaceous material is known to interfinger with the basal sandstone, and carbon has been noted in the Lower Sandstone Conglomerate Unit. Remobilization of the uranium has resulted in the deposition of uranium as fracture fillings around and below the main mineralized zones (MinEx, 1978b-c).

Carbon tends to immediately fix uranium when it comes into contact with uranium in solution. Therefore, much of the mineralization is restricted to the top or bottom of the carbonaceous facies. However, mineralization can occur in the middle of some carbonaceous zones. This latter relationship implies that mineralization occurred during the deposition of the carbonaceous material (MinEx, 1978b-c). Mineralization is also prevalent in calcareous facies.

Uranium mineralization occurs within a host sequence of lacustrine sediments that dip gently towards the south. These sediments become interfingered with siltstones and sandstones toward the south, resulting in a more layered distribution of uranium-bearing zones. Each zone is roughly tabular due to being stratabound by the lacustrine bedding. Within each zone there are multiple mineralized beds separated vertically by beds of low to barren grades.

The most likely scenario for mineralization may be from alteration of tuffaceous sediments, which were deposited in the lacustrine environment combined with solution, mobilization and deposition of uranium contained in the granitic highlands to the north. Liberation of uranium in proximity to organic material resulted in the formation of the semi-cyclic blanket deposits with the richer grades being associated with organic-rich facies. The uranium in the lacustrine host rocks has not been remobilized by geochemical cells, such as those responsible for the well-known roll-front deposits of Wyoming and south Texas. This lack of mobility is demonstrated by the absence of uranium mineralization in the barren sandstone unit, which should be an ideal host for roll-front type deposits (Hertzke, 1997).

Historically, Urangesellschaft (1979) distinguished seven mineralized zones, identified as Zones A, B, C, D, E, F and G, with the youngest (uppermost) being Zone A and the oldest (deepest) being Zone G. Most of the uranium mineralization occurs in Zones A, B and C within the property. A conglomeratic sandstone unit interbeds with these units, but does not contain uranium mineralization; it is referred to as the Barren Sandstone Unit and it lies between Zones C and D. Consequently, Zones A through C have been called the Upper Lakebed Sequence and Zones D through G have been called the Lower Lakebed Sequence. The lowermost G Zone was excluded from modelling as it had too few intercepts to be reliably modeled via the GT contour method.

11.4 GT Contour Modeling: Key Assumptions and Basis of Estimation

The Indicated Mineral Resource model was completed using the inverse distance squared GT (Grade x Thickness) Contour Modeling Method for each of mineralized zones of the deposit. The Contour Modeling Method, also known as the Grade x Thickness (GT) method, is a well-established approach for estimating uranium resources and has been in use since the 1950s in sedimentary hosted uranium deposits in the US. The technique is most useful in estimating tonnage and average grade of relatively planar bodies where lateral extent of the mineralized body is much greater than its thickness, as is observed with the data for Project.

For tabular and roll-front style deposits, the GT method provides a clear illustration of the distribution of the thickness and average grade of uranium mineralization. The GT method is particularly applicable to the tabular, stratigraphically-controlled mineralization of the Property, as it can be effective in reducing the undue influence of high-grade or thick intersections, as well as the effects of widely spaced, irregularly spaced or clustered drill holes. This method also makes it possible for the geologist to fit the contour pattern to the geologic interpretation of the deposit.

Indicated mineral resources were estimated by summation of GT intervals only when Resource Zone drillhole intercepts presented:

- minimum individual intercepts having grades above 0.02% $_eU_3O_8$;
- minimum Sum GT per resource zone at or above 0.1 ft%; and
- proximity of at least three contiguous intercepts each given a maximum isotropic radius of influence of 200 ft.

No smoothing was applied to the internal contours following the creation of the GT model, as smoothing of contours adds resources beyond strict data interpretations by using statistical methodology not justified in this context. Structural influences were not given any special interpretative prominence to be conservative (previous models assumed anisotropic enrichment along faults). Lastly, the bulk density factor applied was the cross-Project sample average of 18.9 cubic ft per ton (1.7 tonne per cubic meter).

For each zone, the limits of mineralization were determined by interpretation of the drill data. Within these limits, the GT and T (Grade x Thickness and Thickness) were contoured, as described above. Although an automated contouring program was used to produce the model surface itself, 3-dimensional limits were established where appropriate to constrain the model by the appropriate 0.1 ft% GT cutoff. The volume of the 3D model was then calculated using AutoCAD Civil 3D program software. To that volume, a bulk unit weight of 18.9 cubic ft per ton (1.7 tonne per cubic meter) was applied to calculate the pounds of eU3O8. Similarly, the tons of mineralization were calculated using the same methodology by constructing a 3D model of mineral thickness within the same area. Average grade was then calculated by dividing GT model $_eU_3O_8$ pounds by the T model-calculated mineralized tons.

In the case of historical mining performed within the A Resource zone, the full area of disturbance was removed from the resource. This was to provide a conservative estimate, as the full disturbance includes spoils piles and haulages.

The GT contour method is used as common practice for mineral reserve and mineral resource modeling for similar sandstone-hosted uranium projects (CIM, 2003). It is the opinion of the author that the GT contour method, when properly constrained by geologic interpretation, provides an accurate estimation of contained pounds of uranium.

11.5 Radiometric Equilibrium

11.5.1 General

Radioactive isotopes decay until they reach a stable non-radioactive state. The radioactive decay products are of two general categories, the first being the sub-atomic energy-generating product (i.e., the radiation) and the second being the atomic isotope. Decay product isotopes are referred to as daughters, and occur down what is known as a decay chain. When all the decay products are maintained in close association with the primary uranium isotope U_{238} for the order of a million years or more, the decay chain will reach equilibrium with the parent isotope, whichmeans that the daughter isotopes will be decaying in the same quantity as they are being created (McKay, 2007).

An otherwise equilibrated decay system may be put into a state of disequilibrium when one or more decay products are mobilized and removed from the system because of differences in solubility between uranium and its daughter isotopes. In addition, both the primary isotope of uranium U238 and its daughters emit different forms of radiation as they decay. The primary field instruments for the indirect measurement of uranium, either surface or down-hole probes, measure gamma radiation. Within the uranium decay, the gamma-emitting elements are primarily $Radium_{226}$, $Bismuth_{214}$, and uranium, with $Radium_{226}$ being the dominant source of gamma radiation.

Disequilibrium is considered positive when there is a higher proportion of uranium present compared to daughters, and negative where daughters are accumulated and uranium is depleted. The disequilibrium factor (DEF) is determined by comparing radiometric equivalent uranium grade eU_3O_8 to chemical uranium grade. Radiometric equilibrium is represented by a DEF of 1, positive radiometric equilibrium by a factor greater than 1, and negative radiometric equilibrium by a factor of less than 1.

Except in cases where uranium mineralization is exposed to strongly oxidized conditions, most of the sandstone roll-front deposits reasonably approximate radiometric equilibrium. Disequilibrium is normally spatially variable in sandstone-hosted deposits. The nose of a roll-front deposit tends to have the most positive DEF, and the tails of a roll-front would tend to have the lowest DEF (Davis, 1969).

11.5.2 Disequilibrium Factor (DEF) Determination

The high carbon reductant content and syngenetic nature of the mineralization of the Project means that very little remobilization of U3O8 is observed across the site. As such, and unlike roll-front style deposits, the mineralization at the Project can be considered at equilibrium. Review of the historic data and information indicates gamma probe and chemical assay procedures were carefully calibrated and compared. Correction for differences between equivalent and chemical assay (disequilibrium) was properly applied per each sample interval. Similarities that exist between historic drilling data (location, style and tenor) suggest that there is no reason to question the results from earlier drilling programs. It is the author's opinion that the sample database is of sufficient accuracy and precision to generate a mineral resource estimate, and that the disequilibrium factor be otherwise treated as 1.0.

11.6 Commodity Price

Unlike many other commodities, uranium does not trade on the open market; buyers and sellers negotiate contract for sales privately. Both spot marker and long-term uranium prices are published by private companies, including Trade Tech and UxC. The spot price of uranium as of May 21, 2022, according to UEC's website (https://www.uraniumenergy.com/#) is $63.00 per pound.

By their nature, all commodity price assumptions are forward-looking. No forward-looking statement can be guaranteed, and actual future results may vary materially.

11.7 Reasonable Prospects of Economic Extraction and Cutoff Determination

Based on the depths of mineralization, average grade, thickness and GT, it is the QP's opinion that the mineral resources at the Project can be reasonably and economically recoverable through a combination of surface and underground mining methods using a long-term price of $65/lb. An additional portion of resource, outside of the pit shell, can be reasonably extracted through highwall mining and underground mining methods where justified.

A Preliminary Economic Assessment (PEA) for the Project was completed under NI 43-101 and CIM guidance and regulations in 2014 (Beahm, et al, 2014). The cutoff applied was a minimum grade of 0.02% U_3O_8 and a GT of 0.1%Ft for mineral resources within the envelope of the conceptual open pit. This equates to a minimum economic extraction criterion of 0.02% recoverable grade over 5 ft of minimum mining thickness. For the portions of the resource falling within the 2014 Pit Bottom, highwall mining and underground mining envelope and extracted via underground and highwall mining methods, the 0.02% average grade cutoff criteria was applied to composited mining thicknesses of a minimum of 3 ft and a sum GT cutoff of 0.1%Ft. Within the conceptual mine limits and at the cutoff criteria applied, the operating costs (Beahm, et al 2014) were estimated at $34/lb, or about half of the Project commodity price. For this IA, the mine limits and cutoff criteria of the 2014 PEA were applied to the mineral resource estimate solely to segregate mineral resources having reasonable prospects for eventual economic extraction from within the overall envelope of mineralization. It is recommended that mining methods and costs be updated in a future preliminary economic assessment or pre-feasibility study.

11.8 Confidence Classification of Mineral Resource Estimate

The continuity of individual mineralized lacustrine bedding deposits is demonstrated by drill hole results, as displayed in the cross sections in Figure 6.4. Thickness and grade continuity within the Resource Zones is typical of tabular stratabound uranium deposits.

For this IA, the Project mineral resource for only indicated mineral resource was evaluated. For the indicated mineral resource, the classification strategy was based on the following criteria:

1) where contiguous mineralized drill holes were less than 200 ft apart;

2) a minimum of three contiguous mineralized drill holes were needed to justify reasonable economic extraction;

3) whether a portion of the Resource volume fell within UEC's property boundary; and

4) whether the resource volume fell within the 2014 PEA Conceptual Mining Envelope and thus had reasonable prospects of economic extraction.

11.9 Mineral Resource Statement

Mineral resources were estimated separately for each resource zone. First, the total contained mineralized material was estimated. Then, reasonable prospects for economic extraction were applied. Indicated Mineral Resources are reported within the 2014 PEA Conceptual Mining envelopes as having reasonable prospects for economic extraction, and represent an 18% reduction from the estimate of total mineralized material. The results of the estimation of indicated mineral resources for the Project are reported in Table 11-2.

Figures illustrating spatial distribution of the U_3O_8 resource in the five Resource Areas of the Project with respect to GT and Thickness contours follow at the end of this section.

Table 11-2: Project Indicated Mineral Resources

Mineral Resource Estimates (0.1% Sum GT Cutoff)	Tons (millions)	Average Sum Thickness (ft)	Average Grade (%eU3O8)	Pounds eU3O8 (millions)
Resource Zone A				
Reasonably Extractable Indicated Resource	0.862	3.8	0.111	1.907
Resource Zone B				
Reasonably Extractable Indicated Resource	7.347	9.5	0.108	15.816
Resource Zone C				
Reasonably Extractable Indicated Resource	6.211	10.4	0.094	11.730
Resource Zone D				
Reasonably Extractable Indicated Resource	0.760	3.2	0.093	1.421
Resource Zone E				
Reasonably Extractable Indicated Resource	0.911	7.6	0.060	1.095
Resource Zone F				
Reasonably Extractable Indicated Resource	0.084	4.6	0.051	0.086
ALL ZONES GRAND TOTALS				
Extractable Indicated Resource	**16.175**	**8.2**	**0.099**	**32.055**

Note:
1. Mineral Resources are not mineral reserves and do not have demonstrated economic viability.
2. Economic factors have been applied to the estimates in consideration of reasonable prospects for economic extraction.
3. Totals may not sum due to rounding

11.10 Risk Factors That May Affect the Mineral Resource Estimate

Factors that may affect the mineral resource estimate include:

- assumptions as to forecasted uranium price;
- changes to the assumptions used to generate the GT cutoff;
- changes to future commodity demand;
- variance in the grade and continuity of mineralization from what was interpreted by drilling and estimation techniques;
- density assignments;
- assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits and maintain the social license to operate.

Mineral resources do not have demonstrated economic viability, but they have technical and economic constraints applied to them to establish reasonable prospects for economic extraction. The geological evidence supporting indicated mineral resources is derived from adequately detailed and reliable exploration, sampling and testing, and is sufficient to reasonably assume geological and grade continuity. The indicated mineral resources are estimated with sufficient confidence to allow the application of technical, economic, marketing, legal, environmental, social and government factors to support mine planning and economic evaluation of the economic viability of the Project.

The QP expects that the majority of the indicated mineral resources could be upgraded to indicated mineral resources with additional drilling. Larger inferred and indicated resources may also be quantified with further evaluation of current Project economics.

11.11 QP Opinion on the Mineral Resource Estimate

In the opinion of the authors, the data available for the Project is sufficiently reliable for estimation of mineral resources for the purpose of this IA.

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Figure 11-1: GT Model Zone A

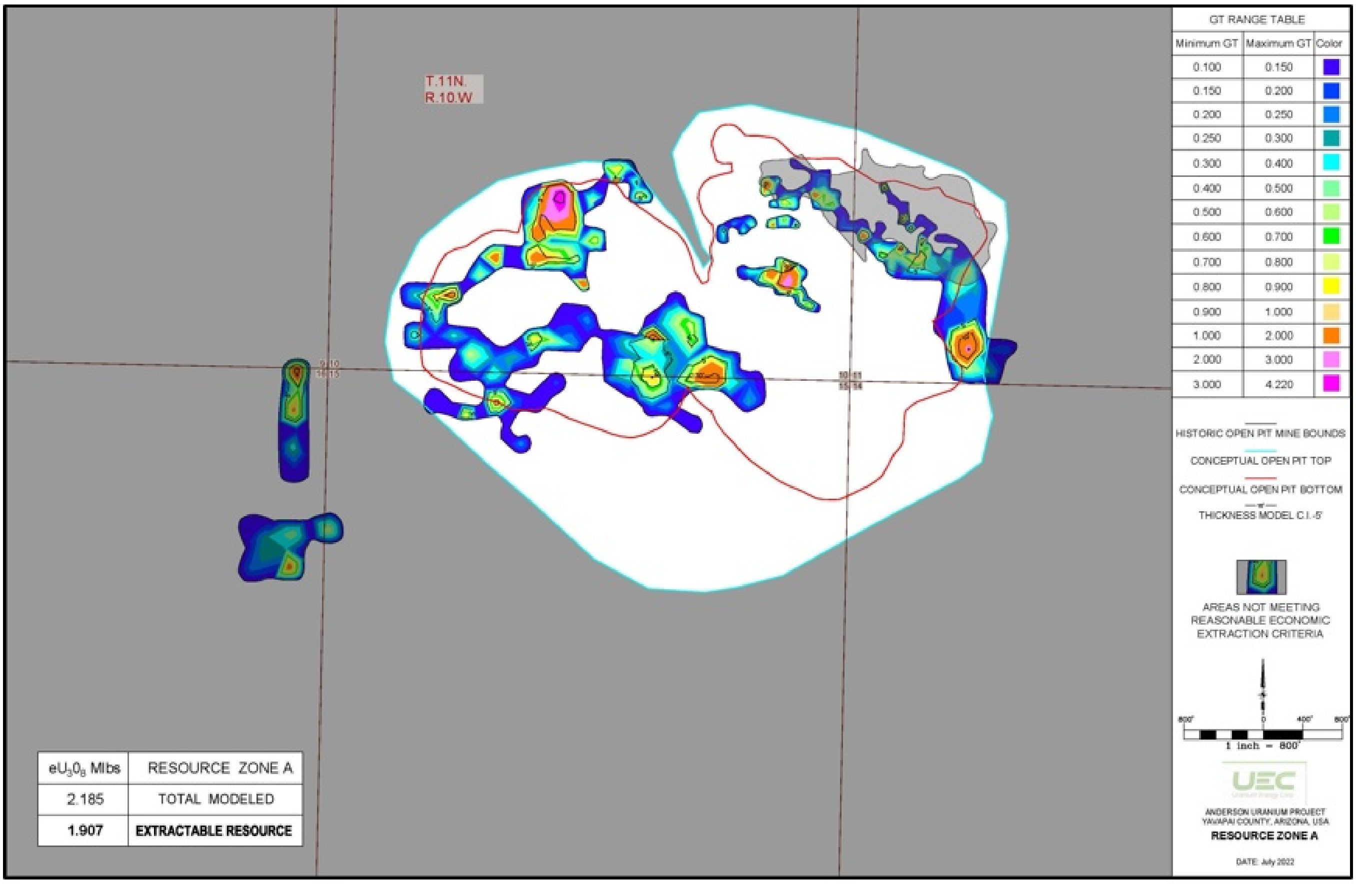


Figure 11-2: GT Model Zone B

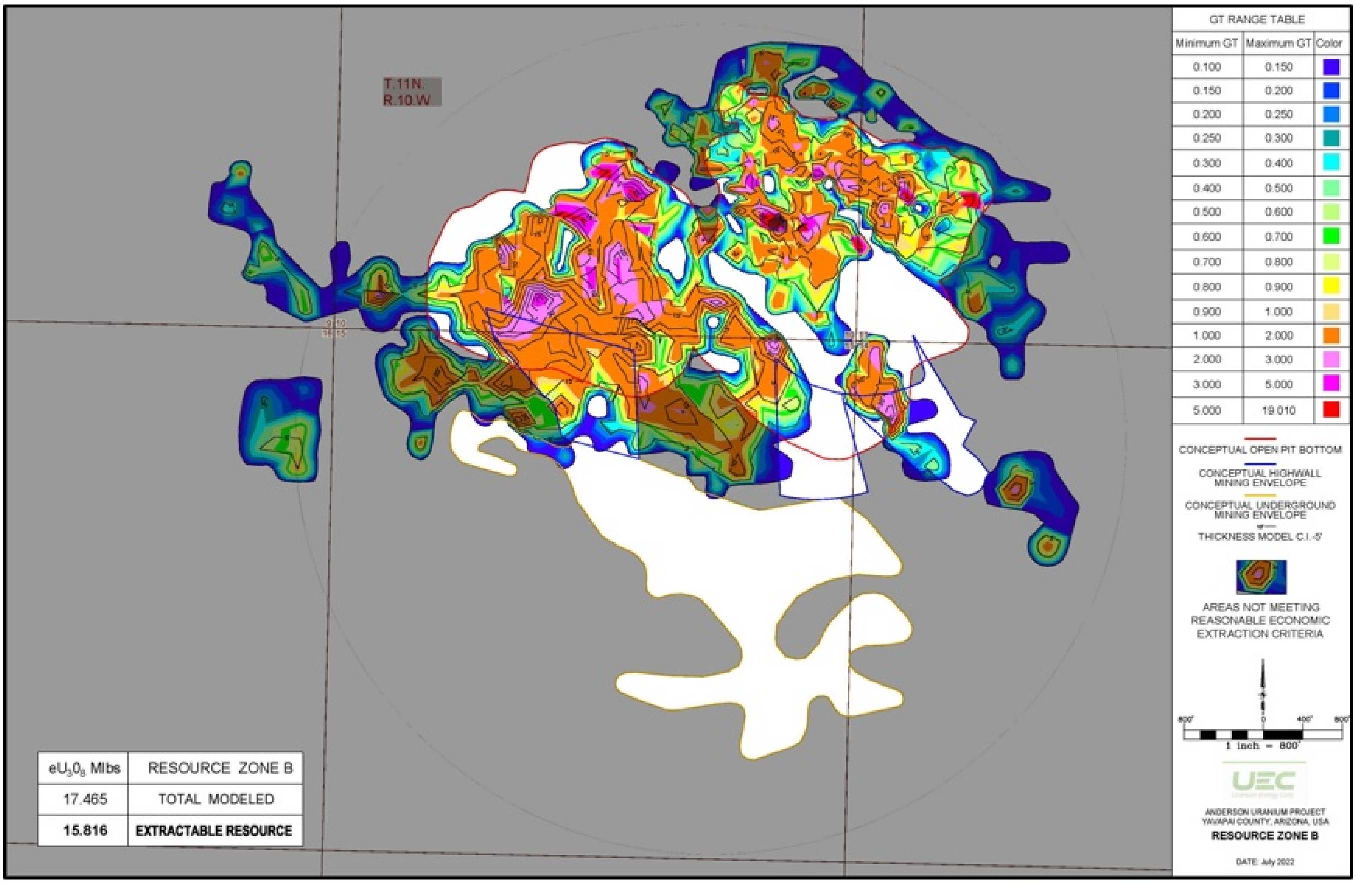


Figure 11-3: GT Model Zone C

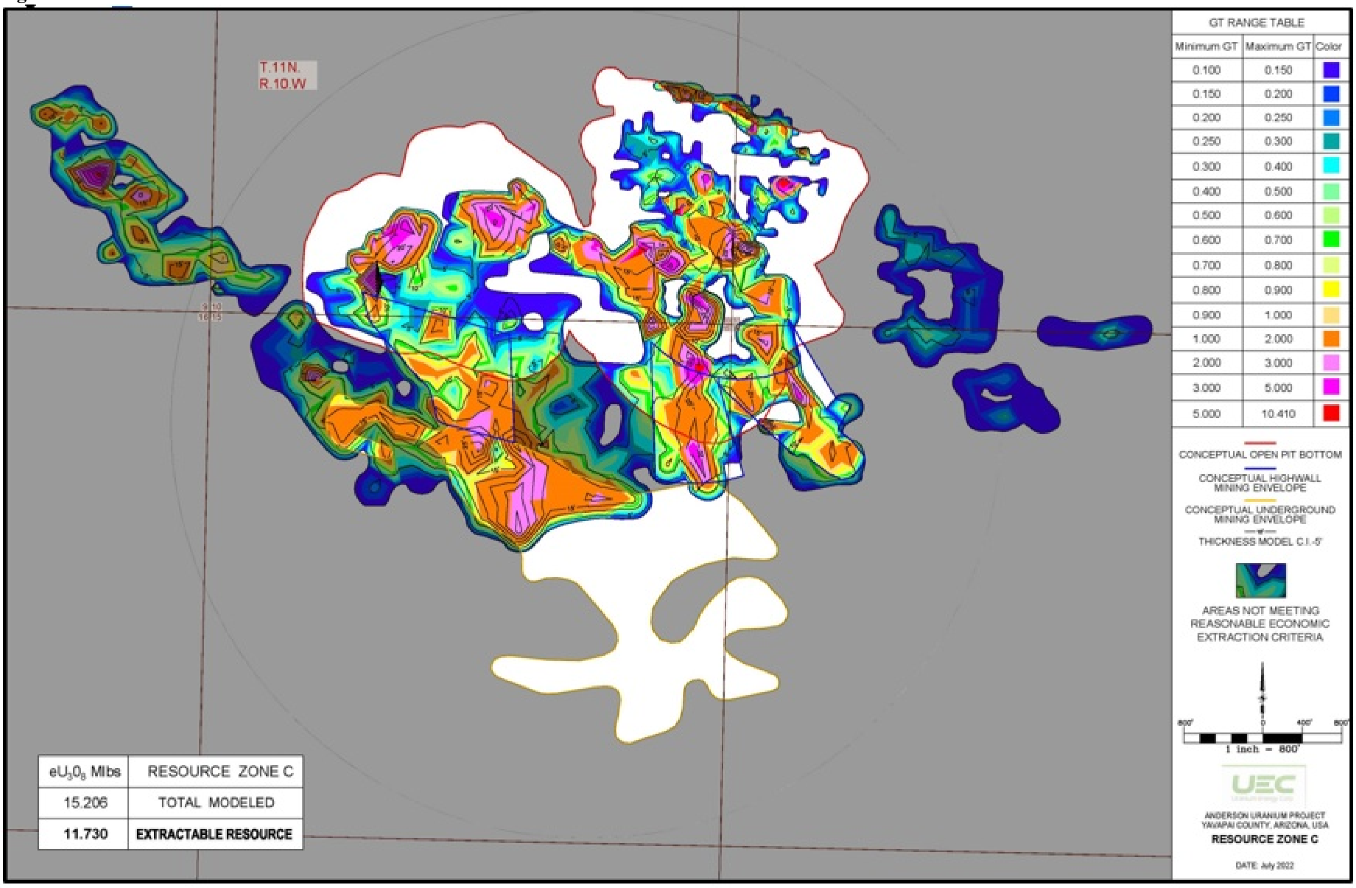


Figure 11-4: GT Model Zone D

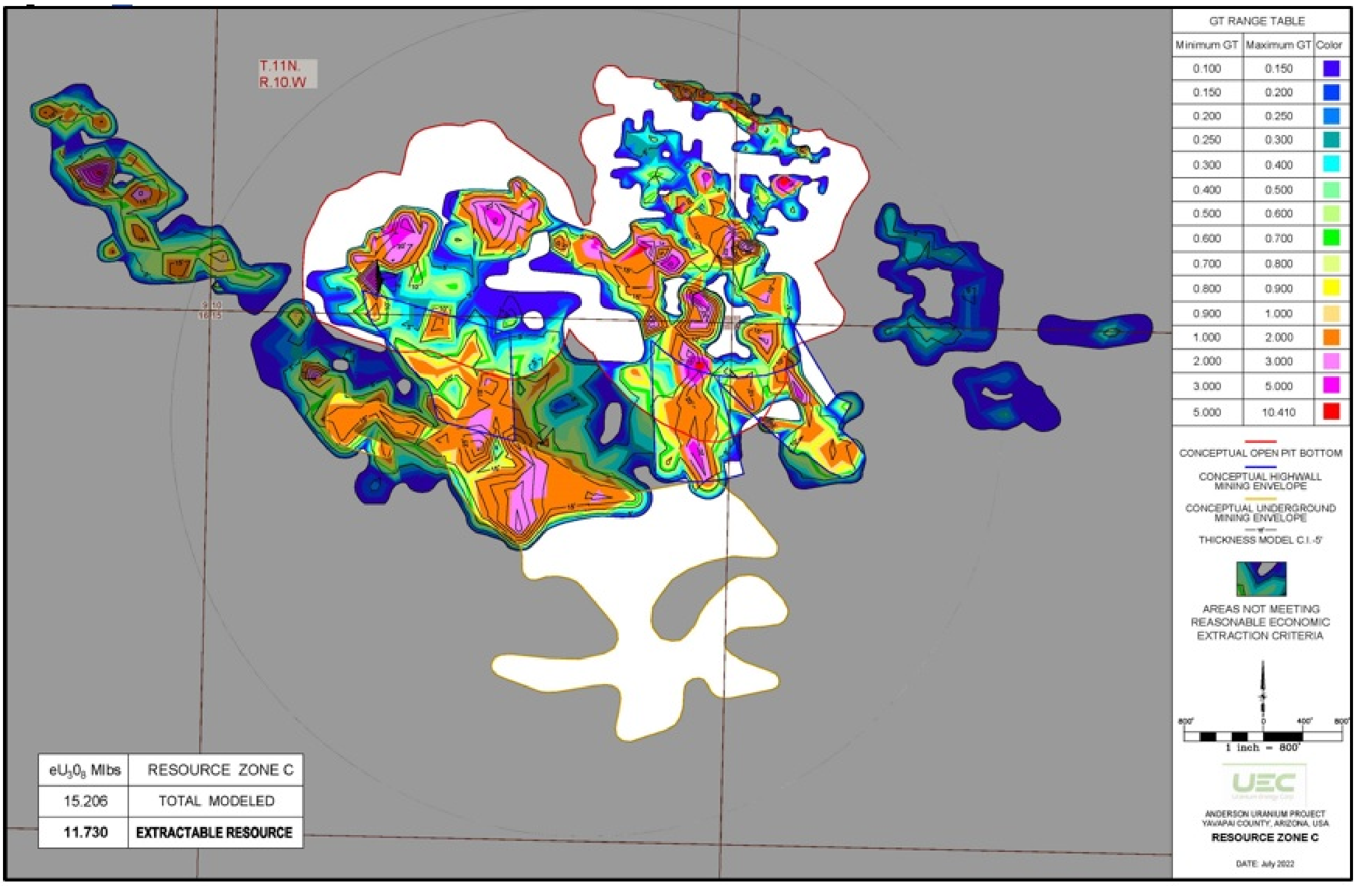


Figure 11-5: GT Model Zone E

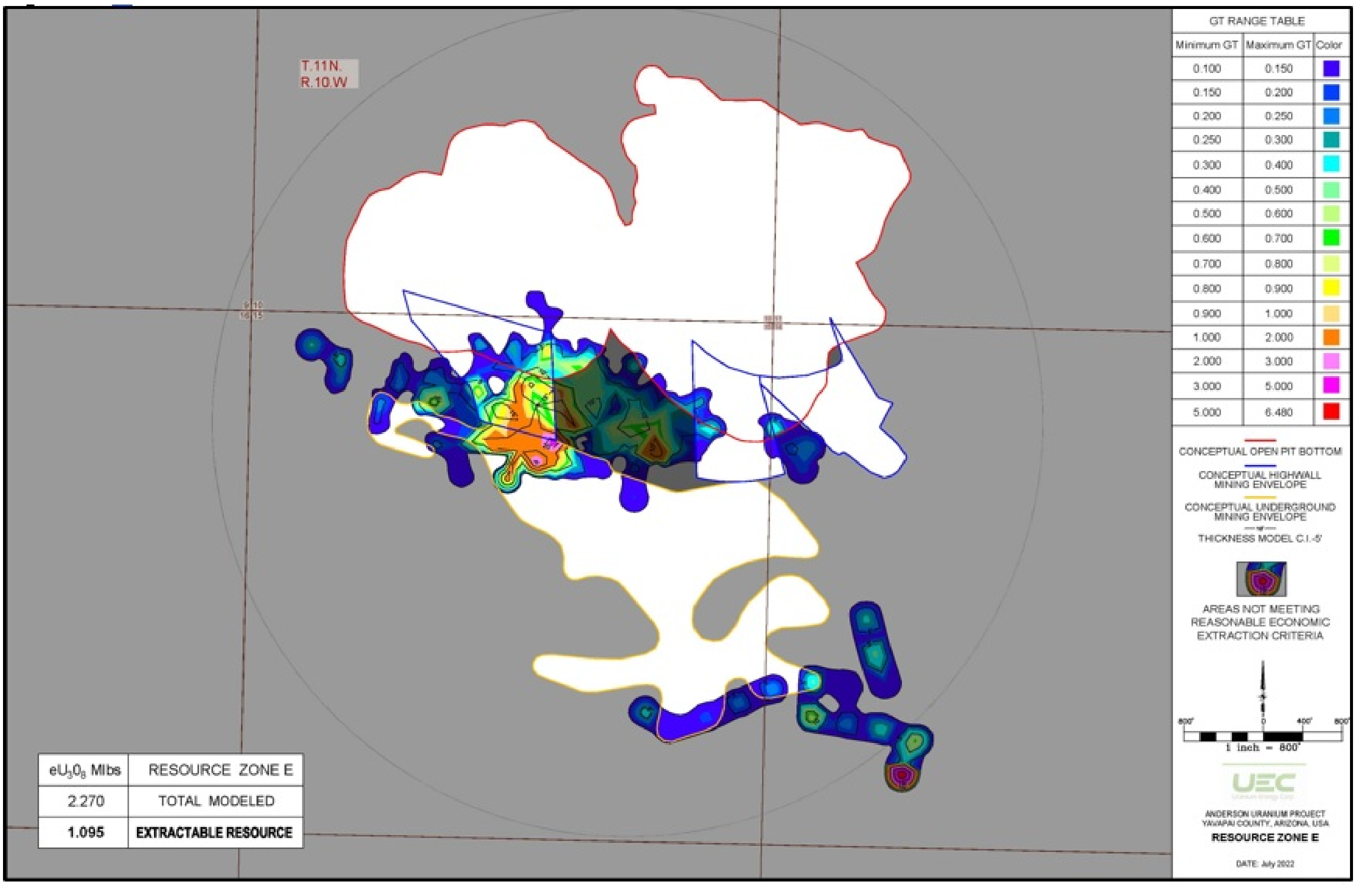


Figure 11-6: GT Model Zone F

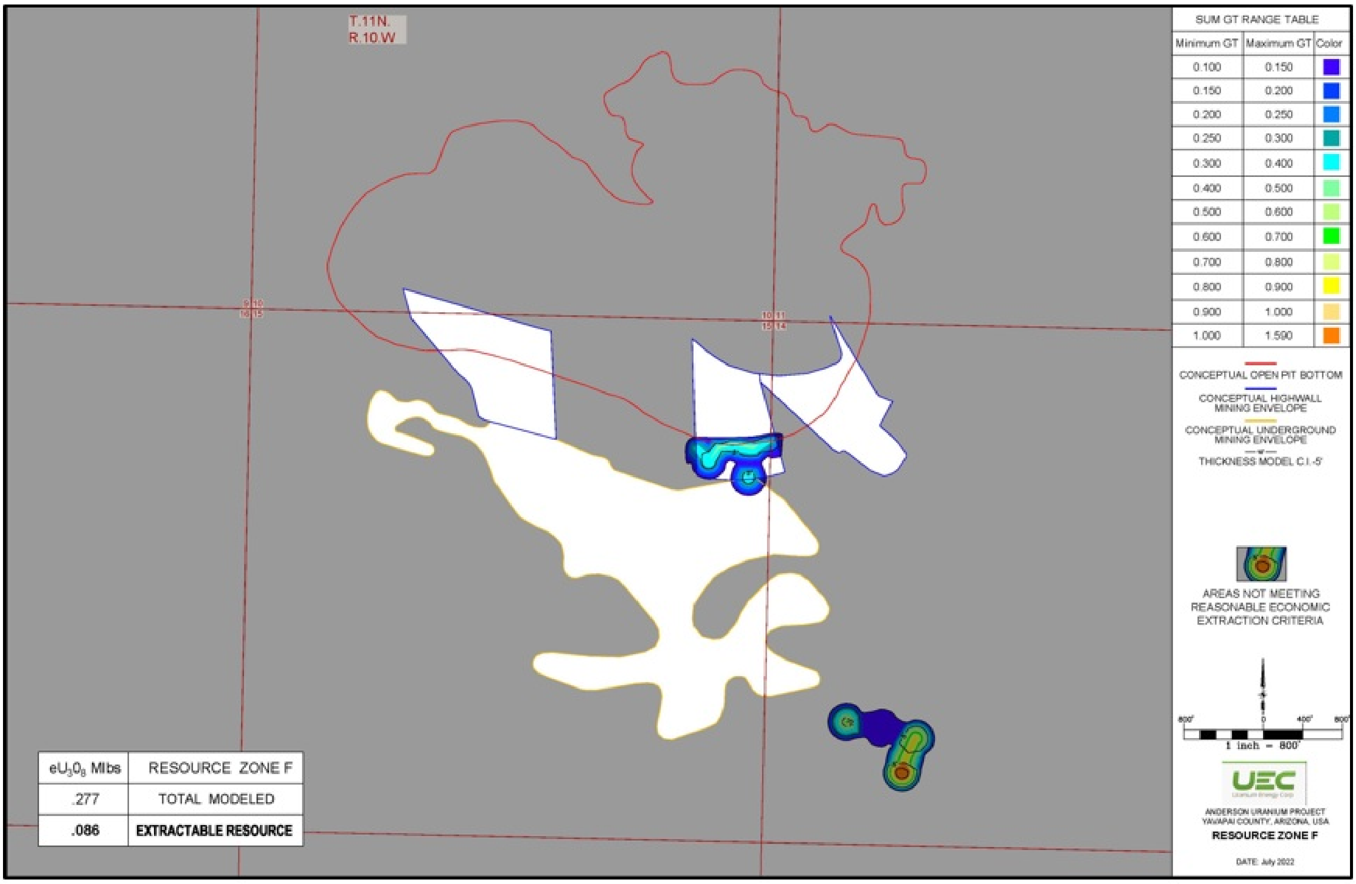
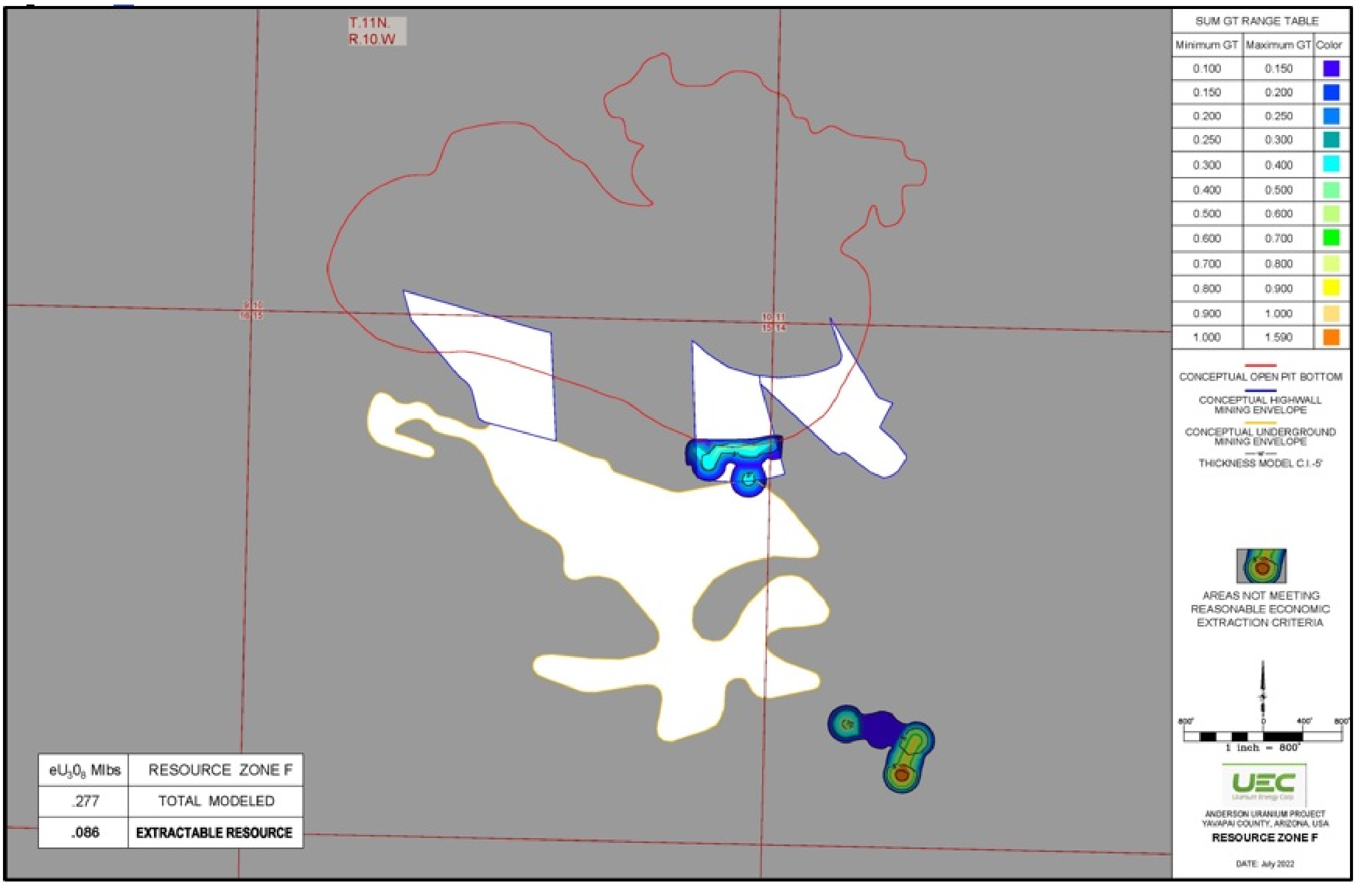


Figure 11-7: GT Model Zone G



12.0 MINERAL RESERVE ESTIMATES

This section is not relevant to this Report.

13.0 MINING METHODS

This section is not relevant to this Report.

14.0 RECOVERY METHODS

This section is not relevant to this Report.

15.0 INFRASTRUCTURE

This section is not relevant to this Report.

16.0 MARKET STUDIES AND CONTRACTS

This section is not relevant to this Report.

17.0 ENVIRONMENTAL STUDIES, PERMITTING, AND PLANS, NEGOTIATIONS, OR AGREEMENTS WITH LOCAL INDIVIDUALS OR GROUPS

This section is not relevant to this Report.

18.0 CAPITAL AND OPERATING COSTS

This section is not relevant to this Report.

19.0 ECONOMIC ANALYSIS

This section is not relevant to this Report.

20.0 ADJACENT PROPERTIES

There are no adjacent properties not held by UEC that are considered relevant to this IA.

21.0 OTHER RELEVANT DATA AND INFORMATION

Neither the authors or UEC are aware of any other data or information which would materially change the conclusions of this report.

22.0 INTERPRETATION AND CONCLUSIONS

This IA for the Project has been prepared in accordance with the regulations set forth in S-K 1300 (Part 229 of the 1933 Securities Act). Its objective is to disclose the mineral resources at the Project.

22.1 Conclusions

Based on the density of drilling, continuity of geology and mineralization, testing and data verification, the mineral resource estimates meet the criteria for indicated mineral resources as summarized herein.

Assumptions regarding uranium prices, mining costs and metallurgical recoveries are forward-looking and the actual prices, costs and performance results may be significantly different. The authors are not aware of any relevant factors that would materially affect the mineral resource estimates. Additionally, the authors is not aware of any specific environmental, regulatory, land tenure or political factors that will materially affect the Project from moving forward to mineral resource recovery operations.

22.2 Risks and Opportunities

UEC has not completed a pre-feasibility nor a feasibility study to apply detailed capital and operational expenditures to the Project. Since these studies have not been completed for the Project, there has not been a formal demonstration of economic and technical capability.

Technical risks related to the Project exist as large-scale conventional mining methods have not been demonstrated in the area. However, these risks are considered to be moderate, as conventional mining and recovery methods are proven in similar sandstone-hosted environments.

Risks related to permitting and licensing the Project exist, as the regulatory process is not mature. However, these risks are considered to be moderate, as a variety of environmental baseline studies have been completed, and no specific impediments to the permitting process are known to the authors.

Mineral tenor is based on the Mining Law, which established the mining claim process in the US. Changes in this law could affect mineral tenor.

Other risk factors are typical for similar mining projects, including, without limitation:

- risks associated with mineral resource estimates, including the risk of errors in assumptions or methodologies;
- geological, technical and processing problems, including unanticipated metallurgical difficulties, less than expected recoveries, ground water control and other factors relating to ISR mining methodology;
- risks associated with labor costs, labor disturbances and unavailability of skilled labor;
- risks associated with the availability and/or fluctuations in the costs of raw materials and consumables used in the production processes;

- risks associated with environmental compliance and permitting, including those created by changes in environmental legislation and regulation, and delays in obtaining permits and licenses that could impact expected mineral extraction and recovery levels and costs; and

- estimation of costs and uranium price for the purposes of constraining the mineral resource estimate based on reasonable prospects for economic extraction.

Readers are cautioned that any estimate of forward cost or commodity price is by its nature forward-looking. It would be unreasonable to rely on any such forward-looking statements and information as creating any legal rights. The statements and information are not guarantees and may involve known and unknown risks and uncertainties, actual results are likely to differ (and may differ materially) and objectives and strategies may differ or change from those expressed or implied in the forward-looking statements or information as a result of various factors. Such risks and uncertainties include risks generally encountered in the exploration, development, operation and closure of mineral properties and processing facilities. Forward-looking statements are subject to a variety of known and unknown risks and uncertainties.

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23.0 RECOMMENDATIONS

The following actions are recommended for the Project with respect to Exploration:

The recommended drilling and assaying will attempt to confirm historic results and upgrade the classification of resources in some areas. Chemical assays will also be used to confirm historic results and determine the propriety of the disequilibrium correction applied to current eU3O8 grades.

The following work items related to additional exploration are recommended for the Project:

Table 23-1: Exploration Budget

Item	Cost (USD)
Permitting and reclamation	\$100,000
5 diamond drill holes (300 ft average, 1,500 ft total)	\$200,000
45 RC and rotary holes (600 ft average 27,000 ft total)	\$800,000
Site supervision including geological services and surveying	\$200,000
Geophysical Logging 50 holes	\$100,000
Assay of core and RC chips (2,000 samples by ICP-MS)	\$150,000
Metallurgical heap leach testing	\$200,000
Resource model update, reporting and preparation of PEA	\$200,000
Road maintenance	\$50,000
Total	\$2,000,000

Based on the successful completion of this program and a decision to move the Project forward to production, the following general recommendations and cost estimates are provided for future reference. Only the exploration program is recommended at this time.

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The following additional work items related to baseline environmental studies are recommended for the Project:

Table 23-2: Environmental Baseline and Related Studies

Item	Cost (USD)
Baseline studies ground water quality	$100,000
Baseline studies surface water quality and sediment surveys	$50,000
Baseline studies air Quality	$150,000
Flora & fauna studies and T&E studies	$100,000
Background radiological studies	$125,000
Archaeological studies	$75,000
Land Use	$25,000
Geology and Overburden	$150,000
Soils and Vegetation for Reclamation Planning	$50,000
Socio-economic studies	$75,000
Section 106 Tribal Consultation	$100,000
Environmental Baseline Total	**$1,000,000**

The recommendations outlined in Tables 26.1 and 26.2 refer to a concurrent work schedule.

Following completion of the previous work items and presuming the Project is proceeding to development, the following work items related to final mine and facility design are recommended:

Table 23-3: Project Design Budget

Item	Cost (USD)
Delineation and Development Drilling	$500,000
Geotechnical Investigations and Design Recommendations	$250,000
Detailed Mine Design and Scheduling	$500,000
Detailed Closure and Reclamation Design and Scheduling	$250,000
Detailed Heap and Plant Design	$500,000
Pilot Scale Heap Leach	$500,000
Feasibility Study	$500,000
Rounded Use	**$3,000,000**

Following completion of the previous work items and presuming the Project is proceeding to development, the following work items related to final mine and facility design are recommended:

Table 23-4: Environmental Baseline and Permitting

Item	Cost (USD)
BLM Plan of Operations and Environment Impact Statement (Mine)	$1,000,000
State and Local Mine and Related Permitting	$500,000
U. S. NRC Licensing and Environmental Impact Statement (Mill)	$1,500,000
Environmental Baseline Total	**$3,000,000**

The recommendations outlined in Tables 23-3 and 23-4 related to final design and development and permitting and licensing would need to be implemented on concurrent work schedules.

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24.0 REFERENCES

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25.0 RELIANCE ON INFORMATION PROVIDED BY THE REGISTRANT

Reliance on information provided by UEC is identified in Table 25-1 below.

Table 25-1: Information Provided by the Registrant

Category of Information	Section of Report	Reason
Macroeconomic trends, data, and assumptions	Section 11	UEC provided data regarding future commodity price estimates. The authors believe that it is reasonable to rely on this information, as it was sourced from industry consultants who specialize in uranium price forecasting.

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26.0 DATE AND SIGNATURE PAGE

CERTIFICATE OF AUTHOR

I, Clyde L. Yancey, Texas Professional Geologist, of 1846 Tramway Terrace Loop, Albuquerque, New Mexico, do hereby certify that:

- I am currently employed by Uranium Energy Corporation, 500 N. Shoreline, Suite 800N, Corpus Christi, Texas, USA, as Vice President of Exploration.
- I graduated with a Bachelor of Arts degree in Geology in May 1975 from Trinity University in San Antonio, Texas, and a Master of Science degree in Geology in May 1978 from South Dakota School of Mines and Technology, Rapid City, South Dakota.
- I am a licensed Professional Geologist in the State of Texas. My registration number is 129 and I am a member in good standing. I am a Registered Member of the Society of Mining, Metallurgy and Exploration. My Registration Number is 03580620 and I am in good standing.
- I have worked as a geologist for over 40 years in uranium exploration, production and restoration.
- My direct experience with uranium involves uranium exploration, resource analysis, uranium ISR project development, project feasibility and licensing, and project closures. My relevant experience for the purpose of this analysis includes Field Geologist for the U.S. Geological Survey, Branch of Uranium and Thorium Resources, Senior Geologist for Wyoming Minerals Corporation at their Bruni, Texas ISR Mine; Mine Geologist for Caithness Mining Corporation at their McBride ISR Mine in Duval County, Texas; Exploration Geologist for Mobil Oil/Nufuels Uranium Division, South Texas District; Senior Uranium Exploration Geologist for Moore Energy Corporation, South Texas District; Consulting Geologist for the U.S. Department of Energy, Uranium Mill Tailings Remedial Action (UMTRA) Title I. Consulting Geologist for UMTRA Title II clients such as Umetco Minerals, Rio Algom Mining, Conoco Minerals, and the Hopi Tribe. Vice President of Exploration for Uranium Energy Corporation with responsibility of their projects in Canada, U.S.A. and South America.
- I have read the definition of “qualified person” set out in Subpart 1300 of Regulation S-K (S-K 1300) and certify that by reason of my education, professional registration, and relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of S-K 1300.

Dated this 1st day of July 2022

Clyde L. Yancey, P.G., SME Register Member

I, Douglas L. Beahm, P.E., P.G., do hereby certify that:

1. I am the Principal Engineer and President of BRS, Inc., 1130 Major Avenue, Riverton, Wyoming 82501.
2. I am a co-author author of the report "ANDERSON URANIUM PROJECT INITIAL ASSESSMENT".
3. I graduated with a Bachelor of Science degree in Geological Engineering from the Colorado School of Mines in 1974. I am a licensed Professional Engineer in Wyoming, Colorado, Utah, and Oregon; a licensed Professional Geologist in Wyoming; a Registered Member of the SME.
4. I have worked as an engineer and a geologist for over 48 years. My work experience includes uranium exploration, mine production, and mine/mill decommissioning and reclamation. Specifically, I have worked with numerous uranium projects hosted in sandstone environments in Wyoming.
5. I was last present at the site on the 17th and 18th of December, 2013.
6. I am independent of the issuer. I hold no stock, options or have any other form of financial connection to UEC. UEC is but one of many clients for whom I consult.
7. I do have prior working experience on the property as stated in the report.
8. I have read the definition of "qualified person" set out in Subpart 1300 of Regulation S-K (S-K 1300) and certify that by reason of my education, professional registration, and relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of S-K 1300.

Dated this 1st day of July 2022

Douglas L. Beahm, PE, PG, SME Registered Member

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