URANIUM ENERGY CORP

TECHNICAL REPORT and PEA on the ANDERSON URANIUM PROJECT

Yavapai County, Arizona, USA

NI 43-101 Technical Report

Report Prepared for:

Uranium Energy Corp
500 North Shoreline
Suite 800N
Corpus Christi, TX 78401

Qualified Persons/Authors:
Douglas Beahm, PE, PG
BRS Inc.
1130 Major Avenue
Riverton, Wyoming 82501

Terence P. McNulty, D. Sc., P.E.
T. P. McNulty and Associates
4550 North Territory Place
Tucson, Arizona 85750

Bruce Davis, FAusIMM
BD Resource Consulting, Inc.
4235 Cheyenne Drive
Larkspur, CO 80118

Robert Sim P.Geo
SIM Geological Inc.
6810 Cedarbrook Place
Delta, BC V4E 3C5

Effective Date: July 6, 2014 **Signature Date: July 31, 2014**

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

APPENDIX A - "Scoping Metallurgical Testing of Anderson Mine Samples" dated May 30, 2014, Resource Development Inc.

1 EXECUTIVE SUMMARY

This Technical Report was prepared by BD Resource Consulting Inc. (BDRC), BRS Inc. and T.P. McNulty and Associates for Uranium Energy Corporation (UEC) and provides an updated mineral resource and Preliminary Economic Assessment (PEA) for the Anderson Uranium Project (Anderson Project). The report was written under the direction of Robert Sim P.Geo., Bruce Davis, F.AusIMM, Douglas Beahm, P.E, P.G, and Dr. Terry McNulty, P.E.; who are independent "qualified persons" as defined by CSA National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and described in Section 28.

The Anderson Project covers 9,852 acres (15.4 square miles) and is comprised of 459 contiguous, unpatented lode mining and placer claims and two Arizona State land sections. It is located in western Yavapai County approximately 75 miles northwest of Phoenix and approximately 180 miles from UEC's Workman Creek Project. The northern section of the Anderson Project area holds the open-pit resource and the adjacent southern section holds the underground resource.

In May 2011, UEC acquired the Anderson property after their merger with Concentric Energy Corporation (Concentric). According to the terms of the agreement, UEC Concentric Merge Corp., a wholly-owned subsidiary of UEC, was vested with all of Concentric's assets and property. At this same time, Uranium Energy Corp. completed the full assignment to UEC of Global Uranium Corporation (Global) rights under the terms and conditions of an underlying Option and Joint Venture Agreement dated 13 April 2010 between Global and Concentric with respect to the Anderson property.

In January 1955, T. R. Anderson of Sacramento, California detected anomalous radioactivity in the vicinity of the Anderson 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% U_3O_8 and 33,230 pounds of U_3O_8 (Table 6.1) were shipped to Tuba City, Arizona for custom milling. In 1959, production stopped when the Atomic Energy Commission (AEC) ended the purchasing program.

Between 1967 and 2006, the property was explored by several companies including: Getty Oil Company, Urangesellschaft USA, Inc., Minerals Exploration Company, and Concentric Energy Corp. Details of exploration activity are presented in Table 1.1.

TABLE 1.1: EXPLORATION HISTORY

Company	Period	Exploration Activities
Mining Group Led by Mr. T. R. Anderson	1955−1959	Aerial scintillometer surveying, ground prospecting, and outcrop mining
Getty Oil Company	1967−1968	Limited exploration drilling
Urangesellschaft U.S.A., Inc.	1973−1982	Exploration drilling: 352 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 235 ha area
Concentric	2006	Confirmation drilling 24 RC holes and one RC-core

The Anderson 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).

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). Mineralization is also prevalent in calcareous facies.

A review of the sample collection and analysis practices used during the various drilling campaigns indicates that this work was conducted using procedures which are accepted within the industry. 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. 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 qualified person's opinion that the sample database is of sufficient accuracy and precision to generate a mineral resource estimate.

The uranium mineral resource estimate was based on a total of 202,707 meters 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. The mineral resource estimate was generated from drill hole sample assay results and the interpretation of a geologic model which relates to the spatial distribution of uranium in the deposit. Interpolation characteristics were defined based on the geology, drill hole spacing and geostatistical analysis of the data. U_3O_8 grades were estimated in the resource model using ordinary kriging with a nominal block size measuring 10 meters long, 10 meters wide and 2 meters high.

Potentially anomalous outlier grades were identified and their effects were controlled during interpolation. Average bulk density values, based on analysis of available data, were used to estimate resource tonnage.

The results of the modeling process were validated using a series of methods; the results indicate that the resource model is an appropriate estimation of global resources based on the underlying database.

The resources were classified by their proximity to sample locations and are reported according to the Canadian Institute of Mining, Metallurgy and Petroleum's definition standards on Mineral Resources and Reserves.

As required under NI 43-101, mineral resources must exhibit reasonable prospects for economic viability. This report segregates resources into two types: those potentially amenable to open pit extraction methods versus deeper resources that would be exploited through underground methods. The 2012 Anderson mineral resource estimates are summarized at various cut-off grades for comparison purposes in Table 1.2. The "base case" cut-off grade of 0.01% eU_3O_8 for potential open pit mineralization and 0.035% eU_3O_8 for potential underground mineralization is bolded in each section of Table 1.2. These assumptions are derived from operations with similar characteristics, scale and location. Note that the mineral resources stated below are not mineral reserves as they have not demonstrated economic viability.

There are no known factors relating to environmental, permitting, legal title, taxation, socio-economic, marketing or political issues which could materially affect the mineral resource estimates.

TABLE 1.2: SUMMARY OF MINERAL RESOURCES

Cut-off Grade eU_3O_8%	K tonnes	K tons	eU_3O_8 (%)	Contained U_3O_8 (Mlbs)
	OPEN PIT RESOURCES			
	INDICATED			
0.005	28,034	30,902	0.026	16.0
0.01	**25,422**	**28,023**	**0.028**	**15.5**
0.015	19,834	21,863	0.032	14.0
0.02	15,008	16,543	0.037	12.3
0.025	11,355	12,517	0.042	10.5
0.03	8,584	9,462	0.047	8.9
0.035	6,445	7,104	0.052	7.3
	INFERRED			
0.005	5,478	6,038	0.022	2.6
0.01	**4,633**	**5,107**	**0.024**	**2.5**
0.015	3,341	3,683	0.029	2.2
0.02	2,324	2,562	0.035	1.8
0.025	1,670	1,841	0.040	1.5
0.03	1,192	1,314	0.045	1.2
0.035	897	989	0.049	1.0
	UNDERGROUND RESOURCES			
	INDICATED			
0.005	38,177	42,083	0.015	12.5
0.01	25,209	27,788	0.019	10.5
0.015	15,213	16,769	0.024	7.9
0.02	8,570	9,447	0.029	5.4
0.025	4,494	4,954	0.035	3.5
0.03	2,555	2,816	0.042	2.3
0.035	**1,426**	**1,572**	**0.049**	**1.5**
0.04	871	960	0.057	1.1
0.045	581	640	0.065	0.8
0.05	382	421	0.074	0.6
	INFERRED			
0.005	123,286	135,899	0.016	44.6
0.01	85,483	94,229	0.021	38.8
0.015	52,298	57,649	0.026	30.2
0.02	32,330	35,638	0.032	22.8
0.025	20,423	22,512	0.038	17.1
0.03	12,164	13,408	0.045	12.1
0.035	**8,362**	**9,218**	**0.052**	**9.5**
0.04	6,046	6,665	0.057	7.6
0.045	3,572	3,937	0.068	5.4
0.05	2,896	3,192	0.073	4.7

Notes: "Base case" cut-off for resources amenable to open pit extraction methods is 0.01%U_3O_8. "Base case" for remaining resources extracted by underground mining methods is 0.035%U_3O_8. Effective Date of Mineral Resource Estimate April 15, 2012.

In the late 1970s and early 1980s, metallurgical testwork was done by both Minerals Exploration Company (MinEx) and Urangesellschaft USA, Inc. (Urangesellschaft). At that time, acid and alkaline leaching tests were researched, but never completed because the property was dropped.

Limited testing of acid-heap leaching was done by Urangesellschaft. Mineralized material was leached using a re-circulating 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. Following that, the flow rate and effluent's uranium content drastically decreased, and after 45 days of leaching, an overall solubilization of 79% was achieved (Urangesellschaft, 1978).

Conclusions

Based on the recent assembly and verification of data by UEC on the Anderson Project, the following conclusions can be made:

Mineral Resources:

- The level of understanding of the geology is relatively good.
- The practices used during the various drilling campaigns were conducted in a professional manner and adhered to accepted industry standards.
- There are no evident factors that would lead one to question the integrity of the database.
- A significant uranium deposit was outlined. Mineralization is hosted in lacustrine facies fixed by the presence of carbonaceous material.
- Drilling to date has outlined an Indicated open pit resource (at a 0.01% eU_3O_8 cut-off) of 25.4 Mtonnes (28.0 M tons) at 0.028% eU_3O_8 which contains 15.5 million pounds of uranium and an Inferred resource (at a 0.01% eU_3O_8 cut-off) of 4.6 Mtonnes (5.1 M tons) at 0.024% eU_3O_8 which contains 2.5 million pounds of uranium.
- The underground Indicated resource (at a 0.035% eU_3O_8 cut-off) is 1.4 Mtonnes (1,6 M tons) at 0.049% eU_3O_8 which contains 1.5 million pounds of uranium and an Inferred resource (at a 0.035% eU_3O_8 cut-off) of 8.4 Mtonnes (9.2 M tons) at 0.052% eU_3O_8 which contains 9.5 million pounds of uranium.

Preliminary Economic Assessment:

- Conceptual mine plans were developed for a conventional mine operation which includes open pit, highwall, and underground mining. Portions of the current mineral resource, both indicated and inferred mineral resources, were included within the conceptual mine designs for the PEA. The indicated and inferred mineral resources used in the PEA are fully included in the total Indicated and Inferred mineral resources reported in Section 14 of this report. They are that portion of the mineral resources which meet minimum cutoff criterion and are incorporated within conceptual mine designs and represent approximately 80% of the mineral resources as stated in Section 14 of the report/

- Conceptual plans were developed for the processing of the mined material via conventional heap leach methods using an acid lixiviant.
- Recent metallurgical testing of mineralized material from on-site stockpiles was completed which indicates greater than 90% recovery with an average acid consumption of 50 pounds per ton of material processed.
- The base case for the Preliminary Economic Assessment (PEA) considers conventional mining in conjunction with on-site heap leach recovery, producing an intermediate uranium concentrate in the form of loaded resin which would be shipped to EFR's White Mesa mill in Utah for final processing. However, once the uranium is concentrated and loaded on resin it could be shipped to other central processing facilities.
- Vanadium is present in the mineralized material. The PEA is based on the recovery of uranium only. Future studies will determine the feasibility of recovering vanadium as a by-product.
- CAPEX for the project is estimated at 8 million $US for pre-production costs, 43.9 million $US for initial capital, and 87.3 million $US for additional capital during operations for a total life-of-mine capital of 139.2 million $US.
- OPEX is estimated at approximately $45/ton or $34/lb U_3O_8 recovered including all operating and reclamation costs exclusive of income tax.
- The current The PEA shows a positive return on investment. Table 22.1 shows the IRR and NPV at various discount rates both before and after taxes.
- This is a restricted disclosure as allowed under section 2.3(3) of NI 43-101 which includes a Preliminary Economic Sssessment (PEA). It is also preliminary in nature such that it includes portions of both indicated and inferred mineral resources, as reported in Section 14 of the report. The PEA is based on open pit mining and heap leach extraction of uranium values, utilizing methodologies, equipment, and a generalized design criterion which has been employed at the site and/or similar sites in the past but has not been specifically developed for the Project. Mineral resources are not mineral reserves and do not have demonstrated economic viability in accordance with CIM standards. Inferred mineral resources are too speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the outcomes estimated in the PEA will be realized.

Recommendations

The following actions are recommended for the Anderson Project:

- Additional drilling to expand confirmation results from historic drilling in both the open pit and underground portions of the deposit. The use of both PFN and chemical assay should be used for the confirmation of grade. A budget of US$780,000 has been proposed to complete this test work (Table 1.3).
- Additional metallurgical testing on both open pit and underground areas. A budget of US$100,000 has been proposed to complete this work (Table 1.3).
- After drilling is completed, an updated resource estimate should be prepared. A budget of US$75,000 has been proposed to complete this work (Table 1.3).
- Environmental studies to provide a baseline for future exploration and possible development work on the project. A budget of $675,000 has been proposed to complete this work (Table 1.4).

Recommended drilling and assaying will aim to further confirm historic results and upgrade the classification of resources in some areas. The Prompt Fission Neutron (PFN) logging 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 26.1: EXPLORATION BUDGET

Item	Cost (USD)
Permitting and reclamation	$25,000
5 diamond drill holes (100 m average, 500 m total)	$150,000
20 RC holes (200 m average 4,000 m total)	$400,000
PFN probing 25 holes	$125,000
Assay of core and RC chips (2,000 samples by ICP-MS)	$88,000
Metallurgical heap leach column testing	$100,000
Resource model update and report	$75,000
Road maintenance	$25,000
Exploration TOTAL	**$988,000**
Rounded Use	**$1,000,000**

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

TABLE 26.2: 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 26.3: 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 26.4: ENVIRONMENTAL BASELINE AND RELATED STUDIES

Item	Cost (USD)
BLM Plan of Operations and Environment Impact Statement (Mine)	$1000,000
State and Local Mine and Related Permitting	$500,000
U. S. NRC Licensing and Environmental Impact Statement (Mill)	$1500,000
Environmental Baseline TOTAL	**$3,000,000**

The recommendations outlined in Tables 26.3 and 26.4 related to final design and development and permitting and licensing would need to be implemented on concurrent work schedule.

2 INTRODUCTION

Mineral Resource Estimation and Reporting:

Uranium Energy Corporation (UEC) commissioned Robert Sim, P.Geo. of SIM Geological Inc. and Bruce Davis, F.AusIMM of BD Resource Consulting Inc. (BDRC) to provide an updated mineral resource for the Anderson Uranium Project (Anderson Project). Robert Sim, P. Geo. and Bruce Davis, F.AusIMM are both independent "qualified persons", within the meaning of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101). They are responsible for the preparation of Sections 4, 5, 6, 7, 8, 9, 10,11,12 and 14, and contributed to Sections 1, 2, 3, 13, 23, 24, 25, 26, and 27 of this Technical Report on the Anderson property (Technical Report) which has been prepared in accordance with NI 43-101 and Form 43-101F1.

Bruce Davis, F.AusIMM visited the site on 8 May 2012, inspected uranium mineralization in outcrop, reviewed sampling procedures, inspected historical information and visited selected drill sites.

To prepare this Technical Report, the authors relied on geological reports, maps and miscellaneous technical papers listed in the References section of this Technical Report. This report is based on drilling and sampling data available as of 01 April 2012. The resource model, including subsequent validation and review, was completed in mid-April and released in a UEC press release on 8 May 2012.

The information in Sections 4, 5, 6, 7, 8, 10, 11 and 13 is taken from the Agapito NI 43-101 (Gilbride *et al.*, 2010) and the SRK NI 43-101 (Arseneau, 2011).

The effective date for the mineral resource estimate is 15 April 2012.

Preliminary Economic Assessment:

UEC commissioned BRS Inc. and T. P. McNulty and Associates to complete a PEA for the Anderson Project and act as co-authors with Robert Sim, P.Geo. of SIM Geological Inc. and Bruce Davis, F.AusIMM of BD Resource Consulting Inc. (BDRC) and to update the previous technical report for the project. Douglas Beahm, P. E., P. G., BRS Inc., and Dr. Terry McNulty, P. E., T. P. McNulty and Associates, are co-authors of the technical report and are both independent "qualified persons", within the meaning of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101). They are responsible for the preparation of Sections 15, 16, 17, 18, 19, 20, 21, and 22, and contributed to Sections 1, 2, 3, 13, 23, 24, 25, 26, and 27 of this Technical Report on the Anderson property (Technical Report) which has been prepared in accordance with NI 43-101 and Form 43-101F1.

Mr. Beahm visited the site on the 17th and 18th of December, 2013. Dr. McNulty visited the site on the 18th of December, 2013. During this period the authors 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. 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.

This is a restricted disclosure as allowed under section 2.3(3) of NI 43-101 which includes a Preliminary Economic Assessment (PEA). It is also preliminary in nature such that it includes portions of both indicated and inferred mineral resources, as reported in Section 14 of the report. The PEA is based on open pit mining and heap leach extraction of uranium values, utilizing methodologies, equipment, and a generalized design criterion which has been employed at the site and/or similar sites in the past but has not been specifically developed for the Project. Mineral resources are not mineral reserves and do not have demonstrated economic viability in accordance with CIM standards. Inferred mineral resources are too speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the outcomes estimated in the PEA will be realized.

The effective Date of the PEA and associated cost model is July 6, 2014.

Unless otherwise noted, all historic measurements reported in this document are provided in both metric and US units (. All currency in this report is expressed in US dollars (US$) unless otherwise noted. Table 2.1 provides metric/English conversions and abbreviations. Abbreviations and acronyms used in the report are provided in Table 2.2.

TABLE 2.1: METRIC/US UNITS AND CONVERSIONS

GENERAL TERMS AND ABBREVATIONS					
	METRIC		US		Metric : US
	Term	Abbreviation	Term	Abbreviation	Multiply By
Area	Square Meters	m^2	Square Feet	Ft^2	10.76
	hectare	Ha	Acre	Ac	2.47
Volume	Cubic Meters	m^3	Cubic Yards	Cy	1.308
	Cubic Meters	m^3	Cubic feet	CF	35.315
Length/Distance	Meter	m	Feet	Ft	3.28
	Meter	m	Yard	Yd	1.09
	Kilometer	km	Mile	mile	0.6214
	Centimeter	cm	Inch	in	0.394
Weight	Kilogram	Kg	Pound	lb	2.20
	Tonne	t	Short Ton	Ton	1.1023
Density	Tonnes per Meter^3	t/m^3	Pounds per Cubic Foot	Lbs/CF	12.95
URANIUM SPECIFC TERMS AND ABREVATIONS					
Grade	Parts Per Million	ppm U_3O_8	Weight Percent	$\%U_3O_8$	
Radiometric Equivalent Grade		ppm eU_3O_8		$\% eU_3O_8$	
Thickness	meters	m	Feet	Ft	
Grade Thickness Product	grade x meters	GT(m)	grade x feet	GT(Ft)	
Counts per Sceond		CPS		CPS	

TABLE 2.2: LIST OF ABBREVIATIONS AND ACRONYMS

.txt	Text file
°F	Degree Fahrenheit
ASCII	American Standard Code for Information Interchange
cfm	Cubic feet per minute
dpi	Dots per inch
ft^2/tpd	Feet squared per tons per day
ft^3/ton	Cubic foot per short ton
gph	Gallons per hour
gpm	Gallons per minute
gpt	Grams per tonnes
KB	Kilobyte
kg/ton	Kilograms per ton
ktonnes	Kilotonnes
lb/ton	Pounds per ton
MB	Megabyte
Mlbs	Million pounds
Mtonnes	Million tonnes
M tons	Million tons
PLS	Pregnant leach solution
RAS	Radial arm stacking
UTS	Uranyltrisulfate
UDS	Uranyldisulfate
IDW	Inverse distance weighted
NN	Nearest neighbor
PFN	Prompt fission neutron
QA/QC	Quality assurance/quality control
TDH	Total dynamic head
TIFF	Tagged image file format
tpd	Tons per day
NEPA	National Environmental Policy Act
MSHA	Mine Safety and Health Administration
SHPO	State Historical Perseveration Officer
T&E	Threatened and Endangered
TENORM	Technically Enhanced Naturally Occurring Radioactive Materials
U_3O_8	Uranium Oxide
US$	US dollar

3 RELIANCE ON OTHER EXPERTS

The report was prepared by Robert Sim, P.Geo. of SIM Geological Inc. and Bruce Davis, F.AusIMM of BD Resource Consulting Inc. (BDRC), both independent “qualified persons” for the purposes of NI 43-101. The information, conclusions, opinions and estimates contained herein are based on the qualified person’s field observations and data, reports and other information supplied by UEC and other third parties.

For the purpose of Sections 4.1 (Property Location) and 4.2 (Property Ownership) of this report, BDRC relied on the ownership data (mineral, surface and access rights) provided by UEC (Harris and Thompson, 2007). BDRC believes that this data and information are essentially complete and correct to the best of its knowledge and that no information has been intentionally withheld that would affect the conclusions made herein. BDRC has not researched the property title or mineral rights for the Anderson Project and expresses no legal opinion as to the ownership status of the property.

BRS relied on the mineral resource database prepared and verified by BDRC to estimate of mineral resources included within conceptual mine designs and the PEA. With this exception, neither BRS nor T. P. McNulty and Associates relied on others in the preparation of the portions of the report relating to the PEA.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

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 through 36 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.

4.2 OWNERSHIP

The Anderson Project comprises the majority of the claim positions historically held by Minerals Exploration Company (MinEx) and Urangesellschaft USA, Inc. (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 Energy Corp. (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 property. Under the terms of the agreement, Global would earn a 70% in the Anderson 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 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, Uranium Energy Corp. (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.

Pursuant to the merger agreement:

1. Concentric's stockholders received 0.1075 of one share of UEC's common stock for every one share of Concentric's common stock. With 11,659,905 shares of Concentric common stock outstanding immediately prior to the merger, UEC issued 1,253,440 of its own common shares to former Concentric stockholders. This represented approximately 1.7% of the issued and outstanding stock of UEC.

2. UEC issued 375,834 common stock purchase warrants to the former holders of Concentric's common stock purchase warrants based on the exchange ratio of 0.1075 of one UEC Exchange Warrant for every one Concentric Exchange Warrant. The exercise price of each UEC Exchange Warrant was determined by dividing the per share exercise price of the corresponding Concentric Warrant by 0.1075, yielding exercise prices that ranged between $9.30 and $65.12.

3. At this same time, UEC completed the full assignment to UEC of Global's rights under the terms and conditions of an underlying Option and Joint Venture Agreement dated 13 April 2010 between Global and Concentric with respect to the Anderson property. Pursuant to an Acquisition Agreement with Global to acquire the rights, UEC delivered to Global an initial payment of $150,000, an additional $200,000 that released and assigned to UEC the security previously assigned to Global by Concentric, and 350,000 restricted shares of UEC's stock along with a final payment of $150,000.

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). The entire claim block and the two State mineral leases comprise an area of approximately 9,852 acres (15.4 square miles).

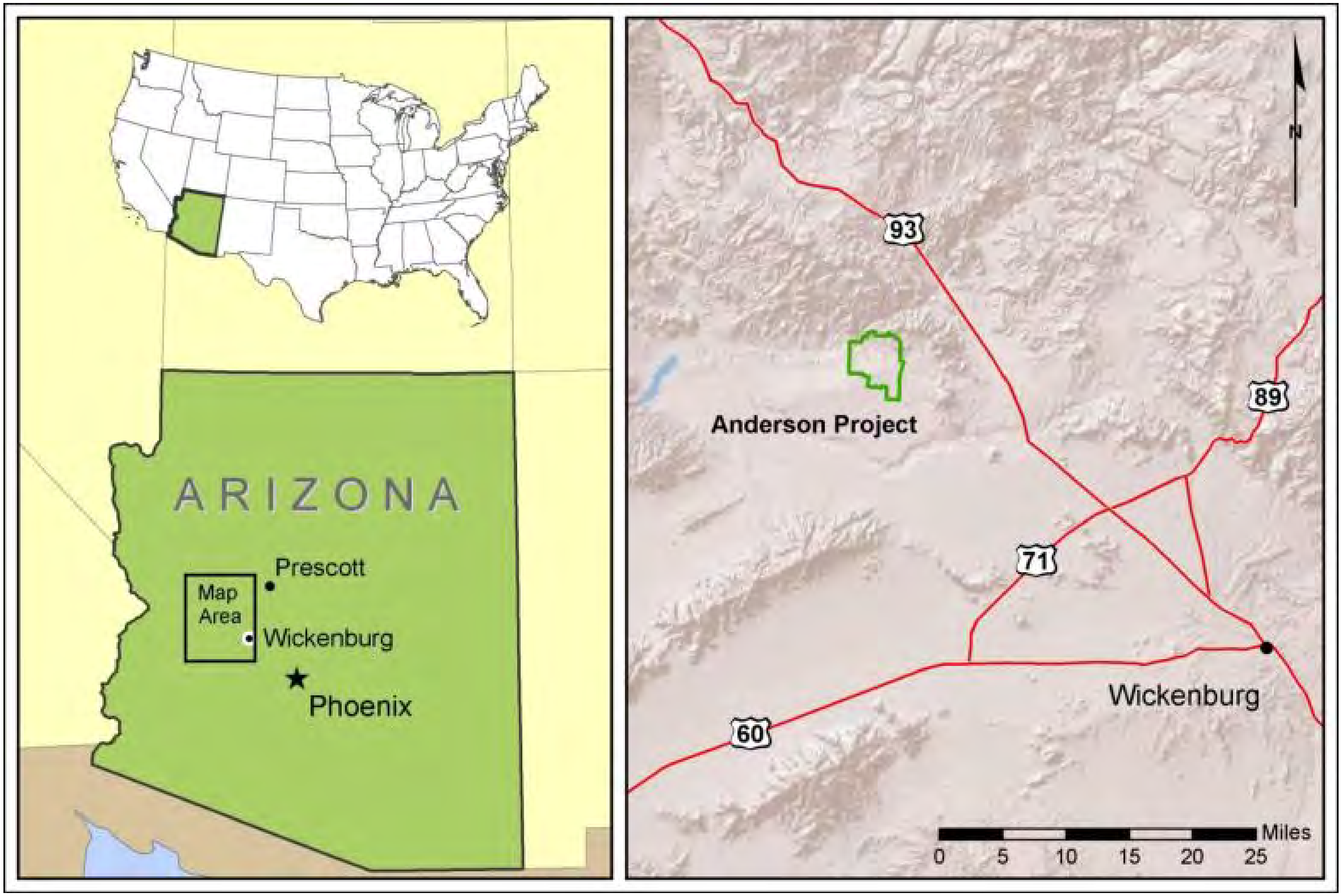


FIGURE 4-1: PROJECT LOCATION MAP

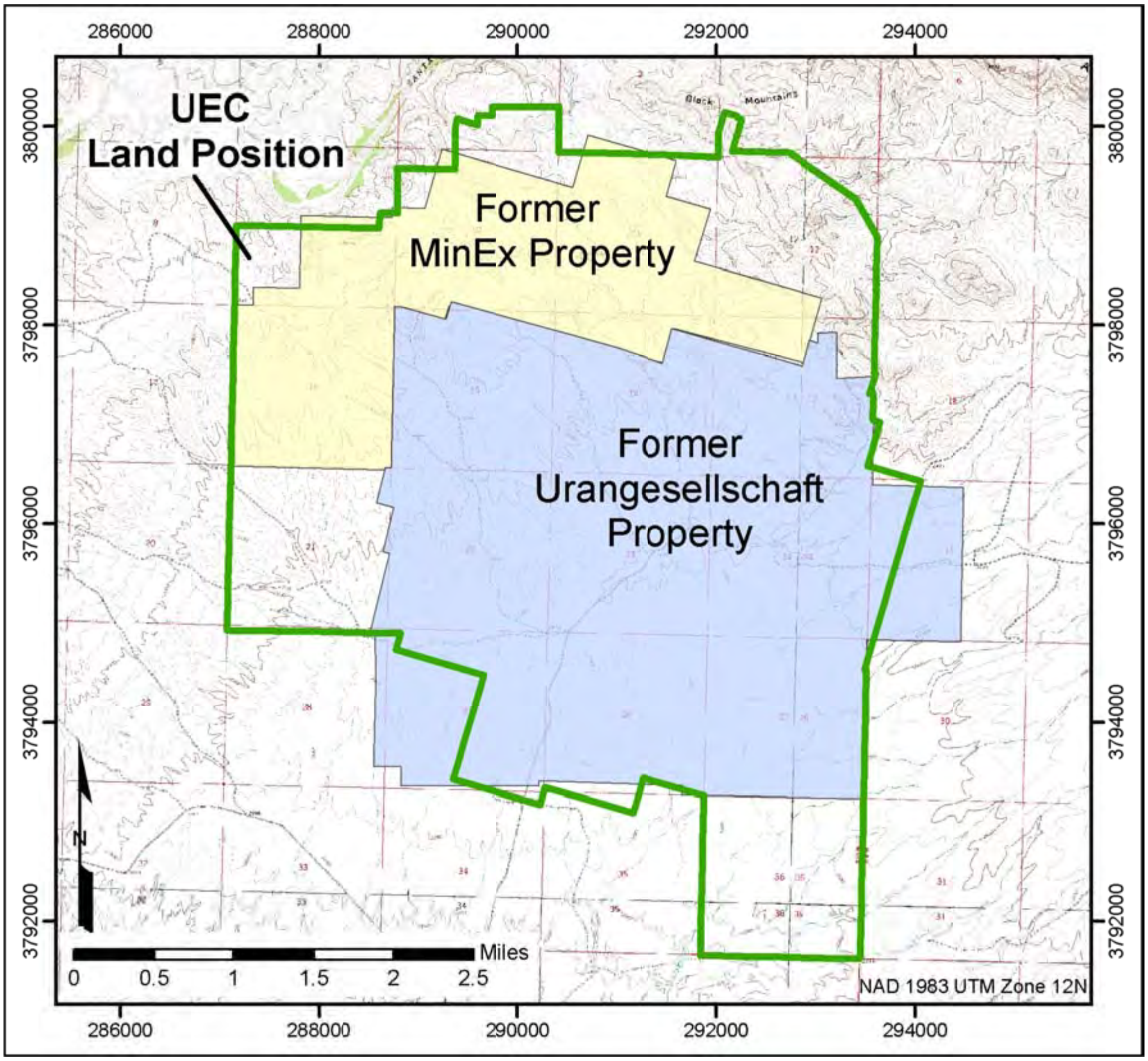


FIGURE 4-2: PROJECT LAND POSITION MAP

4.3 MINERAL TITLES

Unpatented mining claims, either lode or placer, are located under the authority of the Mining Law of 1872 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 $140 per claim are due on September 1 of each year. Unpatented federal lode mining claims in Arizona are designated in the field by four corner posts, two end-center posts and a location monument. Claim location notices for each unpatented claim are recorded in the county recorder’s office of the county in which the claims are located, and then filed with the BLM Arizona State office.

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 exploration expenditure requirement. Each state section requires a separate exploration permit.

4.4 SURFACE RIGHTS

All of the Anderson 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.

4.5 MINERAL EXPLORATION PERMITTING

Exploration and mining activities for the mining claims of the Anderson 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 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.

4.6 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.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Anderson 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 Anderson 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 5-1).

5.2 PHYSIOGRAPHY

The Anderson Project is located in the northeast portion of the Date Creek Basin (Figure 5-2). 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 Anderson 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 Anderson Project drains southward to Date Creek Wash. Both the Santa Maria River and Date Creek Wash are generally intermittent. Significant flows can occur in 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 include: jackrabbits, rattlesnakes, roadrunners, desert tortoise, various lizards, and less common 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).

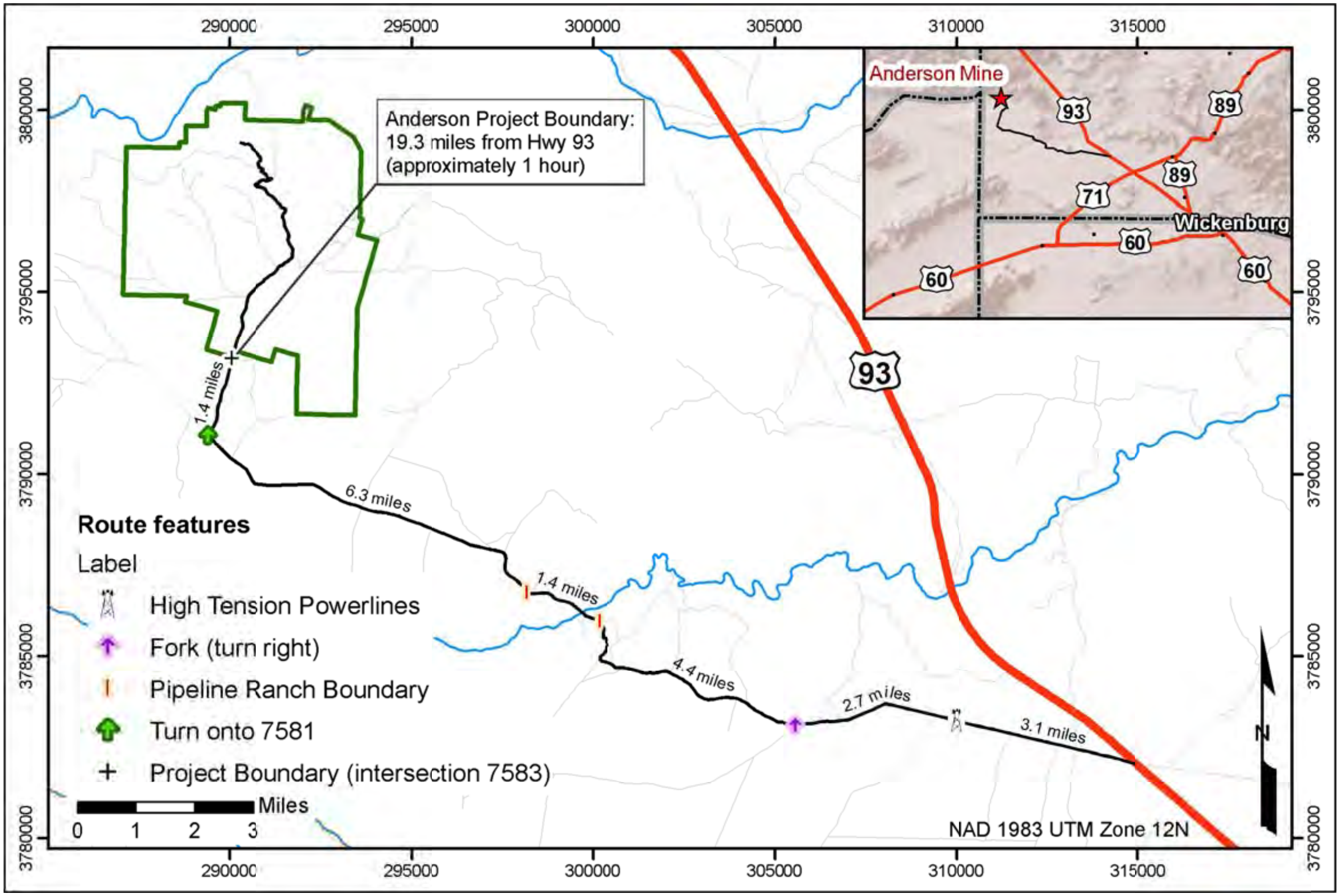


FIGURE 5-1: PROJECT ROAD ACCESS MAP

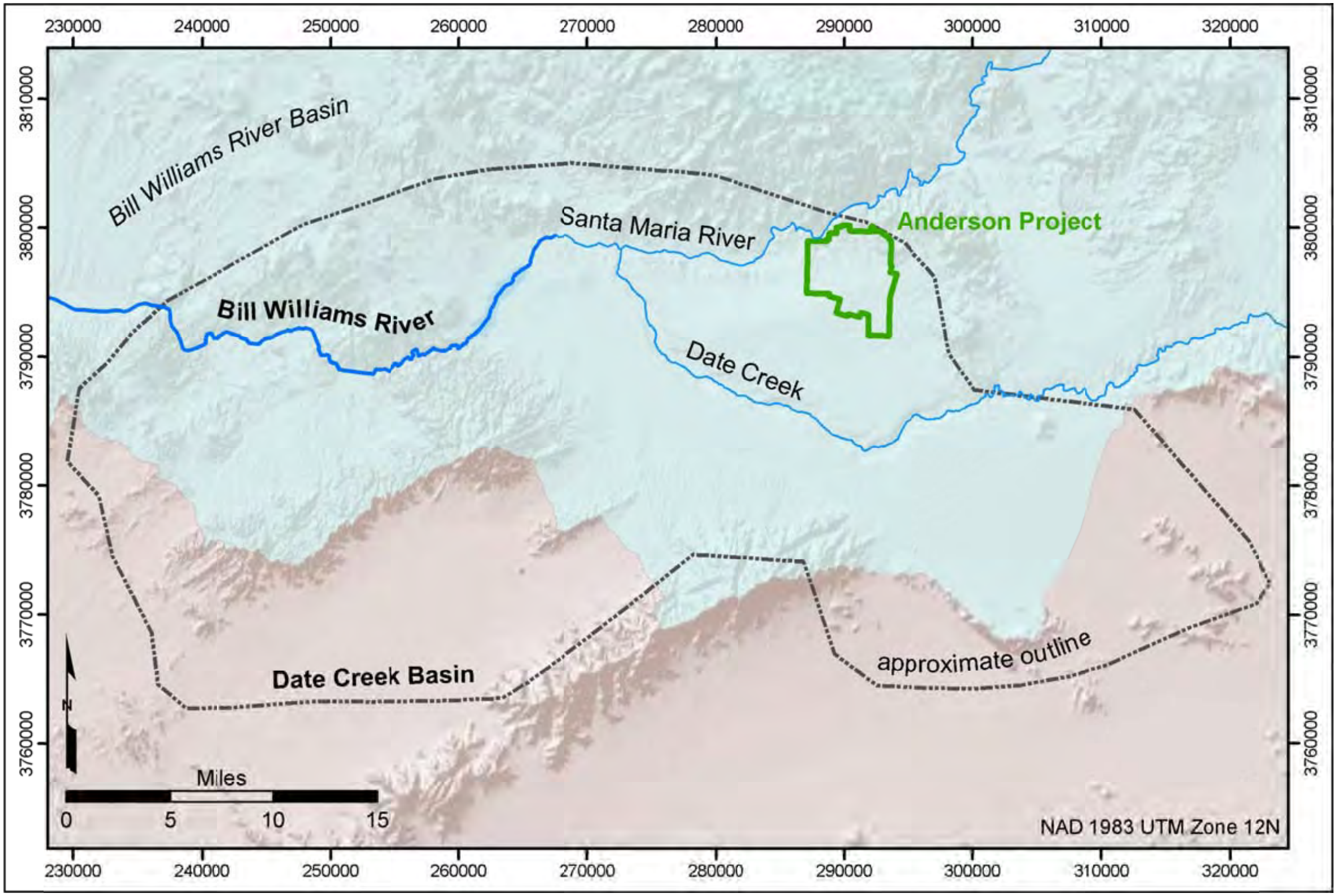


FIGURE 5-2: PROJECT LOCATION WITHIN DATE CREEK BASIN

5.3 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 during summer thunderstorms. 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.

5.4 LOCAL RESOURCES

Various water wells exist on and near the Anderson 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 Anderson 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.

Both Urangesellschaft and MinEx investigated the groundwater in the immediate vicinity of the Anderson Project. Hydrologic studies were conducted by the Water Development Corporation of Tucson, Arizona in 1978 and 1979.

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 in this zone by MinEx yielded an average flow rate of 57 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). This low yield did not warrant the calculation of formation constants. Drill holes to the south on the former Urangesellschaft property yielded an average flow of 5 to 10 gpm from the same unit (Urangesellschaft, 1979a).

The deeper Barren Sandstone Unit is generally 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 Anderson Project has resulted in sufficient movement of water between these units so they will not be treated independently; this applies particularly to the north of the former MinEx property where faults cut the Lower Sandstone Conglomerate Unit. To the south of 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).

5.5 INFRASTRUCTURE

The Anderson 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 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 Anderson 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.

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 Anderson 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.

6 HISTORY

6.1 PRIOR OWNERSHIP AND EXPLORATION WORK

In January 1955, T.R. Anderson of Sacramento, California detected anomalous radioactivity in the vicinity of the Anderson 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% U_3O_8 and 33,230 pounds U_3O_8 (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 6.1: HISTORICAL PRODUCTION AT THE ANDERSON MINE (ARSENEAU, 2011)

Year	Tons	Grade (%U_3O_8)	Pounds (U_3O_8)
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 Anderson 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 Anderson Project. Following a field check and an evaluation of the 1968 Getty drill data, MinEx optioned the northern portion of the current Anderson Project (MinEx, 1978a).

In 1975, MinEx purchased the northern portion of the current Anderson 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 Anderson 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). Table 6.2 summarizes the phases of the historical exploration.

TABLE 6.2: EXPLORATION HISTORY AT THE ANDERSON PROPERTY (ARSENEAU, 2011)

Company	Period	Exploration Activities
Mining Group Led by Mr. T. R. Anderson	1955−1959	Aerial scintillometer surveying, ground prospecting, and outcrop mining
Getty Oil Company	1967−1968	Limited exploration drilling
Urangesellschaft USA, Inc.	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

Depressed uranium prices stalled exploration activities until 1995 when Hanson Exploration, Inc. (Hanson) of Phoenix, Arizona 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 Anderson 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 Anderson Project for the 2007 field season, but the drill program was never done. UEC has not conducted any drilling activity to date.

The data generated from this level of historic drilling activity resulted in numerous reports and required additional studies by outside consulting firms such as Chapman, Wood & Griswold; Morrison Knudson; Hazen Research, Inc.; and many others. These reports and studies are listed in the References section of this report.

6.2 HISTORICAL RESERVE ESTIMATES

Pre-feasibility and feasibility level studies were completed by MinEx (1978a-c) and Urangesellschaft (1979a-c); these studies included reported and published mineral reserve estimates. Note that the previous mineral reserves are quoted here for historical purposes only. There are no current mineral reserves as defined by NI 43-101 for the Anderson Project.

The historical mineral reserves are relevant because they are an indication of the size and grade of the uranium mineralization found on the property. The historical mineral reserves were estimated by major mining companies using the best methodologies available at the time, but they were prepared before the implementation of NI 43-101 and do not necessarily use the categories for mineral reserve and mineral resource reporting as defined in NI 43-101. The reader should not rely on the historical reserve estimates as they are superseded by the mineral resource estimate presented in *Section 14* of this report.

Mineable reserves, as reported by MinEx, were to be mined using open pit methods. Urangesellschaft reported reserves using two scenarios:

- expansion of MinEx's proposed open pit onto Urangesellschaft property
- stand-alone underground mine on Urangesellschaft property

Mineable reserves defined by MinEx (1978b) were estimated using radiometric and chemical data from 513 MinEx drill holes; a cut-off thickness of 0.6 m (2.0 ft) and a cut-off grade of 0.04% U_3O_8 were used to define the open pit limit, and an internal cut-off of 0.028% U_3O_8 was used to define reserves within the pit. A dry tonnage factor of 20.46 ft^3/ton (1.58 t/m^3) was used to calculate pounds of uranium. At that time, the reserve base was drilled using a 60 m spacing; MinEx considered this sufficient to classify the reserve as "Indicated." This use of the term "Indicated reserve" does not correspond to a "reserve" as defined by the CIM Definition Standards.

Urangesellschaft used the same cut-off criteria to define reserves for the open pit expansion (0.60 m thickness and 0.04% U_3O_8 grade). A smaller tonnage factor of 19.3 ft^3/ton (1.68 t/m^3) was used to reflect changing lithologies in the south compared to MinEx in the north. Underground mining reserves were defined by a cut-off thickness of 1.8 m (6.0 ft) and a cut-off grade of 0.05% U_3O_8. The underground reserve was based on the same dry density tonnage factor as the open pit. Urangesellschaft classified 26% of the reserve pounds U_3O_8 as "Indicated" based on 60 m (200 ft) drill hole centers and the remaining 74% as "Inferred" based on 122 m (400 ft) drill hole centers.

Note that the terms “Indicated reserve” and “Inferred reserve” are not categories used in NI 43-101.

Historical uranium reserves, as defined by MinEx and Urangesellschaft are summarized in Table 6.3. This reserve represents in-place mineable mineralization without an applied recovery factor. Urangesellschaft suggested an 85% mining recovery factor for underground mining (Hertzke, 1997).

TABLE 6.3: HISTORICAL RESERVE ESTIMATES FOR THE ANDERSON PROJECT (ARSENEAU, 2011)

Company	Mining method	Cut-off	tonnes million	Tons millions	Grade (%eU_3O_8)	Lbs millions
MinEx	Open Pit	0.04% and 60 cm (2 ft)	6.5	7.2	0.072	10.3
Urangesellschaft	Open Pit	0.04% and 60 cm (2 ft)	4.1	4.5	0.054	4.9
Urangesellschaft	Underground	0.05% and 1.8 m (6 ft)	0.5	0.6	0.093	1.0
TOTAL	Open pit+ underground	0.04 to 0.05%	11.1	12.2	0.073	17.9

The estimates in Table 6.3 predate NI 43-101, and BDRC has not done sufficient work to classify the historical estimate as a current resource. UEC is not treating the historical estimates as current.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

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. 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 Figures 7-1A (with the legend shown in Figure 7-1B).

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).

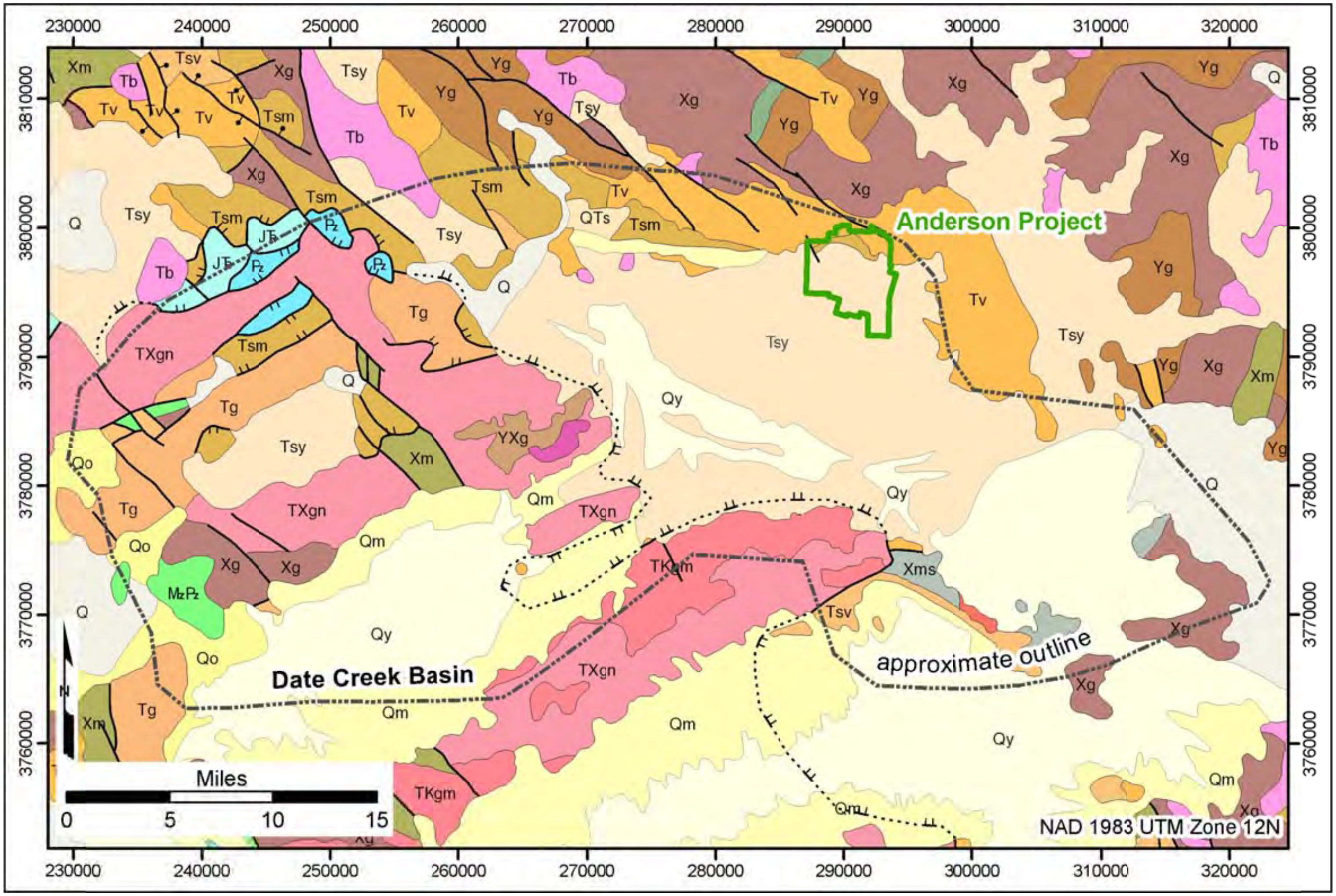


FIGURE 7-1A: GEOLOGIC MAP OF DATE CREEK BASIN

Note: Refer to Figure 7-1B for the legend

Distribution of Map Units

Geologic Map Units

- Q - Quaternary Surficial deposits, undivided (0-2 Ma)
- Qy - Holocene Surficial Deposits (0-10 ka)
- Qm - Late And Middle Pleistocene Surficial Deposits (10-750 ka)
- Qo - Early Pleistocene to Latest Pliocene Surficial Deposits (0.75-3 Ma)
- QTs - Early Pleistocene to Late Miocene Basin Deposits (0.75-10 Ma)
- Tsy - Pliocene to Middle Miocene Deposits (2-16 Ma)
- Tb - Late to Middle Miocene Basaltic Rocks (8-16 Ma)
- Tsv - Middle Miocene to Oligocene Volcanic And Sedimentary Rocks, Undivided (11-32 Ma)
- Tsm - Middle Miocene to Oligocene Sedimentary Rocks (11-32 Ma)
- Tv - Middle Miocene to Oligocene Volcanic Rocks (11-38 Ma)
- Tg - Middle Miocene to Oligocene Granitic Rocks (14-26 Ma)
- TXgn - Tertiary to Early Proterozoic Gneissic Rocks (15-1800 Ma)
- TKgm - Early Tertiary to Late Cretaceous Muscovite-Bearing Granitic Rocks (50-80 Ma)
- TKg - Early Tertiary to Late Cretaceous Granitic Rocks (50-82 Ma).
- Kv - Early Tertiary to Late Cretaceous Volcanic Rocks (50-82 Ma)
- KJo - Orocopia Schist (Cretaceous - Jurassic, 65-165 Ma)
- KJs - Cretaceous to Upper Jurassic Sedimentary Rocks with Minor Volcanic Rocks (80-160 Ma)
- Kmv - Sedimentary Rocks of the Upper Cretaceous Mesaverde Group (84-88 Ma)
- Ks - Cretaceous Sedimentary Rocks (about 88-97 Ma)
- Jm - Morrison Formation (Late Jurassic, about 145-160 Ma)
- Jsv - Jurassic Sedimentary and Volcanic Rocks (150-170 Ma)
- Jg - Jurassic Granitic Rocks (150-180 Ma)
- JTr - Jurassic And Triassic Sedimentary and Volcanic Rocks (160-240 Ma)
- MzPz - Jurassic to Cambrian Metamorphosed Sedimentary Rocks (160-540 Ma)
- Pz - Paleozoic Sedimentary Rocks (248-544 Ma)
- Yg - Middle Proterozoic Granitic Rocks (1400-1450 Ma)
- Xg - Early Proterozoic Granitic Rocks (1600-1800 Ma)
- Xms - Early Proterozoic Metasedimentary Rocks (1600-1800 Ma)
- Xmv - Early Proterozoic Metavolcanic Rocks (1650 to 1800 Ma)
- Xm - Early Proterozoic Metamorphic Rocks (1600-1800 Ma)

FIGURE 7-1B: LEGEND FOR FIGURE 7-1A

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

7.2 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 Anderson 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 Anderson 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 Anderson 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 Anderson 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).

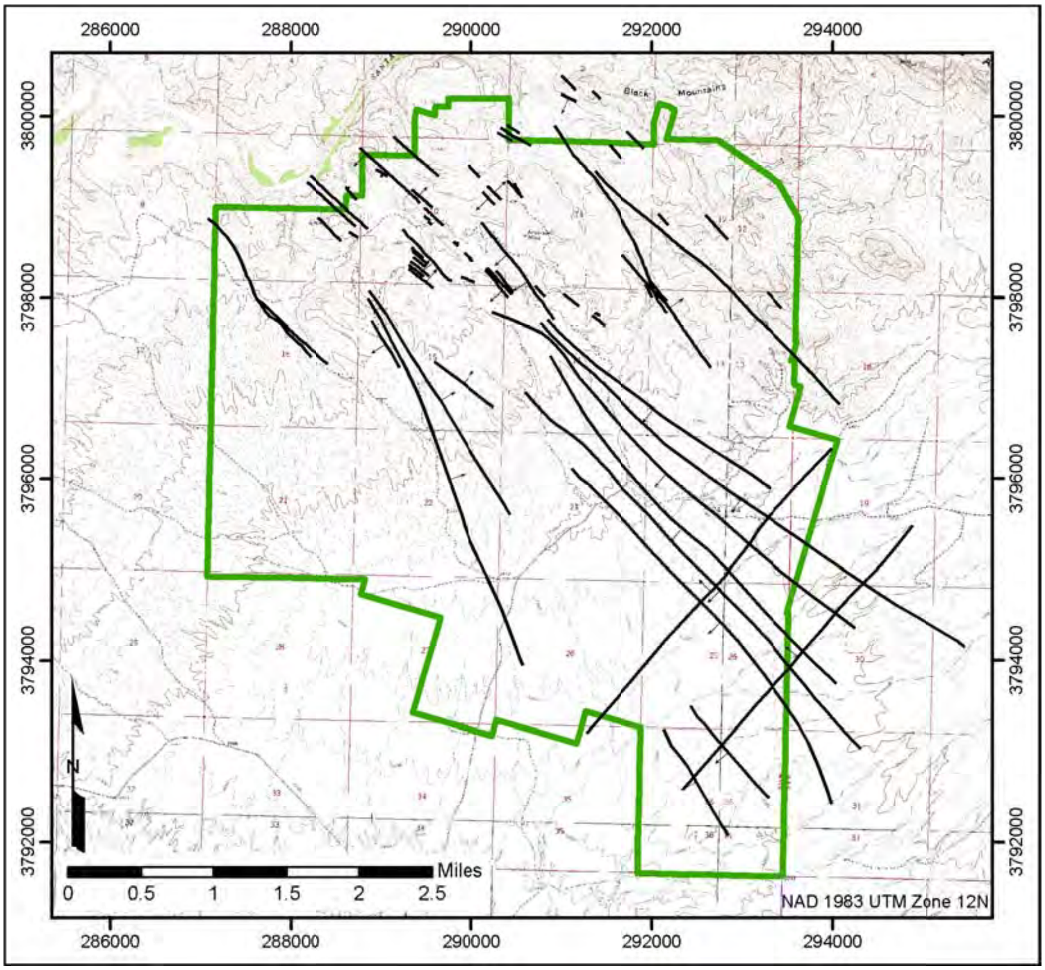


FIGURE 7-2: SURFACE TRACE OF FAULTING ON ANDERSON PROJECT
(ARROWS INDICATE DOWN DROPPED SIDE WHERE KNOWN)

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.

7.3 STRATIGRAPHY

Nine stratigraphic units were identified on the Anderson 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 Anderson Project is shown in Figure 7-3.

Uranium mineralization at the Anderson 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 Anderson Project. However, to the east of the Anderson 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, 1978b).

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, 1978b).

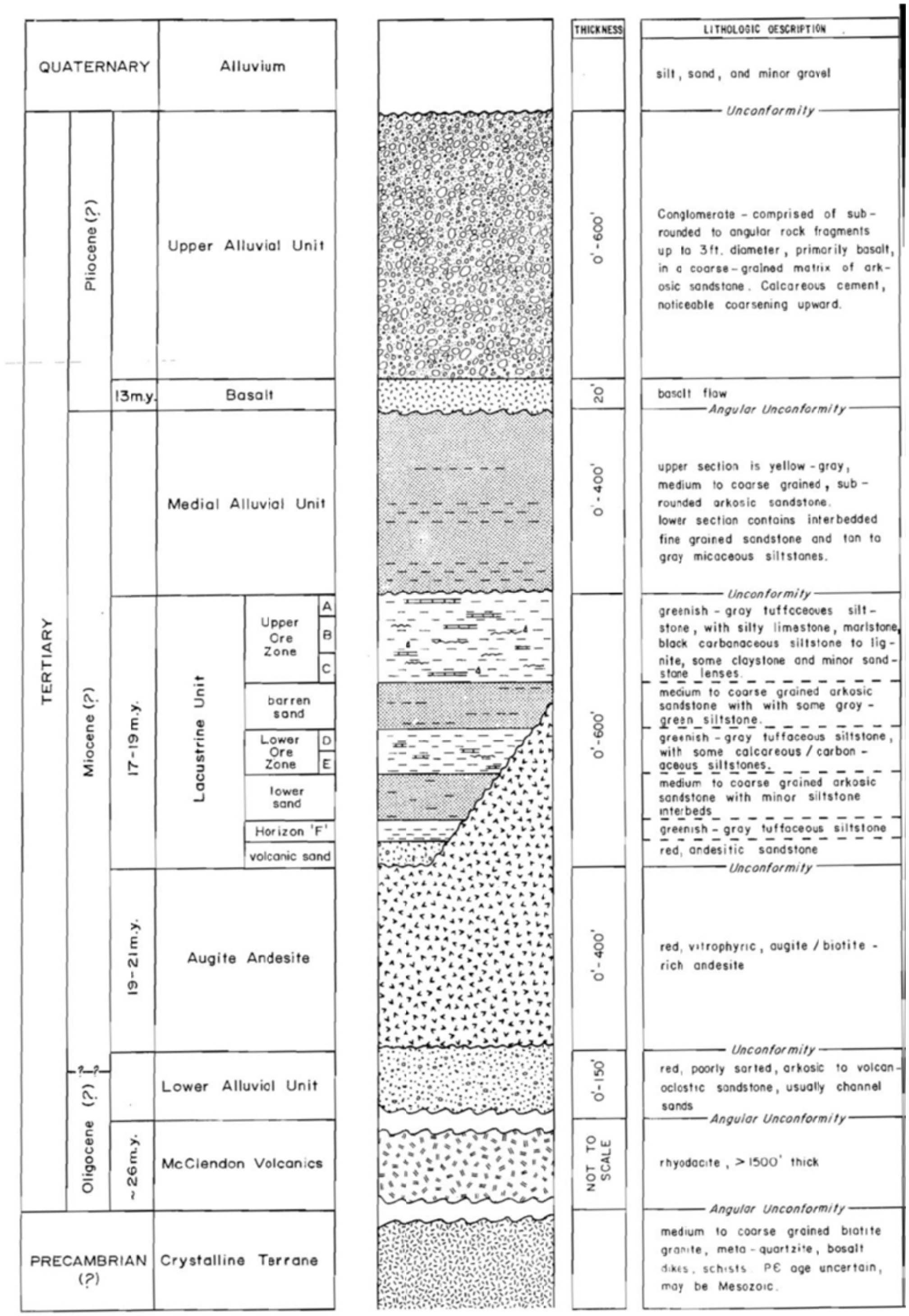


FIGURE 7-3: STRATIGRAPHIC COLUMN FROM THE ANDERSON PROJECT (SCHNEIDER, 1979)

7.4 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 ($K_2(UO_2)_2(Si_2O_5)_3{\cdot}4H_2O$), was 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 (Figure 7-4).

FIGURE 7-4: SECONDARY URANIUM MINERALIZATION IN FLOAT WESTERN PIT AREA

The uranium mineralization found at depth on the former Urangesellschaft property was reported by Hazen Research, Inc. (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% U_3O_8 to normal highs of 0.3 to 0.5% U_3O_8 with intercepts on occasion of 1.0% to 2.0% U_3O_8. Secondary enrichment of the syngenetic mineralization is observed along faults and at outcrops (Hertzke, 1997).

8 DEPOSIT TYPES

The uranium host rock sequence consists predominantly of a green to gray-green tuffaceous mudstone, which is interbedded with calcareous mudstone, carbonaceous mudstone, limestone, marl, lignite, chert and minor sand lenses. This sequence has been called the Anderson Mine Formation by Sherborne (1979) and ranges from 100 m to more than 500 m in thickness. This section has been tentatively correlated westward with the Chapin Wash Formation and most probably inter-tongues with the Chapin Wash Formation (Hertzke, 1997).

The Anderson 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 facies exhibit some uranium mineralization. The highest grades and most continuous mineralization are confined to the carbonaceous siltstones and lignitic materials. Occasional mineralization has also been noted 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).

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 relationship implies that mineralization occurred during the deposition of the carbonaceous material (MinEx, 1978b). Mineralization is also prevalent in calcareous facies.

Various origins were suggested for the uranium mineralization (Arseneau, 2011):

1. Devitrification of volcanic tuffs in and around the lacustrine environment that hosted the mineralized sediments
2. Solution, mobilization, and deposition from coarse-grained Precambrian biotite granites (with anomalous uranium values as high as 0.025%) that occur along the northern margins of the Date Creek Basin
3. A combination of 1 and 2
4. Hot springs that may have been present along tectonically active zones
5. Hypogene deposition

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).

9 EXPLORATION

A Light Detection and Ranging (LiDAR) survey was performed over the entire project area by Cooper Aerial Surveys Co. (Cooper Aerial) on 9 July 2011, between 13:07 UTC and 15:14 UTC (6:07 A.M. and 8:14 A.M., MST). Aerial imagery was collected at the same time. Data was processed using one of two base stations to obtain positional accuracies of between 3 and10 cm. Twenty-four ground control points showed a root mean square error of 0.219 ft (6.7 cm) between predicted and measured elevations. Cooper Aerial provided UEC with a one meter pixel digital elevation model (DEM) and a 2 ft contour shape-file derived from the LiDAR data. Cooper Aerial also corrected ortho imagery with a 0.15 m pixel size. Coordinates were converted from WGS84 to NAD 1983 UTM Zone 12N in meters, and elevation was reported in NAVD 1988 international feet. The conversion caused no distortion in elevations used in the resource model.

UEC has not performed any drilling to date on the Anderson Project.

10 DRILLING

10.1 MinEx and Urangesellschaft Drilling

Between 1974 and 1980, MinEx drilled a total of 970 rotary holes and 84 core holes on the northern portion of the Anderson Project, covering an area of approximately 425 ha. In the area that MinEx proposed as an open pit resource, MinEx's drill spacing was approximately 60 m (200 ft), the remainder of the MinEx property was drilled at nominal 120 m spacing. A total of 84 core holes, approximately 8%, were drilled as twins to rotary holes; of these, 72 had sufficient records to include in the resource database. Core holes were typically located 2 to 3 m from the twinned rotary hole.

Drill depths to mineralized intervals ranged from surface outcrop to about 250 m along the southern boundary of the historical Urangesellschaft property. Hole diameters varied, ranging from 10 to 16 cm. Drilling was stopped in 1980.

Between 1973 and 1982, Urangesellschaft drilled the southern portion of the Anderson Project. Rotary and core drilling on a 125 m (400 ft) spacing was completed over an area of approximately 610 ha. Between 1978 and 1979, an additional 70 ha was drilled on 60 m (200 ft) spacing near the northern boundary of the historical Urangesellschaft property in the vicinity of MinEx's proposed open pit area.

Drilling totalled more than 122,744 m (402,773 ft) in 319 rotary holes and more than 1,615 m (5,300 ft) in 33 core holes. Drill depths to mineralized intervals ranged from 120 m on the northern boundary of the historical Urangesellschaft property to over 600 m in the southern portion (Hertzke, 1997).

All Urangesellschaft and MinEx holes were drilled vertically. The average dip of the mineralized beds ranged from approximately 2º to more than 15º across the deposit. The true thickness of the beds is typically 96.5% to 99.9% of the apparent intercept length, based on a stratigraphic dip ranging from 2º to 15º. Occasionally, holes are believed to intercept mineralized high-angle faults at shallow angles and, therefore, report intercept lengths are not representative of true bed thickness. Fault intercepts are generally recognizable by non-corroborating intercepts in surrounding holes and the proximity of fault traces on the surface.

10.2 Concentric Drilling

In 2006, Concentric conducted the first drilling program on the Anderson Project since the field exploration programs of MinEx and Urangesellschaft were stopped in 1980 and 1982, respectively.

The 2006 drilling program was designed to confirm the authenticity of the historical MinEx exploration database by twinning a spatially distributed and statistically significant number of existing drill holes. A total of 24 vertical rotary holes and one rotary-core hole were drilled between 23 June and 26 September 2006, totaling 2,465 m (8,087 ft) of drilling. All rotary holes were drilled 14 cm in diameter. Drill depths varied from 30 to 200 m depending on location along dip. The core hole was rotary-drilled to the top of a target radiometric zone and then cored between 23 and 29 m of depth. The core was split and shipped for assaying. The location of the 2006 confirmation holes is shown in Figure 10-1. No confirmation holes were drilled on the former Urangesellschaft portion of the Anderson Project. Drilling was completed by Layne Christensen Company, Exploration and Environmental Division, of Chandler, Arizona under the supervision of Exploration Geologist, Ed Huskinson, Jr. of Kingman, Arizona. Lithologic logging of the drill cuttings was performed on site by Mr. Huskinson.

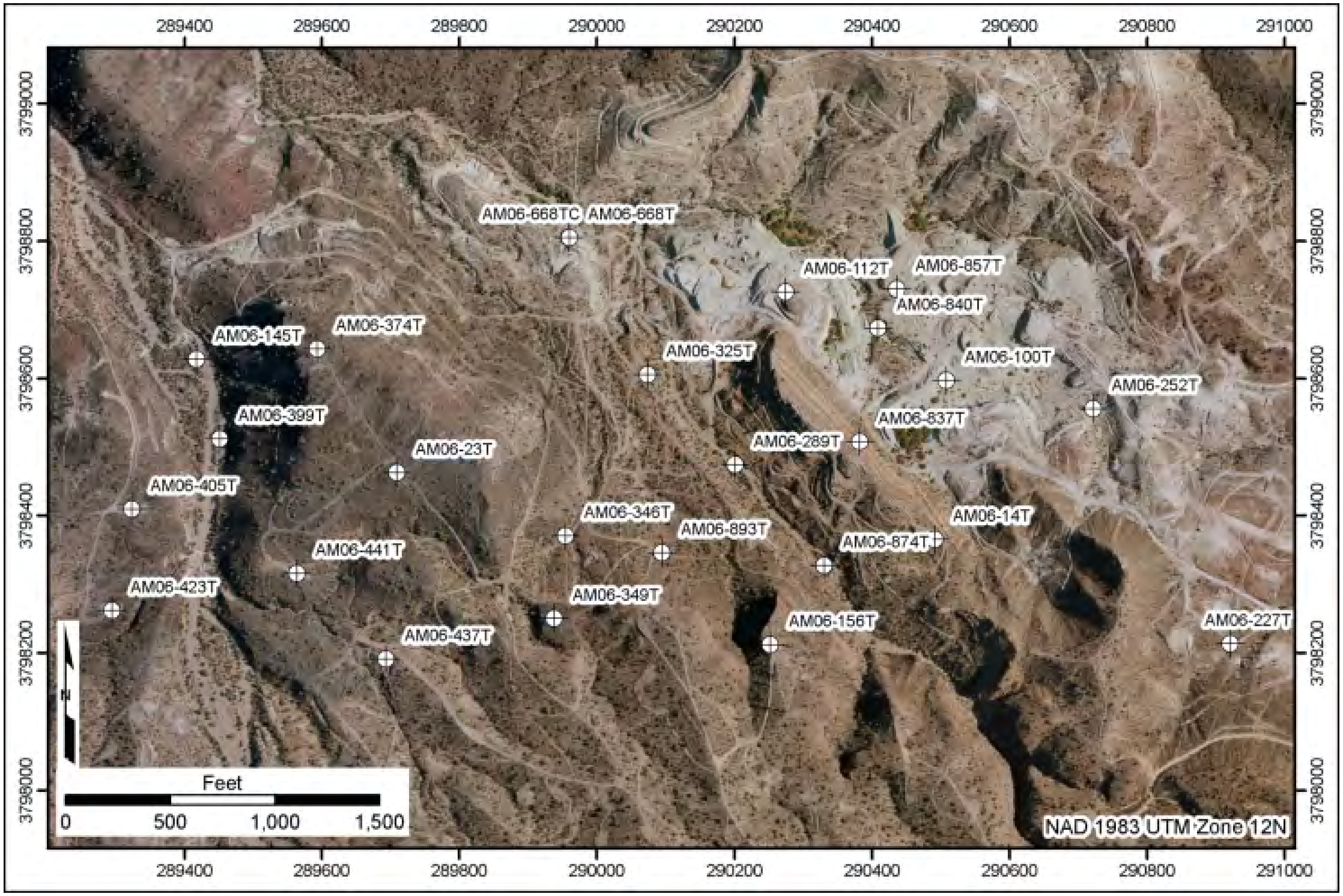


FIGURE 10-1: CONCENTRIC'S 2006 BOREHOLE LOCATIONS

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 CORE HOLES

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. Various sized core bits and core barrels were tried by MinEx contractors; the best recovery was obtained using a 7.6 cm diameter core barrel (NQ core) (Roberts, 1978).

The core from each run was measured, labelled and boxed for shipment. When the core arrived at MinEx office, it was split length wise, and half of each core was dried and pulverized. Pulverized core, representing 15 to 30 cm intervals, was analyzed on a Blake Beta-Gamma scaler (Roberts, 1978). Each interval was analyzed three times and an average was taken. Subsequently, core was sent to a laboratory for closed-can analyses.

Based on the work performed by MinEx, there are elevated levels of vanadium associated with zones of uranium mineralization. Manganese also showed a direct relationship with higher grade uranium mineralization; lithium and fluorine showed an indirect relationship. All rotary and core holes were probed with downhole gamma ray probes. Concentric acquired most of the original paper logs for the MinEx and Urangesellschaft drilling.

11.2 ROTARY HOLES

All of the Urangesellschaft holes were drilled and logged by Century Geophysical Corporation (Century Geophysical) based in Grants, New Mexico and Tulsa, Oklahoma. MinEx conducted electric logging using Century Geophysical, based in Denver, Colorado, and other unidentified contractors. Based on log results, equivalent logging methods were employed by all parties.

In order to use the historical gamma ray logs, Agapito developed a procedure to digitize the existing hard copy geophysical logs: converting the digitized logs to numerical values and processing the values to calculate radiometric-equivalent uranium grades, designated as % eU_3O_8. The Agapito procedure is based on an historical FORTRAN computer code, GAMLOG, designed by the United States Atomic Energy Commission (Scott, 1962).

11.3 LOGGING PRACTICES

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 U_3O_8 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 (% eU_3O_8). The scintillation probes can delineate anomalous uranium mineralization to approximately 7 to 15 parts per million (ppm) eU_3O_8.

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 introduced computerized logging (Compu-Log) to the Urangesellschaft exploration program; this 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 analysis printout (% eU_3O_8)
- a magnetic cartridge tape containing all data contained in the electric log

Downhole measurements were reported for 0.5 ft intervals.

11.4 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).

11.5 GAMMA LOG DIGITIZING

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. For the purpose of converting the paper logs to digital values, 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. Files ranged from 100 KB to 4 MB depending on the scanning resolution and length of the log.

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 the 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; this 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 (QA).

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. Table 11.1 shows the comparison of grades which confirms close agreement between the two calculation methods.

TABLE 11.1: COMPARISON OF GAMLOG AND EXCEL GRADE CALCULATIONS FROM CPS DATA (GILBRIDE ET AL, 2010)

Depth (ft)	Excel®	Original GAMLOG	Difference (GAMLOG vs. Excel®)
Hole 1			
0	0.002	0	0
0.5	1.069	1.07	0
1	0.375	0.37	-0.01
1.5	0.011	0.01	0
2	0.56	0.56	0
2.5	0.764	0.76	0
3	0.544	0.54	0
3.5	0.098	0.1	0
4	0.206	0.21	0
Hole 2			
0	0.226	0.23	0
1	0.454	0.45	0
2	0.487	0.49	0
3	0.528	0.51	-0.02
4	0.121	0.13	0.01
Hole 3			
10	0.121	0.12	0
10.5	0.144	0.14	0
11	0.115	0.13	0.02
11.5	0.33	0.32	-0.01
12	0.157	0.17	0.01
12.5	0.092	0.07	-0.02
13	0	0.03	0.03
13.5	0.008	0.04	0.03
14	0	0.02	0.02
14.5	0.001	0.02	0.02
15	0.016	0	-0.02
15.5	0	0	0
16	0.217	0.2	-0.02
16.5	0.057	0.07	0.01
17	0.007	0	-0.01

Depth (ft)	Excel®	Original GAMLOG	Difference (GAMLOG vs. Excel®)
Hole 4			
4.7	0.057	0.05	-0.01
5.2	0.068	0.07	0
5.7	0.26	0.26	0
6.2	0.322	0.32	0
6.7	0.256	0.26	0
7.2	0.336	0.34	0
7.7	0.478	0.47	-0.01
8.2	0.159	0.17	0.01
8.7	0.273	0.27	0
9.2	0.094	0.1	0.01
Hole 5			
0	0.02	0.02	0
0.5	0.015	0.02	0.01
1	0.052	0.04	-0.01
1.5	0.007	0.02	0.01
2	0.009	0.01	0
17.2	0.002	0.02	0.02
17.7	0.01	0.04	0.03
18.2	0.002	0.04	0.04
18.7	0.003	0.02	0.02

The basic operation of the Excel® VBA (visual basic editor) macro is as follows:

- 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.

To further confirm that the conversion of the cps to eU_3O_8 was done correctly, Arseneau (2011) compared the converted eU_3O_8 data with the chemical assays for the 109 holes containing chemical assay data. The correlation between the assayed U_3O_8 and the eU_3O_8 values showed a wide scatter and the correlation is not very clear (Figure 11-1).

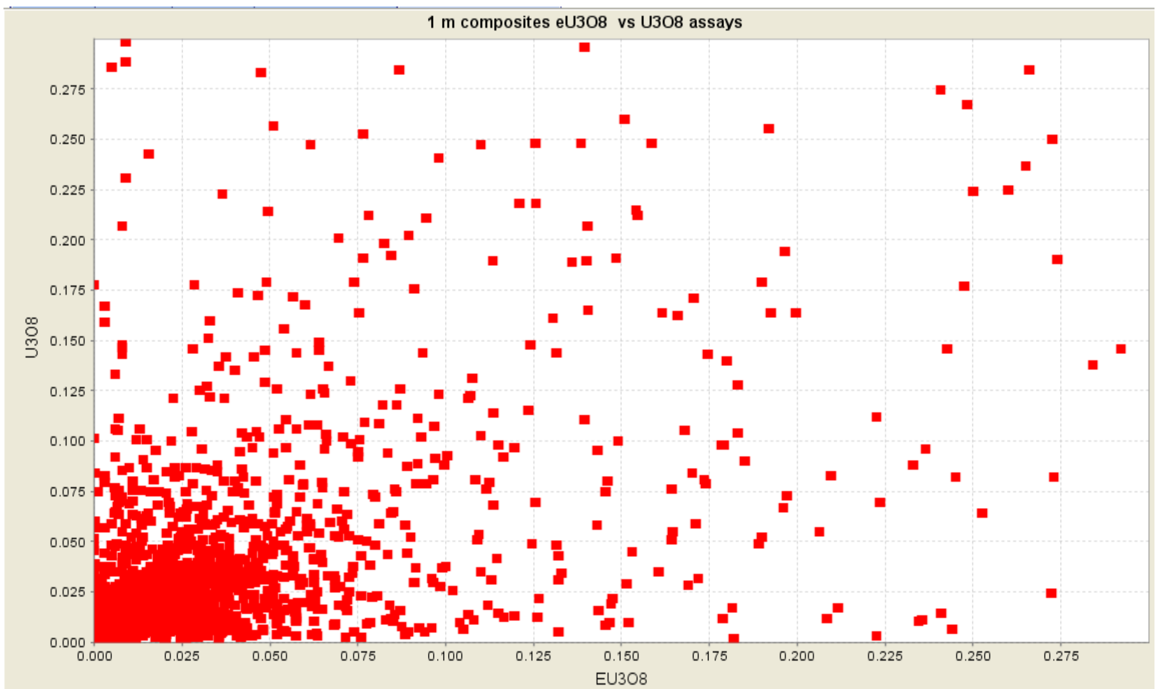


FIGURE 11-1: SCATTER PLOT OF EU_3O_8 AND U_3O_8 FOR CORE HOLES (ARSENEAU, 2011)

A comparison of the Q-Q plot of the same data, however, demonstrates that for quartiles greater than 0.07%, the eU_3O_8 value is slightly less than the corresponding U_3O_8 assay value, which indicates that the eU_3O_8 values may be slightly underestimated or conservatively estimated (Figure 11-2).

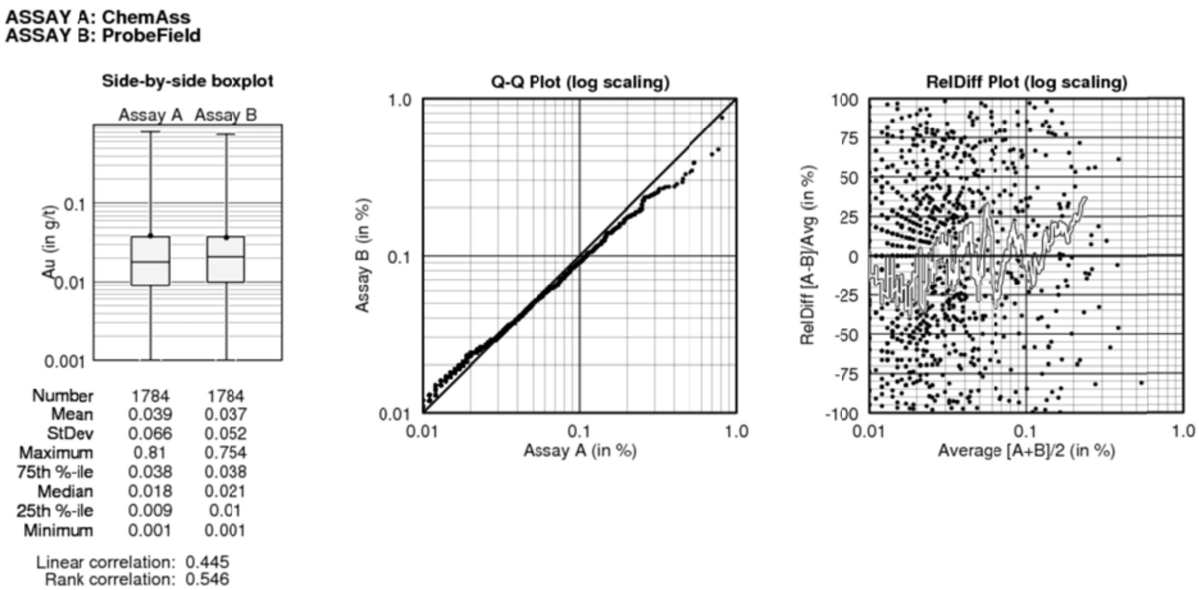


FIGURE 11-2: COMPARISON OF Q-Q PLOT AND RELATIVE DIFFERENCES BETWEEN eU_3O_8 AND U_3O_8 ASSAY VALUES (ARSENEAU, 2011)

11.6 CONCENTRIC DRILLING

All drill cuttings were recovered and stored in 22 liter (5 gallon) plastic containers for future metallurgical testing. Natural gamma logging was conducted in all holes by Mr. Raymond H. Federwisch, Geophysicist, of Geophysical Logging Services, Prescott, Arizona using a Mount Sopris Instrument Co., Inc. 2PGA-1000 Poly-Gamma probe. Resistivity and density were logged in select holes. Because the radiometric beds at the Anderson property can be fairly thin, the mineralized horizons were sampled at 75 cm (2.5 ft) intervals. Non-mineralized zones were sampled at 1.5 m (5 ft) intervals. Because the holes were twins of previously-drilled holes, the historical radiometric logs were used to identify the start points for 75 cm sampling intervals.

11.6.1 COLLECTION OF DRY SAMPLES

Dry samples were collected in a 22 liter bucket suspended beneath the cyclone dust and cuttings collector. When drilling in non-mineralized rock, the sample was split with a Gilson-type splitter to collect an archival sample; a small subsample was also retained by the on-site geologist for logging purposes. In mineralized zones, an assay sample was captured in a clean bucket and poured into the hopper at the top of a Gilson-type splitter. The dividers below the hopper were set to provide about 1 to 2 kg of dry samples for assays. These dry samples were collected in large pans and then poured into orange buckets that were emptied into plastic bags; the sample numbers were recorded on the bag. Tear out sample tags with the sample numbers were placed in the bags and the bags were immediately sealed with blue zip-ties. After each sample was collected, the Gilson splitter and the orange sample buckets were cleaned with a high-pressure air hose and/or steel brush.

The remaining, much larger archival split was collected in a large pan that was emptied into a new, clean, pre-numbered white bucket with the sample number written on the side. Before the archival bucket was sealed, a logging subsample was collected from the archival bucket, lightly washed through a 12- to16-mesh standard sieve, and placed into a sample container that held one dozen samples representing 20 m of drill hole. Oversized material from the wet sieving operation was placed back into the archival bucket. The logging subsample was delivered to the on-site geologist who logged the borehole in real time, but never more than 20 or 30 m from the advancing bit. Often, if the drill penetration rate was low, samples would be logged at 6 m intervals (at the end of each drill steel). Geologic control and understanding of the units was strictly maintained in the field.

After the logging subsample had been collected, the archival sample bucket was sealed with a new lid with the sample number written on it; this was checked against the number on the bucket. The on-site geologist recorded the sample number on the field logging form. Each night, the samples from the mineralized zones were transported to and locked in the Concentric Energy's office/warehouse. Usually the entire sample run for the day was transported to the warehouse, but occasionally only the mineralized zones would be carried back. On two occasions, samples from mineralized zones were left at the drill site overnight in a locked, steel container.

11.6.2 Collection of Wet Samples

Below the water table, the following modified sampling protocol was used: after passing through the cyclone receiver, the wet cutting was sent through an Anaconda wet splitter (a hydraulic wet splitter that turns under the cyclone receiver) where it was divided into 16 pie-shaped segments that could be covered. If all the wedges were left uncovered, then 100% of the sample was recovered at the bottom of the splitter. If eight of the wedges were covered up, then the splitter yields an 8/16, or 50/50 split. The material that passed through the uncovered open wedge fell through the splitter and was collected in a bucket suspended beneath the outlet pipe. The other half passed outside the sampling area and was directed to a pipe on the side of the wet splitter. That side could be sampled as well by suspending a bucket under the discharge pipe.

In non-mineralized zones, the splitter was adjusted to yield a quantity sufficient to fill a standard polypropylene sample bag. After the sample was collected, the sample bag was lifted out of the pipe, the sample tag is inserted, and the bag is sealed and set aside for temporary storage.

In mineralized zones, the number of covered wedges was regulated to yield a split that could be collected in a bucket suspended underneath the splitter. At the end of the 75 cm sample interval, the bucket was removed and a waterproof sample tag was placed in the bucket before the bucket was sealed with a new lid; the sample number was recorded on the lid.

The logging subsample was collected using a standard 12- to 16-mesh testing sieve placed under the Anaconda splitter's side discharge. A representative sample from the sieve was placed into a 12-compartment tray for future logging.

Using both wet and dry samples, the on-site geologist prepared a 20-compartment, plastic archival chip tray using the chip samples that were used to log the geology and mineralogy of the borehole. Each compartment was labelled with the footage interval and the unique sample number. These chip trays provided quick and easy access to the borehole geology for subsequent review or examination, and acted as a field check on sample continuity and accuracy at the drill site.

The mineralized samples were always under the control of the site geologist, either locked in Concentric Energy's warehouse, or, as they had been in two instances, locked at the drill site overnight before being transported to the warehouse the following day.

11.7 HISTORICAL CHEMICAL ASSAYING

MinEx and Urangesellschaft both conducted chemical assays on core 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 report because they are not available. Arseneau (2011) 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 imported: 1,610 from MinEx drill holes and 1,967 from Urangesellschaft drill holes.

11.7.1 ASSAY PRACTICES

Two types of quantitative chemical assays were performed: fluorometric and colorimetric (wet) assays. Both methods reported percent uranium by weight.

The vast majority of chemical assays were fluorometric, a rapid and inexpensive alternative to conventional wet chemical analyses. Fluorometry is very accurate and is still used by uranium laboratories such as Hazen Research. It is quantitative for uranium and free from interference because uranium is one of the few metallic elements that exhibits "intrinsic fluorescence". With X-ray fluorescence (XRF), in contrast, the sample is irradiated by X-rays that cause elements within the sample to fluoresce (emit) X-rays in turn. Characteristic spectra are emitted for each element. The numbers of emitted X-rays per characteristic wave length are proportional to the amount of the element present. A spectrometer is used to measure the count rate of emitted radiation at various wave lengths.

Chemical assays were cross-checked in some cases with closed-can (sealed) and open-can gamma assays where the same piece of core was used in both tests. Closed-can or open-can assays measure gamma radiation in the laboratory, from which equivalent uranium grade is computed as a function of the amount of radiation and mass of the specimen. Hazen Research, Inc. (Hazen Research) performed a limited number of semi-quantitative X-ray fluorescence analyses of minor elements for MinEx.

Almost all chemical assays were performed on 30 cm (1.0 ft) core lengths. MinEx performed chemical assays on 15 cm (0.5 ft) core lengths in limited capacity on early holes.

The MinEx chemical assays were performed by Hazen Research of Golden, Colorado. Hazen Research 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. (Skyline Labs) in Tucson, Arizona.

Skyline Labs hold a State of Arizona Board of Technical Registration certification and ISO 17025 certification, but Arseneau (2011) was not aware of this certification in 1979 when most of the samples were processed.

All Urangesellschaft samples were processed at Hazen Research or Skyline Labs.

Both Hazen Research and Skyline Labs are long-established laboratories that have provided assay services to the mining industry for many years.

11.7.2 ASSAY QUALITY CONTROL

For the purposes of quality control, MinEx conducted cross-checks of assays between the three laboratories used during the exploration program. Originally, average grade equilibrium factors calculated from chemical and closed-can results were compared to check the consistency of chemical and radiometric data between laboratories. Equilibrium factors were compared based on 429 colorimetric and closed-can radiometric assays from Chemical & Geochemical Laboratories and 1,619 fluorometric assays and 838 closed-can assays from Hazen Research (Lucht, 1978). MinEx determined that equilibrium factors varied by location and did not provide a meaningful basis to compare consistency between laboratories.

MinEx achieved tight consistency between laboratories by establishing a set of 30 standardized pulp samples, referred to as the “Anderson Standards.” The standard samples ranged in grade from 0.009% to 0.450% chemical U_3O_8 (Lucht, 1978). Chemical U_3O_8 **(**cU_3O_8) is the grade determined by chemical assay. The samples were first assayed by the fluorometric method at either Chemical & Geochemical Laboratories or Hazen Research, and then re-assayed by Skyline Labs. The re-assayed results were found to be within ± 0.002% cU_3O_8 of the original assays. Assays over 0.1% cU_3O_8 were confirmed using long-term, wet-chemical assays (volumetric analyses). MinEx concluded that the three laboratories were producing uranium assays with sufficient consistency to validate the results of their sampling program.

Urangesellschaft's sampling program was conducted at the same time as MinEx's program and produced the same type of fluorometric chemical assays from Chemical & Geochemical Laboratories and Hazen Research.

11.8 CONCENTRIC CHEMICAL ASSAYING

Anderson Mining Company, a wholly-owned subsidiary of Concentric Energy, conducted a due diligence drilling program on the northern portion of the Anderson Project. The program involved drilling holes adjacent to selected historical exploration holes to confirm stratigraphy and grade.

These twinned holes were selected by an independent consultant, Don Earnest of Resource Evaluation, Inc., Tucson, Arizona.

11.8.1 SAMPLE PREPARATION

The Concentric samples were collected at the drill rig at 0.75 m intervals (2.5 ft) and shipped to Jacobs Laboratory in Tucson for drying and preparation. At Jacobs Laboratory, each sample bag was opened and the contents were carefully poured into large drying pans. The enclosed sample field tag accompanied the sample throughout the lab. The drying pans were placed in an oven, and dried for a minimum of 24 hours. After cooling, the contents of the pans were put into larger pans, then passed through a jaw roll crusher and reduced to nominal 10 mesh. The sample was then homogenized by pouring it from pan to pan, back and forth, several times.

After homogenization, the sample was passed through a Jones-type riffle splitter to produce a 300 gram (g) subsample. The subsample was pulverized to 80% passing -100 mesh with a disc pulverizer. The splitter reject was placed back in the bucket, sealed and stored in the warehouse. The pulp sample was homogenized and a 10 g subsample was collected and shipped to ACTLABS in Ancaster, Ontario, Canada for DNC (Delayed Neutron Count) analysis for U_3O_8 determination. Pulps for ICP were shipped to American Assay Lab (AAL) in Sparks, Nevada for multi-element (package ICP-2A) analysis.

11.8.2 SAMPLE SECURITY

After collection at the drill site, the samples were logged on a *sample submittal sheet* that accompanied the samples when they were either picked up by Jacobs Lab personnel or were transported to the Jacobs Lab facility in Tucson, Arizona, by Concentric Energy personnel. On arrival at Jacobs Lab's facility, the samples were placed in numerical order and logged into the computer system.

After the samples were prepared for analysis, the individual sample containers were placed in a container and shipped to a laboratory together with blanks, duplicates and standards, with a copy of the *sample log sheet*. The containers and the pulp samples were shipped by UPS to ACTLABS for DNC analysis or AAL for ICP analysis. A copy of the UPS tracking number was kept at Jacobs Lab with a copy of the sample submittal sheet in the respective shipment. When the shipment was received at the lab, a copy of the sample submittal sheet was sent back to Concentric to indicate that they had been received. At each transfer, the transferor and the recipient both signed the sample log sheet. When the lab results were submitted to Concentric, a copy of the sample submittal sheet was attached to complete the chain of custody.

11.8.3 ASSAYING PROCEDURES

Samples shipped to ACTLABS were analyzed by neutron-activated DNC. The technique consists of dual cyclic neutron activation in a differentiated neutron flux spectrum at bare and cadmium-covered irradiation positions and subsequent delayed-fission neutron counting. Uranium concentrations are determined from calibration curves determined for samples of known activity.

Samples shipped to AAL were analyzed by ICP-atomic emission spectrometry (ICP-AES). Detection limits, sensitivity, and the optimum and linear concentration ranges of the elements can vary with the wavelength, spectrometer, matrix and operating conditions. Prior to analysis, samples must be solubilized or digested using appropriate sample preparation methods. The instrument measures characteristic emission spectra by optical spectrometry. Samples are nebulized and the resulting aerosol is transported to the plasma torch. Element-specific emission spectra are produced by a radio-frequency ICP. The spectra are dispersed by a grating spectrometer, and the intensities of the emission lines are monitored by photosensitive devices. Background correction is required for trace element determination.

11.8.4 LABORATORY CERTIFICATIONS

Jacobs Labs, of Tucson, Arizona, (520 622 0813) is a recognized and registered State of Arizona Board of Technical Registration lab on the list of recommended assay labs issued by the Arizona Department of Mines and Mineral Resources. ACTLABS in Ancaster, Ontario, Canada L9G 4V5 is used as a check lab by many of the uranium companies in Canada. They are accredited for assaying by the Standard Council of Canada, ISO 17025 (266).

AAL is a registered and approved assay lab located in Sparks, Nevada with the following accreditations:

- Certificate of International Standards Organization (ISO)\IEC 17025
- Certificate of Laboratory Proficiency PTP-MAL, accredited by Standards Council of Canada
- Geostats of Australia certificate
- Society of Mineral Analysts, Round Robin testing

11.8.5 QUALITY ASSURANCE/QUALITY CONTROL

Standards were obtained from CANMET Mining and Mineral Sciences Laboratories in Ottawa, Ontario, Canada for use in calibration of the analysis equipment. Standard #BL-1, which grades 0.022% (± 0.001%) U was used to verify the DNC results. Because the results of DNC analysis of the calibration standard always fell within the acceptable range, no changes or adjustments to the sample preparation were deemed necessary.

BDRC believes that all sample preparation, security and analytical procedures are adequate for the purposes of this Technical Report.

11.9 CURRENT METALLURGICAL SAMPLES

On December 18, 2013 the author's Beahm, and McNulty were on site along with UEC personnel. Surface exposures of mineralization in the northeastern portion of the deposit and mineralized stockpiles from historic surface mining were examined. Uranium and vanadium mineralization was present based on visual observations and measurement of radiological levels. During this time 4 samples of approximately 50 pounds 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. The samples were transferred via chain of custody protocol.

FIGURE 11-3: SITE PHOTO SHOWING URANIUM MINERALIZATION IN STOCKPILE

12 DATA VERIFICATION

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
- 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.

Fourteen 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. The author believes that UEC has done a very good job in verifying and validating the physical locations of all drill holes in the database.

In March 2012, additional historical data was obtained by UEC from the archives at the Arizona Geological Survey (AZGS). This included locations, lithology and downhole data for seven drill holes completed by Minex in 1980 that were not included in the database. One of the new locations was confirmed during fieldwork as an unknown drill hole.

UEC also obtained locations, lithology, and downhole data for six Urangesellschaft drill holes completed in 1980-1982 that were not in the original database. One of these locations was also previously confirmed by fieldwork as an unknown drill hole.

During his site visit in May 2012, the qualified person observed numerous drill hole collars in the field and verified that these correlate with the digital database and are representative of the extent of drilling coverage over the deposit area. The archived rock chips from drilling on the property were inspected at UEC's storage facilities in Wickenburg, Arizona. Hardcopy (paper) gamma logs from storage were also reviewed and correlated with results in the digital database. Orange to greenish-yellow uranium-bearing minerals were observed in outcrop and in rocks located in the dumps remaining from historic production on the property. Observations and inspection during the site visit convinced the qualified person that data collected to characterize uranium mineralization on the property is adequate for resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 HISTORICAL TESTING

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, Inc. of Golden, Colorado. 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 optimum process. MinEx and Urangesellschaft outlined additional research to address outstanding technical concerns, including further research into the feasibility of alkaline leaching.

13.1.1 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 CO_3 content varies from 0.03% to 32% and U_3O_8 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 C_{org} (lignite) and CO_3 (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 Na_2CO_3 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 ft^2/tpd), but improved with roasting (3 to 9 ft^2/tpd) (Hertzke, 1997).

Metallurgical analyses by Hazen Research established near-optimum, acid-leach conditions for the northern portion of the Anderson 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 ($CaCO_3$), corresponding to a minimum acid consumption of 180 kg (400 lbs) of sulfuric acid (H_2SO_4) per ton of ore. Leachability was further complicated by carbonate contents which varied from 0% to over 50% $CaCO_3$ (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).

13.1.2 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 ft^2/tpd.

Studies determined that the feasibility of alkaline leaching was limited by the high organic carbon content, primarily in the form of lignite (C_{org}), 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% C_{org} on average. Actual values varied widely, ranging from 0% to over 15% C_{org} (Urangesellschaft, 1980).

The program concluded that additional research was required to improve the understanding of the relationship between U_3O_8 grade and associated CO_3 and C_{org}, 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).

13.1.3 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 Anderson 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.

13.2 CURRENT METALLURGICAL TESTING

The current metallurgical scoping test results are provided in the report "Scoping Metallurgical Testing of Anderson Mine Samples" dated May 30, 2014, Resource Development Inc. (RDI), Appendix A.

On December 18, 2013 the author's Beahm and McNulty were on site along with UEC personnel. During this time 4 samples of approximately 50 pounds 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 (LOI) 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 pursued 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 be needed as well 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 test 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) and this 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.

The PEA includes provisions in pre-production capital for metallurgical testing including a pilot heap leach.

14 MINERAL RESOURCE ESTIMATE

14.1 INTRODUCTION

The mineral resource estimate was prepared by Robert Sim, P.Geo and Bruce Davis, FAusIMM. Both are independent Qualified Persons within the meaning of NI 43-101 for the purposes of mineral resource estimates contained in this report. Estimations are made from 3-dimensional block models based on geostatistical applications using commercial mine planning software (MineSight® v7.0-3). The project limits are based in the UTM coordinate system using a nominal block size of 10 x 10 x 2 m (L x W x H). All drill holes are vertically oriented with variably spaced holes throughout the deposit: 30 m spacing in the northern area, 60 m spacing in the central area and 120-150 m spacing or more in the southern area.

The resource estimate was generated using drill hole sample results and the interpretation of a geologic model that relates to the spatial distribution of eU_3O_8. Interpolation characteristics were defined based on the geology, drill hole spacing and geostatistical analysis of the data. The resources were classified according to their proximity to the sample locations and are reported, as required by NI 43-101, according to the CIM standards on Mineral Resources and Reserves (CIM, 2005).

14.2 GEOLOGIC MODEL, DOMAINS AND CODING

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. A series of eight individual, uranium-bearing zones were been interpreted from the drill hole sample data. These domains generally define zones that exceed 0.005% eU_3O_8. This threshold ensures that the domains capture all zones of mineralized material, but also allow for the inclusion of some internal lower-grade dilution in the model. It is common that some of the thicker parts of individual zones are comprised of central low-grade samples flanked by high-grade along the upper and lower contacts. Other areas show a series of three or more high grade bands intercalated with low-grade patches. Overall, the eight individual zone domains are separated by intervals with little to no eU_3O_8 mineralization. These barren intervals between Zone domains are variable and range from a few meters to over 50 m in some areas.

The interpretation of the Zone domains honors two NW-SE striking, sub-vertical faults running through the northeastern part of the deposit area. There is relatively little displacement of the main mineralized zones as a result of these faults (other than some apparent local curving of the typically planar bands of mineralization). As a result, the geologic model has been interpreted to honor the distribution of grades present in the sampling and in some cases (Zone 2 and Zone 3), the interpretation shows only minor undulation in the vicinity of the fault zones.

Table 14.1 lists the eight Zone domains

TABLE 14.1: DOMAINS AND CODING

Zone Domain	Zone Code #	Comments
Zone 1	1	Upper east zone.
Zone 2	2	Largest zone measures ~2.5km EW x 5km NS. Contains majority of resource.
Zone 3	3	Immediately below Zone 2. Combined a series of previous zones.
Zone 4	4	Irregular zone on upper west side.
Zone 5	5	Thin upper central zone.
Zone 6	6	Small zone in area of fault zones.
Zone 7	7	Thin upper east zone.
Zone 8	8	Irregular and generally narrow zone at depth.

The distribution of the various Zone domains is shown in Figures 14-1 to 14-4.

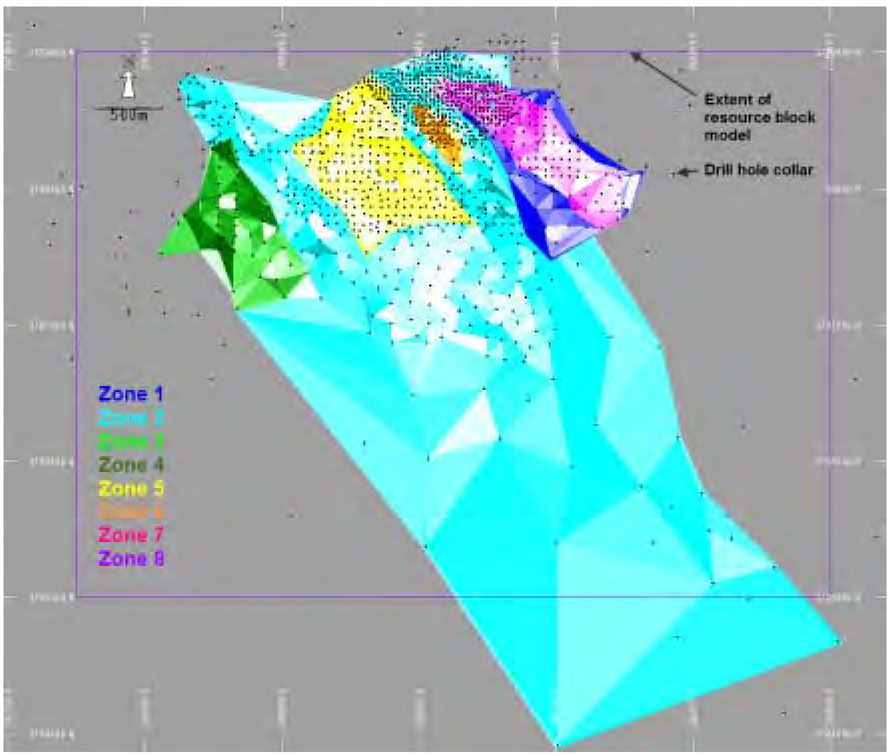


FIGURE 14-1: PLAN SHOWING ZONE DOMAINS, DH COLLARS AND EXTENTS OF THE RESOURCE MODEL

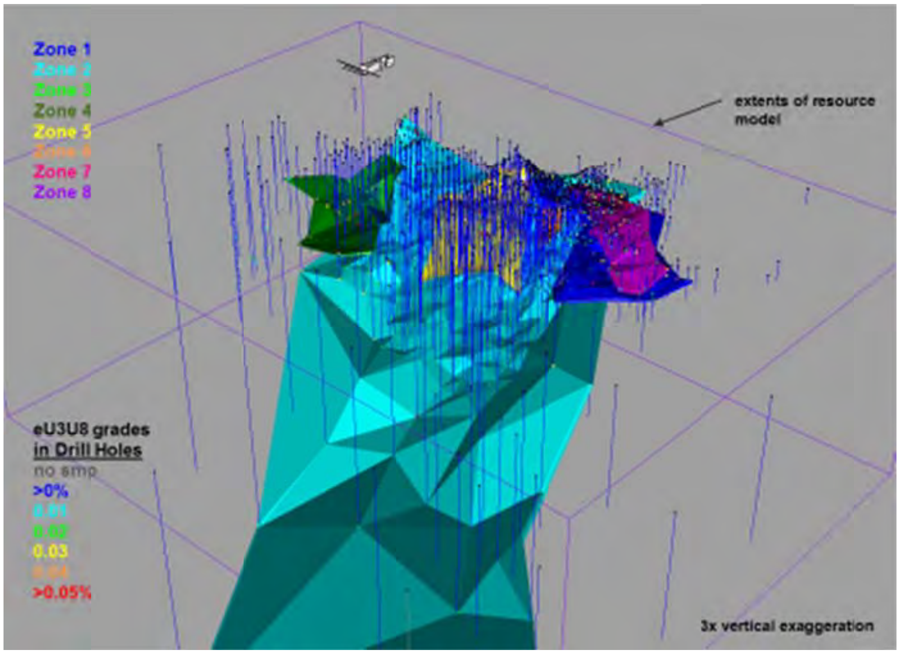


FIGURE 14-2: ISOMETRIC VIEW OF ZONE DOMAINS AND eU_3O_8 GRADES IN DRILL HOLES (VIEW NW)

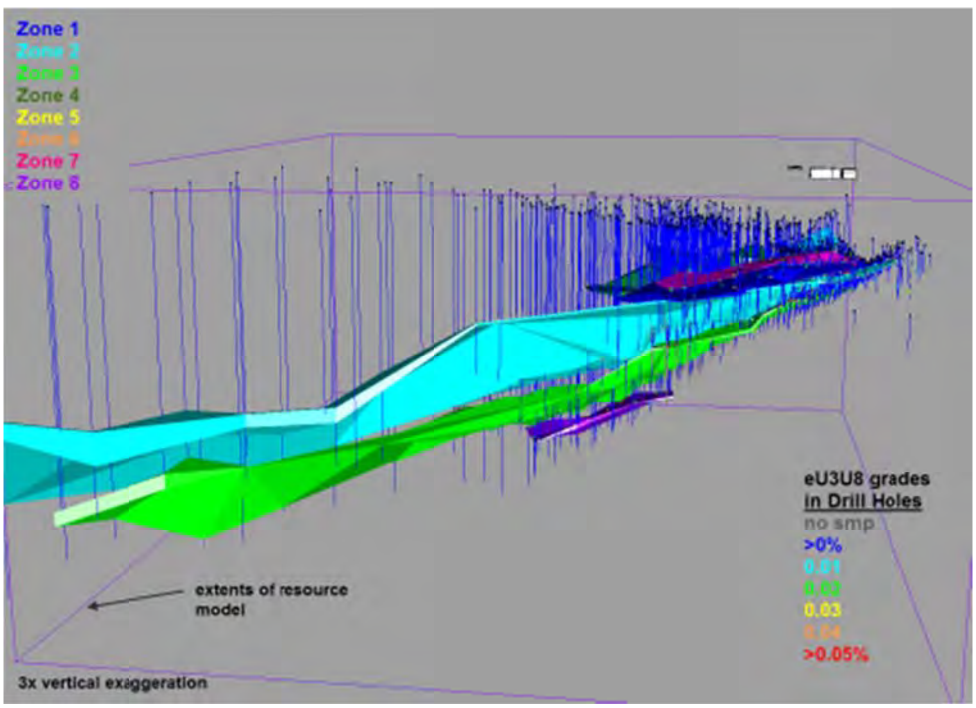


FIGURE 14-3: ISOMETRIC VIEW OF ZONE DOMAINS AND eU_3O_8 GRADES IN DRILL HOLES (VIEW WEST)

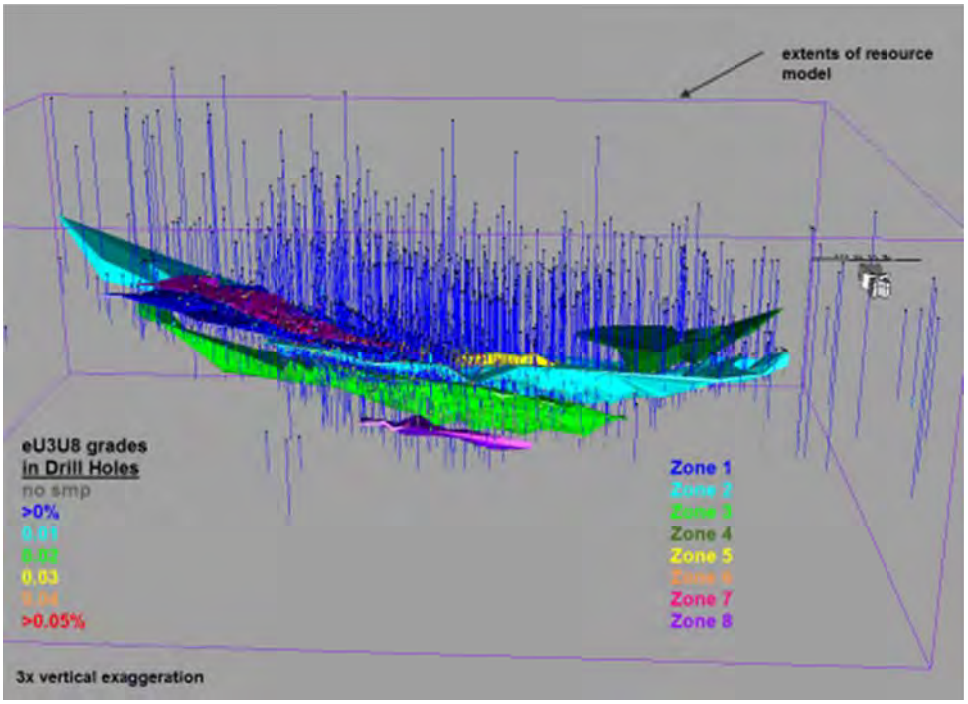


FIGURE 14-4: ISOMETRIC VIEW OF ZONE DOMAINS AND eU_3O_8 GRADES IN DRILL HOLES (VIEW SOUTH)

14.3 AVAILABLE DATA

Sample data has been extracted from an Excel® file (*exported_data.xls*) provided by UEC. Collar data, for 1,343 drill holes, was obtained from spreadsheet tab "MinexUG_all_filtered" (UTM_X, UTM_Y, ELEV_M, TDepth_M). Equivalent U_3O_8 data (eU_3O_8) was obtained from spreadsheet tab "EU3O8_data" comprising 130,956 individual samples. These values represent radiometric gamma log equivalent "assays" that were corrected to account for the disequilibrium compared to chemical assay results.

Of the 214,069 m of drilling, 202,707 m has associated eU_3O_8 sample grades. eU_3O_8 data is present in most samples and this was used to generate the resource model. The assay file also includes values for cU_3O_8, sU_3O_8, V_2O_5, Gamma and Fluor, but these are only present in < 1% of the samples with insufficient coverage for modeling purposes. None of these minor elements were used in the resource model. Limited bulk density (SG) data is also present. The basic statistics of the underlying sample data are listed in Table 14.2.

TABLE 14.2: BASIC SUMMARY OF SAMPLE DATA

Element	# Samples	Total Length of Samples (m)	Minimum	Maximum	Mean	Standard Deviation
eU_3O_8 (%)	120,351	202,707	0	2.642	0.004	0.0149

cU_3O_8 (%)	1,446	437	0.001	2.140	0.033	0.0875
sU_3O_8 (%)	652	131	0.002	0.626	0.046	0.0645
Gamma (%)	1,287	334	0	0.712	0.024	0.0382
Fluor (%)	1,472	493	0	0.630	0.018	0.0379

14.4 COMPOSITING

Drill hole samples are composited to standardize the database for further statistical evaluation. This step eliminates any effect sample lengths may have on the estimate.

Approximately 60% of the samples are 0.03 m long and 23% of the samples are 0.15 m long. Note that all of the shorter samples (0.03 m) were derived from 24 holes drilled by Concentric Energy in 2006. To develop the resource block model, a composite length of 0.5 m was selected.

Drill hole composites are length-weighted and were generated "down-the-hole"; this means composites begin at the top of each hole and are generated at 0.5 m intervals down the length of the hole. The contacts of the Zone domains were honored during compositing of drill holes. Several holes were randomly selected and the composited values were checked for accuracy. No errors were found.

14.5 EXPLORATORY DATA ANALYSIS

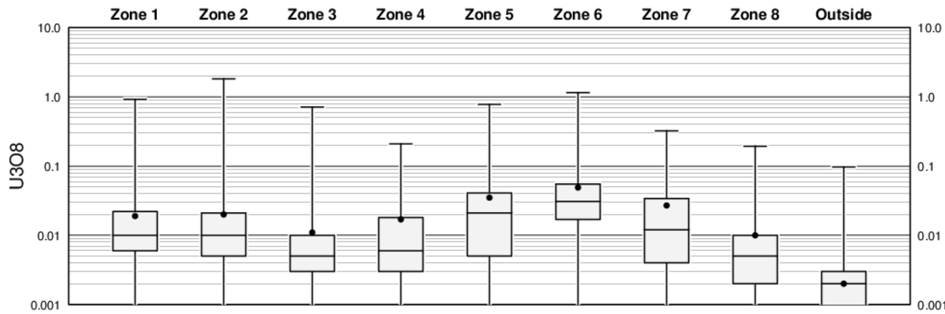
Exploratory data analysis (EDA) involves statistically summarizing the database to quantify the characteristics of the data. One of the main purposes of EDA is to determine if there is any evidence of spatial distinctions in grade; this would require the separation and isolation of domains during interpolation. The application of separate domains prevents unwanted mixing of data during interpolation; this will result in a grade model that better reflects the unique properties of the deposit. However, applying domain boundaries in areas where the data is not statistically unique may impose a bias in the distribution of grades in the model.

A domain boundary, segregating the data during interpolation, is typically applied if the average grade in one domain is significantly different from that of another. A boundary may also be applied where a significant change in the grade distribution exists across the contact.

14.5.1 BASIC STATISTICS BY DOMAIN

The basic statistics for the distribution of eU_3O_8 were generated by Zone domain and are shown in the boxplot in Figure 14-5. Zone 2 is the thickest and most extensive domain interpreted from the sample data. It contains few internal low-grade samples and, as a result, shows a relatively high mean eU_3O_8 grade. Zone 3 also contains a large proportion of the sample data, but drill holes often show two or more higher-grade bands separated by low-grade intervals. As a result, the overall eU_3O_8 mean grade of Zone 2 is relatively low.

Zones 1, 5, 6 and 7 are all relatively thin domains that contain moderate to high-grade eU_3O_8 samples with only rare to no internal low-grade zones. As a result, the mean grades for these thinner zones tend to be higher. Zones 4 and 8 are smaller, somewhat irregular zones with a mix of samples across the grade range. The boxplots show that frequency distributions tend to overlap; this suggests that the domains are spatially, rather than statistically, controlled.



	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5	Zone 6	Zone 7	Zone 8	Outside	
Number of data	2160	29839	9256	547	658	247	620	445	363413	Number of data
Mean	0.019	0.02	0.011	0.017	0.035	0.049	0.027	0.01	0.002	Mean
Variance	0.001	0.001	0.0	0.001	0.003	0.005	0.001	0.0	0.0	Variance
Skewness	12.511	13.582	11.127	3.486	6.394	8.232	3.094	5.251	4.953	Skewness
Coef of Variation	1.812	1.922	1.97	1.711	1.618	1.447	1.394	1.522	0.91	Coef of Variation
Maximum	0.924	1.825	0.711	0.209	0.78	1.156	0.324	0.194	0.097	Maximum
Upper quartile	0.022	0.021	0.01	0.018	0.041	0.055	0.034	0.01	0.003	Upper quartile
Median	0.01	0.01	0.005	0.006	0.021	0.031	0.012	0.005	0.002	Median
Lower quartile	0.006	0.005	0.003	0.003	0.005	0.017	0.004	0.002	0.0	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	Minimum

FIGURE 14-5: BOXPLOT OF eU_3O_8 GRADES BY ZONE DOMAIN

14.5.2 CONTACT PROFILES

Contract profiles evaluate the nature of grade trends between two domains; they graphically display the average grades at increasing distances from the contact boundary. Those contact profiles that show a marked difference in grade across a domain boundary indicate that the two datasets should be isolated during interpolation. Conversely, if a more gradual change in grade occurs across a contact, the introduction of a "hard" boundary (i.e., segregation during interpolation) may result in much different trends in the grade model; in this case, the change in grade between model domains is often more abrupt than the trends seen in the raw data. Finally, a flat contact profile indicates that there are no grade changes across the boundary; in this case, "hard" or "soft" domain boundaries will produce similar results in the model.

A combined contact profile compares the grades inside the interpreted Zone domains with surrounding samples (Figure 14-6). There is a distinct change in eU_3O_8 grade across the Zone domain contacts indicating that the domains host samples are distinctly different from those outside the domains. This difference supports the use of the Zone domains as hard boundaries during model grade estimations.

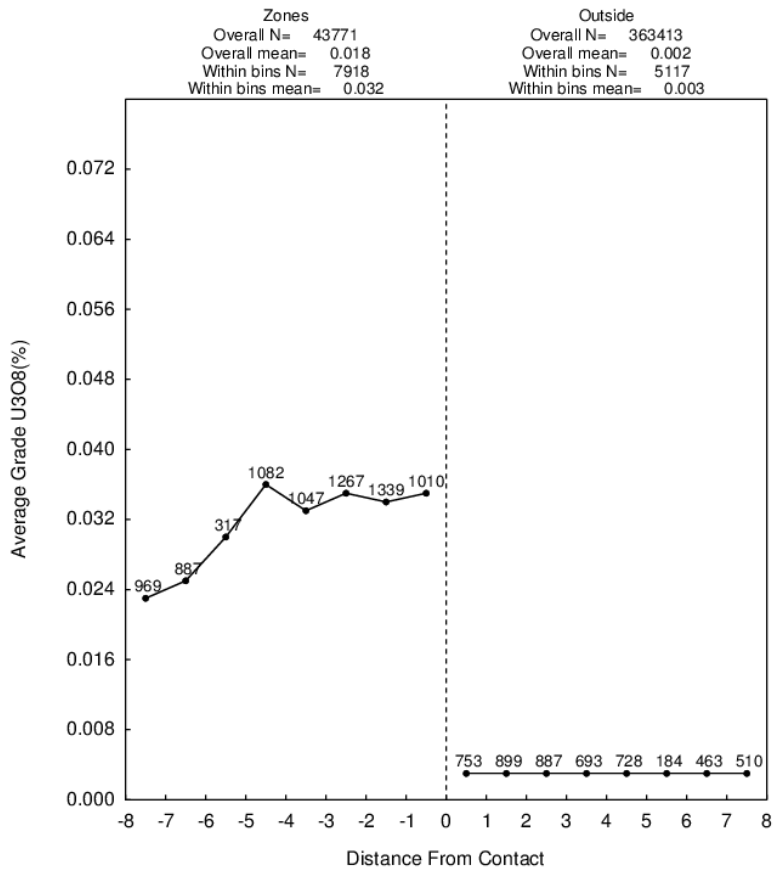


FIGURE 14-6: CONTACT PROFILE eU_3O_8 GRADES INSIDE AND OUTSIDE ZONE DOMAINS

14.5.3 CONCLUSIONS AND MODELING IMPLICATIONS

The boxplots show minor differences between Zone domains but distinct differences compared to the surrounding low-grade sample data. This trend is supported by the contact profile. These results indicate that the sample data contained within the Zone domains should be separated from the surrounding low-grade samples during model grade interpolations. Each Zone domain was interpreted to envelope individual trends, or seams, of higher eU_3O_8 grades, each separated by intervals of low-grade material. As a result, each Zone domain will be treated as a "hard" boundary during block grade estimations and the data between Zones will not be mixed during this process.

14.6 BULK DENSITY DATA

Bulk density sample data, referred here as specific gravity (SG) data, is limited to only 48 drill holes in the project area. The majority of the SG data is from Zone 2 with little or no data available from the other domains. There is insufficient data to estimate (interpolate) the SG values in the block model; therefore, a constant bulk density value was used to determine resource tonnage.

Bulk density data was collected from core intervals within mineralized intervals. The SG sample average is 1.70 t/m^3 and ranges from 1.00 t/m^3 to 2.51 t/m^3. Historically, an average value of 1.56 t/m^3 was used to calculate resource tonnage for the Anderson deposit. Comments in previous technical reports suggest that the SG value represented near-surface rocks and that the density increases to as high as 1.9 t/m^3 with depth. Due to insufficient sample data and the fact that more potentially important economic resources tend to occur near surface, a constant value of 1.65 t/m^3 was used to calculate tonnages from the resource model. This assumption is considered reasonable and somewhat conservative based on available information. Further bulk density testing is recommended.

14.7 EVALUATION OF OUTLIER GRADES

Histograms and probability plots were reviewed to identify the existence of anomalous outlier grades in the composited sample database. Following a review of the physical location of these potentially anomalous samples, it was decided that outlier sample data could be controlled through an "outlier limitation" limiting their effective distance to a maximum of 25 m during interpolation. These parameters and resulting effects on the resource model are summarized by Zone in Table 14.3. The amount of metal lost in some domains is relatively high; however, the overall decrease is only 1.2% which is considered appropriate for a deposit (and database) of this size.

TABLE 14.3: SUMMARY OF OUTLIER LIMITATIONS

Zone Domain	Threshold Grade eU_3O_8%	# Comps Affected	% Metal Loss in Model
Zone 1	0.25	7	-2.1%
Zone 2	0.60	15	-0.6%
Zone 3	0.25	11	-1.8%
Zone 4	0.20	2	-1.0%
Zone 5	0.25	7	-9.1%
Zone 6	0.20.	8	-12.0%
Zone 7	0.20	5	-2.9%
Zone 8	0.10	1	-1.8%
Zone 1-8	As above	56	-1.2%

Note: (0.5 m composited DH data. Outlier data limited to maximum distance of 25m during interpolation)

14.8 TREND CONTROLS AND RELATIVE ELEVATIONS

The distribution of eU_3O_8 in the deposit tends to occur in a series of bands roughly parallel to the overall trend of the interpreted Zone domains at a general orientation of -10 degrees to the south. However, local undulations occur within individual Zones. To replicate these trends in the resource block model, a series of "trend" planes were generated in the center of each Zone domain (the hanging wall and footwall contacts of each domain were averaged to create the trend planes). These trend planes are used to control search orientations during block grade interpolations. Referred to as “relative elevations”, these values are used to match composited sample data and blocks in the model during grade interpolations that result in search orientations that conform to the natural undulations present throughout the deposit area. The resulting models retain more of the inherent banded nature of the deposit. Without relative elevations, simple search ellipses would be very difficult to orient during interpolation, resulting in excessive smoothing (averaging) of sample data. Using relative elevations allows for reproduction of vertical grade trends within the domains.

14.9 VARIOGRAPHY

The degree of spatial variability in a mineral deposit depends on both the distance and direction between points of comparison. Typically, the variability between samples is proportionate to the distance between samples. If the degree of variability is related to the direction of comparison, then the deposit is said to exhibit anisotropic tendencies which can be summarized with the search ellipse. The semi-variogram is a common function used to measure the spatial variability within a deposit.

The components of the variogram include the nugget, the sill and the range. Often samples compared over very short distances (including samples from the same location) show some degree of variability. As a result, the curve of the variogram often begins at some point on the y-axis above the origin; this point is called the *nugget*. The nugget is a measure of not only the natural variability of the data over very short distances, but also a measure of the variability which can be introduced due to errors during sample collection, preparation and assaying.

The amount of variability between samples typically increases as the distance between the samples increases. Eventually, the degree of variability between samples reaches a constant or maximum value. This is called the *sill* and the distance between samples at which this occurs is called the *range*. The variogram parameters for each zone are summarized in Table 14.4.

The spatial evaluation of the data was conducted using a correlogram instead of the traditional variogram. The correlogram is normalized to the variance of the data and is less sensitive to outlier values; this generally gives cleaner results.

Correlograms were generated using “relative elevations” as described in Section 14.8 of this report. This results in spatial relationships that are relative to the overall trend of each of the Zone domains.

Variograms were generated using the commercial software package Sage 2001© (developed by Isaacs & Co.). Multidirectional variograms were generated from composited eU_3O_8 sample data. There are not enough individual samples present in Zones 4-8 and, as a result, the data from these five domains were combined for variogram purposes. Experience has shown that it usually takes more composite samples to infer a reasonable model for directional spatial correlation than occur in each of the individual Zones 4-8. Since there is little reason to believe the mineralization deposition mechanisms are significantly different from zone to zone, a correlogram inferred from the combined Zones 4-8 is likely to perform better than more uncertain correlograms from the separate zones.

TABLE 14.4: VARIOGRAM PARAMETERS – eU_3O_8

				1st Structure			2nd Structure		
Zone Domain	**Nugget**	**Sill 1**	**Sill 2**	**Range (m)**	**Azimuth**	**Dip**	**Range (m)**	**Azimuth**	**Dip**
Zone 1	0.500	0.398	0.102	130	11	0	133	86	0
	Spherical			17	101	0	63	356	0
				2	90	90	5	90	90
Zone 2	0.280	0.668	0.052	17	65	0	5674	296	0
	Spherical			10	155	0	909	26	0
				2	90	90	16	90	90
Zone 3	0.600	0.339	0.061	20	341	0	616	169	0
	Spherical			3	71	0	507	79	0
				3	90	90	15	90	90
Zone 4-8	0.450	0.416	0.134	45	113	0	423	14	0
	Spherical			6	23	0	286	284	0
				2	90	90	8	90	90

Note: Correlograms conducted on 0.5m DH composite data.

14.10 MODEL SETUP AND LIMITS

A block model was initialized in MineSight® and the dimensions are defined in Table 14.5. The selection of a nominal block size measuring 10 x 10 x 2 mV is considered appropriate with respect to the current drill hole spacing and the selective mining unit (SMU) size typical of an operation of this type and scale. Note that selectivity in an open pit mining method may be smaller than 2 mV; however, this approach allows for evaluation of the entire deposit and provides an indication of what portion may be amenable to open pit versus underground mining methods.

TABLE 14.5: BLOCK MODEL LIMITS

Direction	Minimum	Maximum	Block size (m)	# Blocks
East	287500	293000	10	550
North	3795000	3799000	10	400
Elevation	50	750	2	350

(Note: block model is not rotated)

Using the domain wireframes, blocks in the model are assigned Zone code values on a majority basis. During this stage, blocks that occur within two Zone domains are coded if greater than 50% of the block occurs within the boundaries of that domain. The percentage of each block inside the Zone domains is also stored in each block in the model. Zone percent values allow for the accurate determination of resource tonnage that occurs along Zone domain boundaries.

The proportion of blocks which occur below the topographic surface are also calculated and stored within the model as individual percentage items. These values are used as a weighting factor to determine the in-situ resources for the deposit.

14.11 INTERPOLATION PARAMETERS

The block model grades for eU_3O_8 were estimated using Ordinary Kriging (OK). The results of the OK estimation were evaluated using a series of validation approaches (described in the Section 14.12 of this report). The interpolation parameters were adjusted until the appropriate results were achieved.

The Anderson model was generated with relatively few (less than a maximum of 25 per block) samples. This approach reduces the amount of smoothing (or averaging) in the model and, while there may be some uncertainty on a localized scale, this approach produces reliable recoverable grade and tonnage estimates for the overall deposit. (The method is outlined in Isaaks & Co., “The Kriging Oxymoron”, SME Preprint, 1999)

The interpolation parameters are summarized here:

- Search ellipse: 200 x 200 x 50 mV. Long axis oriented at -10 degrees to the south. Vertical range also limited to 10 m based on vertical distances relative to the local trend planes of each Zone domain.
- Maximum of 6 sample composites from a single drill hole, minimum of 1 composite to estimate the grade in a block and a maximum of 18 composites to estimate the grade of a block.
- Quadrant search limitation with a maximum of 1 drill hole per quadrant.
- “Hard” boundaries between Zone domains (no mixing of data between Zone domains).

14.12 VALIDATION

The results of the modeling process were validated through several methods including a thorough visual review of the model grades in relation to the underlying drill hole sample grades; comparisons with the change of support model; comparisons with other estimation methods; and grade distribution comparisons using swath plots.

VISUAL INSPECTION

A detailed visual inspection of the block model was conducted in both cross section and plan to ensure the desired results following interpolation. This included confirmation of the proper coding of blocks within the respective Zone domains and below the topographic surface. The distribution of block grades was also compared relative to the drill hole samples to ensure the proper representation in the model.

In general, all grade models showed the desired degree of correlation with the underlying sample data. An example of the distribution of block grades in the model is shown in a north-south oriented vertical section in Figure 14-7.

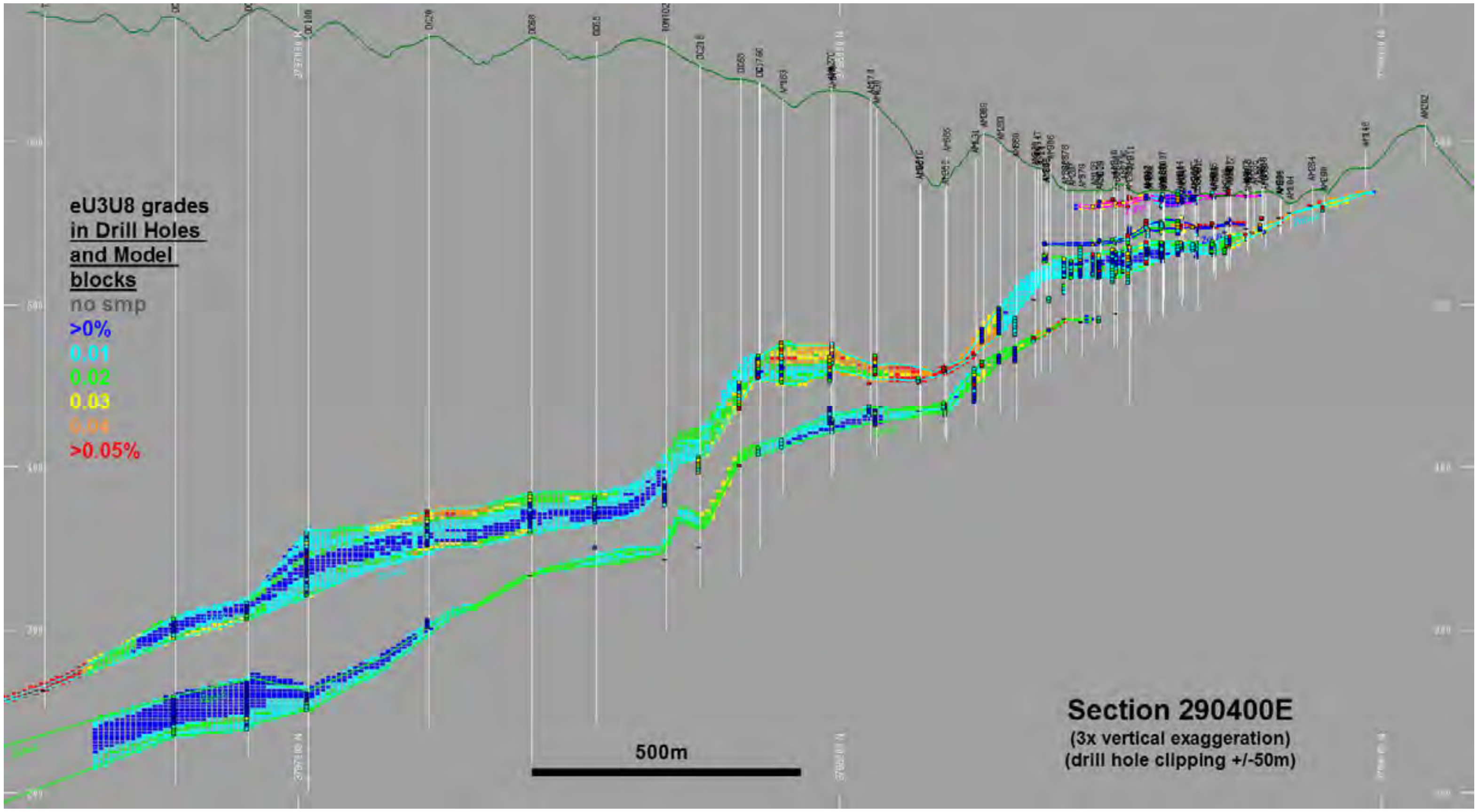


FIGURE 14-7: SECTIONAL VIEW OF eU_3O_8 GRADES IN DRILL HOLES AND MODEL BLOCKS

MODEL CHECKS FOR CHANGE OF SUPPORT

The relative degree of smoothing in the block model estimates were evaluated using the Discrete Gaussian or Hermitian Polynomial Change of Support (Herco) method (Journel and Huijbregts, Mining Geostatistics, 1978). With this method, the distribution of the hypothetical block grades is directly compared to the estimated (OK) model using of pseudo-grade/tonnage curves. Adjustments are made to the block model interpolation parameters until an acceptable match is made with the Herco distribution. In general, the estimated model should be slightly higher in tonnage and slightly lower in grade when compared to the Herco distribution at the projected cut-off grade. These differences account for selectivity and other potential ore-handling issues which commonly occur during mining.

The Herco (*Her*mitian *co*rrection) distribution is derived from the declustered composite grades which were adjusted to account for the change in support moving from smaller drill hole composite samples to the larger blocks in the model. The transformation results in a less skewed distribution but with the same mean as the original declustered samples.

Herco and model grade-tonnage plots were generated for the distribution of eU_3O_8 in the main Zone domains (Zones 1, 2, and 3, and combined 4-8). The results are shown in Figures 14-8 to 14-11. Overall, there is an acceptable degree of correlation between the models. The results for Zones 2 and 3 are very good. These domains represent the majority of the potential resource in this deposit. Zone 1 is somewhat thin and represents a smaller portion of resources and, as a result, shows somewhat more erratic results.

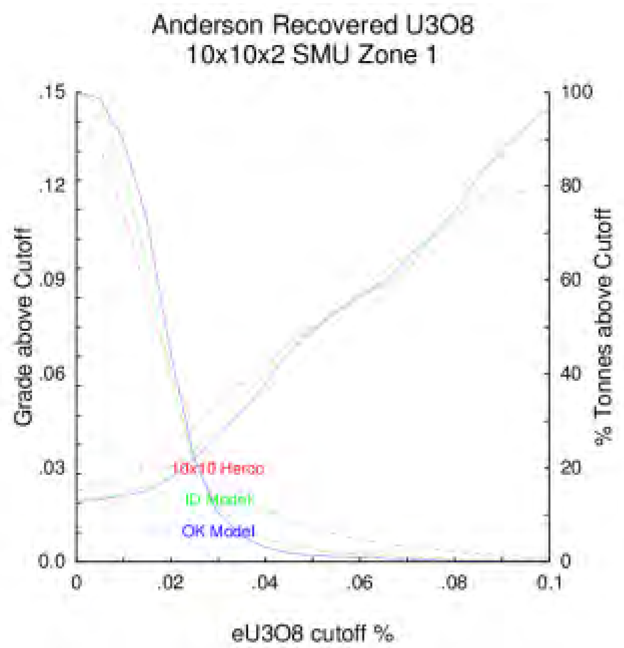


FIGURE 14-8: HERCO ZONE 1

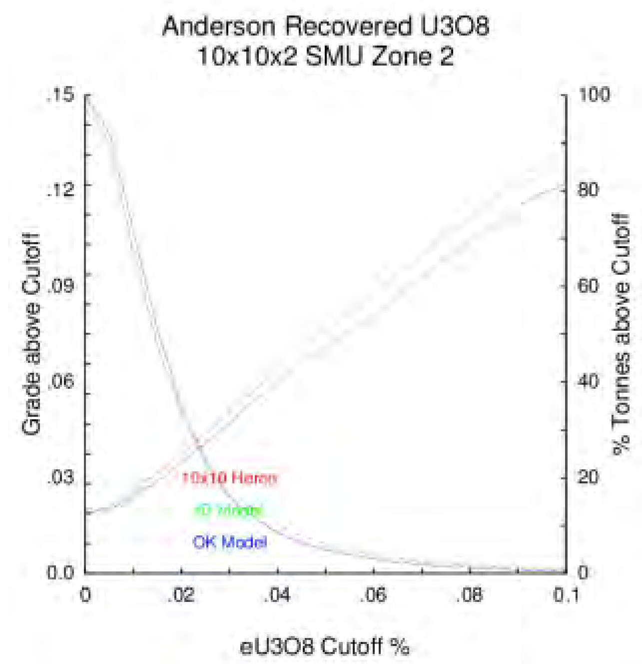


FIGURE 14-9: HERCO ZONE 2

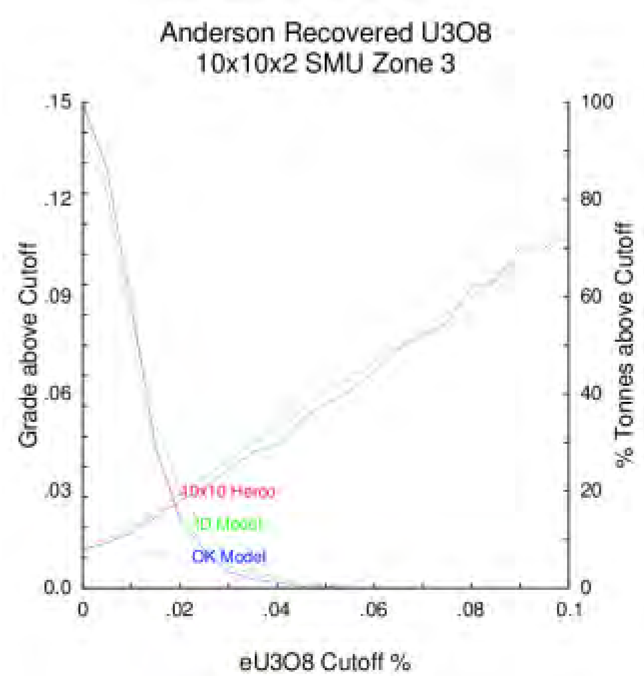


FIGURE 14-10: HERCO ZONE 3

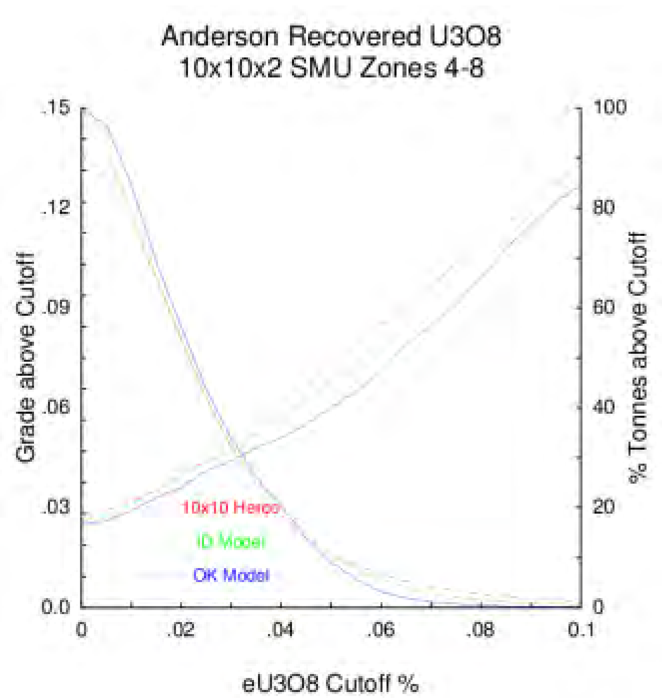


FIGURE 14-11: HERCO ZONES 4-8

COMPARISON OF INTERPOLATION METHODS

For comparison purposes, additional models for eU_3O_8were generated using both the inverse distance weighted (IDW) and nearest neighbour (NN) interpolation methods. (The NN model was created using data composited to 2 m intervals.) The results of these models are compared to the OK models at various cut-off grades in the grade/tonnage graphs shown in Figure 14-12. These curves were generated from all Zone domains combined. Overall, there is an acceptable degree of correlation between these models. Reproduction of the model using different methods tends to increase the level of confidence in the overall resource.

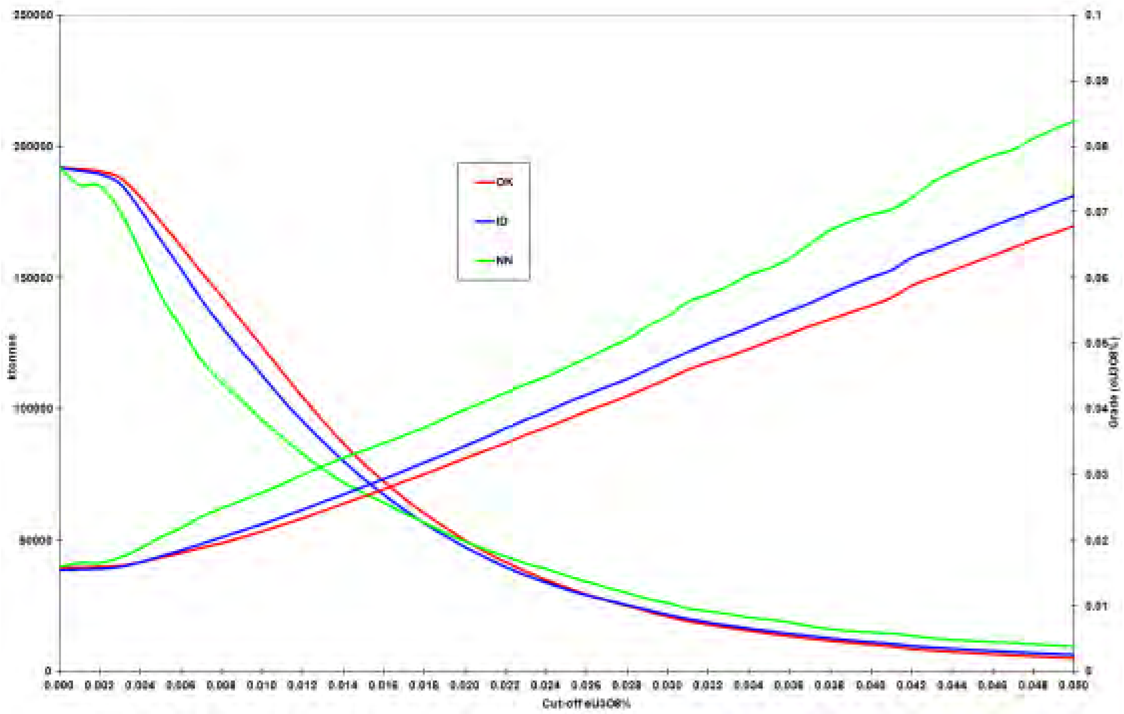


FIGURE 14-12: GRADE TONNAGE COMPARISON OF MODEL TYPES

SWATH PLOTS (DRIFT ANALYSIS)

A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions through the deposit. Grade variations from the OK model are compared using the swath plot to the distribution derived from the declustered (NN) grade model.

On a local scale, the NN model does not provide reliable estimations of grade but, on a much larger scale, it represents an unbiased estimation of the grade distribution based on the underlying data. Therefore, if the OK model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the NN distribution of grade.

Swath plots were generated in three orthogonal directions comparing the OK and NN distributions of eU_3O_8 in the deposit. Examples from Zones 2 and 3 by northing (WE swaths) are shown in Figures 14-13 and 14-14.

Overall, there is good correspondence between the models through most of the deposit area. The somewhat poor correlation north of 3798500N represents only a small volume of resources as the deposit pinches out to the north. The large swing in grades south of 3796600N is the result of the wider-spaced drilling in the deeper parts of the deposit.

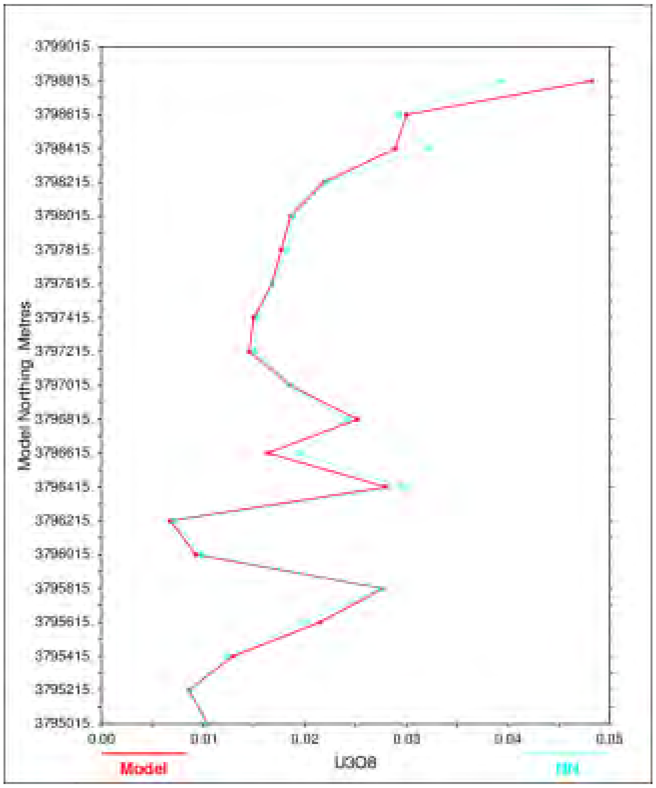


FIGURE 14-13: W-E SWATH PLOT OF ZONE 2

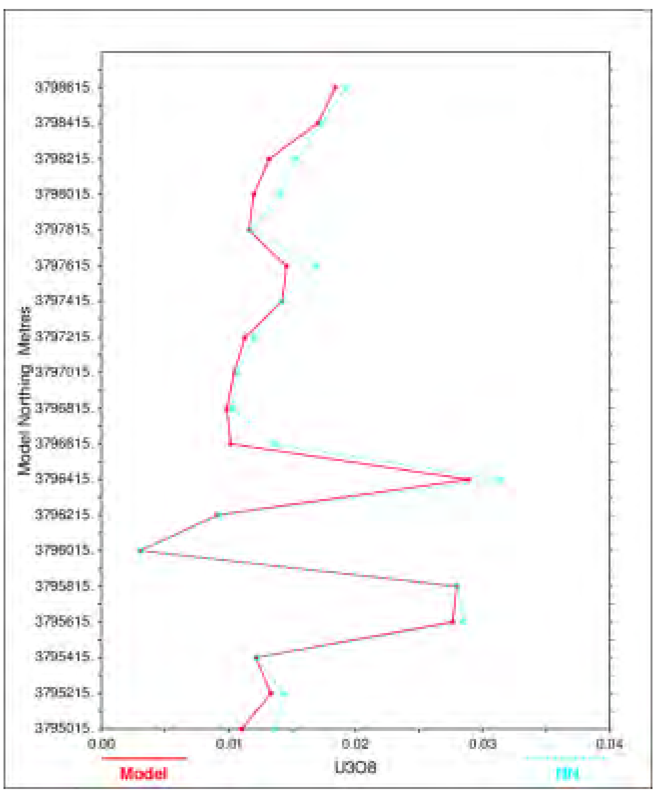


FIGURE 14-14: W-E SWATH PLOT OF ZONE 3

14.13 RESOURCE CLASSIFICATION

Mineral resources for the Anderson project were classified according to the CIM definition standards for Mineral Resources and Reserves (CIM, 2005). The classification parameters are defined relative to the distance between sample data and are intended to encompass zones of reasonably continuous mineralization.

Variograms, generated from both eU_3O_8 grade and indicator variograms, were reviewed, together with evidence gained from the visual interpretation of the drilling results, to understand the classification criteria for the mineral resources at the Anderson deposit.

At this stage, more substantial work needs to occur on the historic data to gain the level of confidence required to classify resources in the Measured category.

Indicated resources are defined as areas delineated by continuous drilling with a maximum grid spacing of about 75 m. Portions of the deposit that meet these initial criteria were further reviewed to ensure that they exhibit the appropriate continuity to support the level of confidence required for resources in the Indicated category.

Inferred resources include blocks within a maximum distance of 150 m from a drill hole.

The extent of resources in the Indicated and Inferred categories are shown in Figure 14-15.

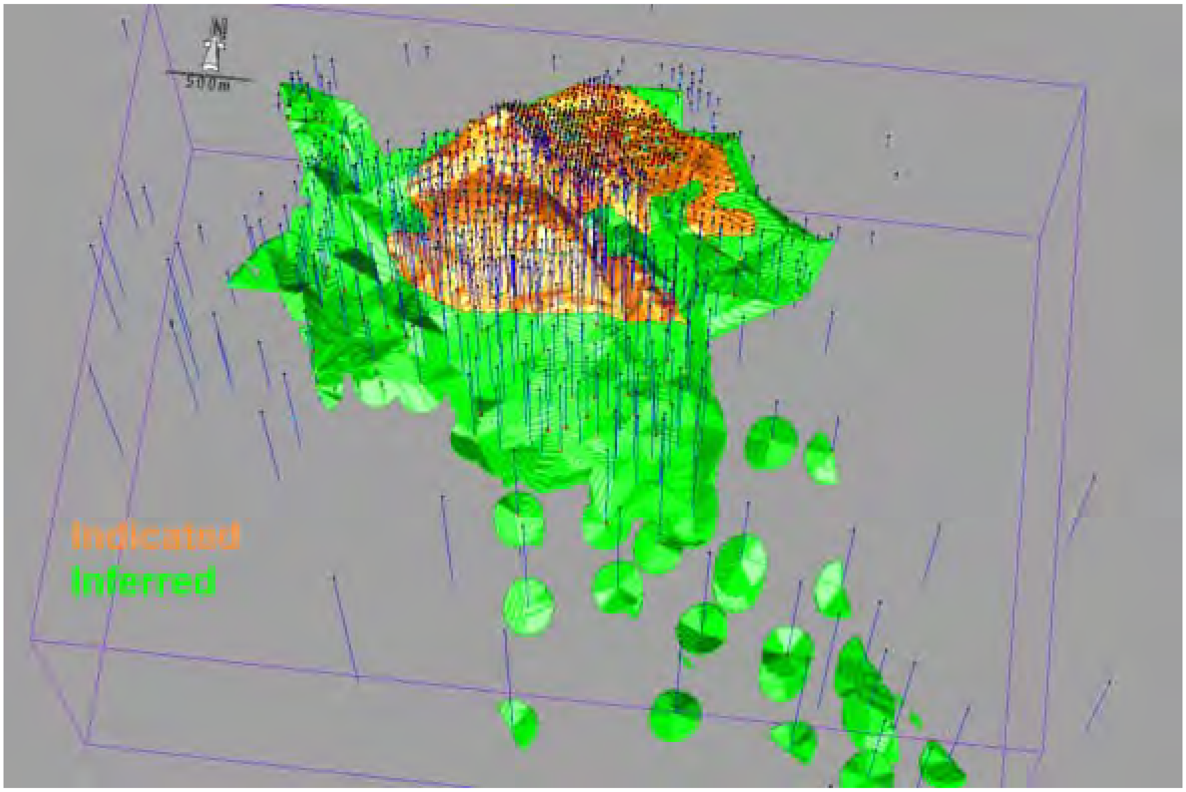


FIGURE 14-15: DISTRIBUTION OF RESOURCE CATEGORIES IN THE DEPOSIT AREA

14.14 MINERAL RESOURCES

When stating mineral resources, the requirements of NI 43-101 state that resources must exhibit reasonable prospects for economic extraction. The shape, location and grade distribution of the Anderson deposit indicates that it could be extracted through a combination of surface and underground mining methods. Initially, the resource is limited within a pit shell generated about the economic and technical parameters listed below. The remaining resource, outside of the pit shell, can be possibly extracted through underground mining methods.

It is believed that these parameters represent reasonable assumptions based on the type, scale, shape and location of this deposit:

U_3O_8 Price: US$65/lb
Open Pit Operating Costs:
Mining$2/tonne
Total Site Operating Costs $14/tonne
Pit Slope Angle:45 degrees
Underground Operating Costs:
Total Site Operating Costs$50/tonne

In the southeastern part of the deposit, the wider drill hole spacing results in a discontinuous or patchy distribution of potential resources that does not fulfill the requirements for mineral resources under NI 43-101. Additional drilling is required in this part of the deposit area to demonstrate continuity of the mineralization.

Mineral resources for the Anderson Project are summarized at various cut-off grades for comparison purposes in Table 14.6. These resources were separated into two categories: Open Pit and Underground resources. These two resources have different operating costs and the respective "base case" cut-off grades are highlighted in Table 14.6. The assumed base case cut-off for open pit mining is 0.01%U_3O_8, and the base case for potential underground resources is 0.035%U3O8. The distribution of base case resources is shown in Figure 14-16.

There are no known factors related to environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues which could materially affect the mineral resource.

TABLE 14.6: SUMMARY OF MINERAL RESOURCES

Cut-off Grade eU_3O_8%	K tonnes	K tons	eU_3O_8 (%)	Contained U_3O_8 (Mlbs)
	OPEN PIT RESOURCES			
	INDICATED			
0.005	28,034	30,902	0.026	16.0
0.01	**25,422**	**28,023**	**0.028**	**15.5**
0.015	19,834	21,863	0.032	14.0
0.02	15,008	16,543	0.037	12.3
0.025	11,355	12,517	0.042	10.5
0.03	8,584	9,462	0.047	8.9
0.035	6,445	7,104	0.052	7.3
	INFERRED			
0.005	5,478	6,038	0.022	2.6
0.01	**4,633**	**5,107**	**0.024**	**2.5**
0.015	3,341	3,683	0.029	2.2
0.02	2,324	2,562	0.035	1.8
0.025	1,670	1,841	0.040	1.5
0.03	1,192	1,314	0.045	1.2
0.035	897	989	0.049	1.0
	UNDERGROUND RESOURCES			
	INDICATED			
0.005	38,177	42,083	0.015	12.5
0.01	25,209	27,788	0.019	10.5
0.015	15,213	16,769	0.024	7.9
0.02	8,570	9,447	0.029	5.4
0.025	4,494	4,954	0.035	3.5
0.03	2,555	2,816	0.042	2.3
0.035	**1,426**	**1,572**	**0.049**	**1.5**
0.04	871	960	0.057	1.1
0.045	581	640	0.065	0.8
0.05	382	421	0.074	0.6
	INFERRED			
0.005	123,286	135,899	0.016	44.6
0.01	85,483	94,229	0.021	38.8
0.015	52,298	57,649	0.026	30.2
0.02	32,330	35,638	0.032	22.8
0.025	20,423	22,512	0.038	17.1
0.03	12,164	13,408	0.045	12.1
0.035	**8,362**	**9,218**	**0.052**	**9.5**
0.04	6,046	6,665	0.057	7.6
0.045	3,572	3,937	0.068	5.4
0.05	2,896	3,192	0.073	4.7

Notes: "Base case" cut-off for resources amenable to open pit extraction methods is 0.01%U_3O_8. "Base case" for remaining resources extracted by underground mining methods is 0.035%U_3O_8. Effective date of mineral resource estimate April 15, 2012.

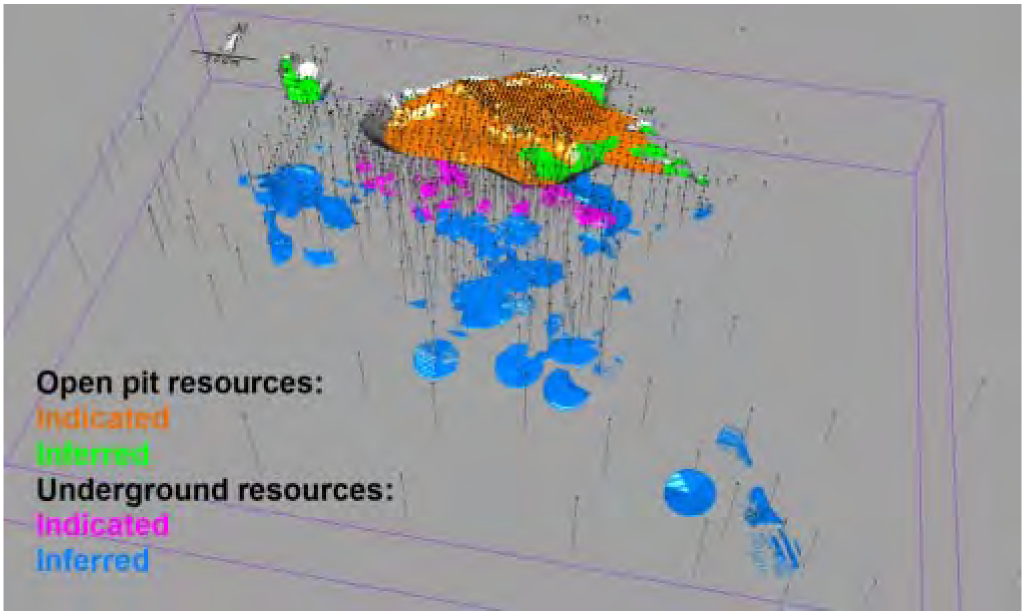


FIGURE 14-16: DISTRIBUTION OF BASE CASE RESOURCES

15 MINERAL RESERVE ESTIMATE

This section is not applicable.

16 MINING METHOD

16.1 INTRODUCTION

The PEA is based on conventional open pit and underground mining utilizing methodologies, equipment, and a generalized design criterion appropriate to the project site and morphology of mineralization. The geologic and topographic setting of the Anderson Uranium Project is such that mineralization occurs at the surface in the north and reaches depths in excess of 1,800 to the south due the geologic dip and local terrain. As a result portions of the deposit are amenable to open pit mining methods while others are more suited to underground methods. The current conceptual design approach includes conventional mining via open pit, highwall mining, and underground mining. Mineral processing will be accomplished via heap leach producing a loaded resin for shipment and final processing into uranium oxide (yellowcake). The conceptual mine plan which is the basis of the cost PEA is shown on Figure 16-1: Project Layout.

This is a restricted disclosure as allowed under section 2.3(3) of NI 43-101 which includes a Preliminary Economic Assessment (PEA). It is also preliminary in nature such that it includes portions of both indicated and inferred mineral resources, as reported in Section 14 of the report. The PEA is based on open pit mining and heap leach extraction of uranium values, utilizing methodologies, equipment, and a generalized design criterion which has been employed at the site and/or similar sites in the past but has not been specifically developed for the Project. Mineral resources are not mineral reserves and do not have demonstrated economic viability in accordance with CIM standards. Inferred mineral resources are too speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the outcomes estimated in the PEA will be realized.

Specific risks related to the PEA include;

- Mine planning and scheduling is conceptual in nature as is the heap leach and processing facility.
- Reclamation and closure design have not be completed. Cost allowances in the PEA for reclamation and closure are based on previous experience of the authors.
- Geotechnical studies specific to the conceptual mine plans have not been completed.

16.2 MINERAL RESOURCES USED FOR PEA

The Indicated and Inferred Mineral Resources used in the PEA, Table 16.1, are fully included in the total Indicated and Inferred mineral resources reported in Section 14. They are that portion of the mineral resources which meet minimum cutoff criterion and are incorporated within conceptual mine designs, as further discussed herein. This is a restricted disclosure as allowed under section 2.3(3) of NI 43-101 which includes a Preliminary Economic Assessment (PEA). It is also preliminary in nature such that it includes portions of both indicated and inferred mineral resources, as reported in Section 14 of the report. Mineral resources are not mineral reserves and do not have demonstrated economic viability in accordance with CIM standards. Inferred mineral resources are too speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the outcomes estimated in the PEA will be realized.

Based on the three dimensional limits of the conceptual mine layout shown on Figure 1, the mineral resources used in the PEA are summarized on Table 16.1. This estimate was completed for the purposes of the PEA utilizing the verified database as used in the mineral resource estimate, Section14, but differing from that estimate in that the estimate of mineral resources included in the mining plan was based on a bulk dry density of 18 cubic feet per ton and the estimate was completed using the GT contour method. The open pit will intercept the A, B, and C mineralized zones. Highwall and underground mining will be in the C zone only.

TABLE 16.1 - MINERAL RESOURCES USED FOR PEA*

	Tons (x 1,000)	Contained Pounds U_3O_8 (x 1,000)	Recovery Factor	Tons (x 1,000)	Mining Recovered Pounds U_3O_8 (x 1,000)	Grade % eU_3O_8	Heap Recovered Pounds U_3O_8 (x 1,000)
OPEN PIT	7,667	10,342	0.9	6,900	9,308	0.067	8,377
HIGHWALL MINER	2,153	3,411	0.75	1,615	2,558	0.079	2,303
Underground	5,669	9,951	0.6	3,401	5,970	0.088	5,373
TOTAL	11,917	23,704	0.75	11,917	17,836	0.075	16,053

*Includes both Indicated and Inferred Mineral Resources

The conceptual plan, as shown on Figure 16-1, includes open pit, highwall and underground mining. Open pit mining would access approximately 60% of the total recoverable pounds, while highwall mining would access approximately 14%, and underground mining the remaining 26%.

The initial heap leach pad and facility would be located immediately adjacent to the pit. This area is partially disturbed by past mining and is located in a basin which would facilitate hydrologic isolation of the facilities. Conceptually the initial pad could be reused by off-loading spent heap material or additional pads could be constructed in potions of pit as it is sequentially backfilled.

In terms of the cost model there is little impact on the project. Off-loading has higher operating costs and lower capital requirements while successive pad construction is the opposite, higher capital but lower operating costs. The PEA assumed that three additional heap leach pads would be constructed during the project as needed.

16.3 Determination of Mine Cutoff Grade

Mineral resources for the purposes of the PEA were calculated based on a minimum mining thickness of 2 feet; a minimum grade of 0.02 % eU3O8; and a minimum GT of 0.10. Conceptual mine designs focused on the areas with the most extensive mineralization and sought to identify areas with stripping ratios generally less than 15:1, expressed as cubic yards stripped to pounds contained, and average diluted mining grade of 0.05% eU3O8 or greater. The PEA shows total direct operating costs of approximately $47 per ton and capital costs of approximately $12.50 per ton. Based on a commodity price of $65 per pound and a mineral processing recovery of 90%, the overall breakeven grade would be approximately 0.05% eU3O8 or a GT of 0.10. Marginal costs, assuming the material must be excavated as part of the overall mine plan regardless, including only direct mining, mineral processing and severance tax, are approximately $25 per ton. Mineralized material encountered below the mine GT cutoff, which has to be excavated as part of the mine plan and would otherwise be disposed of as mine waste, could be salvaged at grades as low as a calculated breakeven grade of 0.022 % U3O8 based on the marginal costs. Both highwall mining and underground mining require a greater thickness and thus a higher GT. For these areas a minimum GT of 0.50 was targeted in the conceptual designs.

16.4 Geotechnical Studies

Previous studies indicate that open pit highwall slopes of 1:1 are stable up to 600 feet in height (MinEx, 1978). With respect to highwall and underground mining, specific geotechnical studies have not been completed.

For open pit conceptual planning an average pit slope of 1:1 was used. The pit slope affects the amount of overburden that must be removed to allow mining of the mineralized material and thus affects stripping ratio and mining costs. The author considers the 1:1 pit slope to be a conservative value.

For underground mining geotechnical conditions related to roof support will affect mine recoveries. The PEA used manufactures' recommendations for the highwall mining recovery. For room and pillar underground mining the extraction ratio for underground mining was based on handbook values (SME, 2011).

16.5 OPEN PIT

Open pit mining has two major facets, primary stripping or the removal of overburden and the mining of the mineralization as it is exposed by the stripping. Primary stripping would operate 2, 10 hour shifts per day on a continuous basis. Manning would consist of 3 rotating shifts with each operator working approximately 208 days per year or 2,080 hours. Mining would be accomplished on a single daylight shift operating 5, 10 hour days for approximately 260 days per year. If it were necessary to increase production, it is recommended that the mining remain a daylight operation for grade control purposes but the days and shifts be extended, i.e., 2 rotating crews working 4, 10 to 12 hour shifts on a continuous basis.

The pit would begin in the shallower areas to the north (lower stripping ratio) and proceed to the deeper portions of the pit to the south and west. Mining will be predominantly in the C zone although the pit will intercept some mineralization in the A and B zones. For the purposes of the PEA the open pit was sequenced into 9 open pits over a 10 year period as shown schematically of Figure 16.2, Pit Sequencing. It was assumed that ¼ of the volume of the pit material would be re-handled to reclaim the pit and heap facilities. This sequence is conceptual and has not been optimized. A pit highwall sloped of 1:1 was used in the conceptual designs. The overall conceptual pit sequencing can be seen in Figure 16-2. Pit sequence by year can be seen in Figures 16-3 through 16-11.

Mine CAPEX includes Cat 657 twin engine scrapers and the primary stripping fleet supported by 2 dozers, a motorgrader, water truck and ancillary equipment. Initially 4 scrapers would be employed with an additional 4 scrapers added in year 2 of the mine life. With respect to mining, the cost model is based on selective mining accomplished by a 3cy excavator operating with 4 haul trucks and ancillary equipment.

16.6 HIGHWALL MINING

Highwall mining is applicable in situations where the open pit stripping ratio exceeds reasonable economic limits and there is access to mineralization continuing from an open pit highwall, trench cut, or outcrop. The conceptual pit design reaches a depth of 600 feet along the southern pit limits. Along the southern wall of the pit mineralization in the C zone is continuous and extends below the highwall. Highwall mining can begin in the east and central pit areas in year 4.

Highwall mining is an historic mining practiced with the most common method employing large augers. The highwall miner developed by Bucyrus (pictured below) has only been available since 2010. This equipment is gaining popularity in thin seam coal mining and conceptually would be applicable to the Anderson Project for mineralization which extends from the pit wall approximately 1,000 feet or less. The advantage over conventional underground mining is lower cost operation, lower capital costs, and higher recovery. For the base case 75% extraction was applied to the highwall mining areas..

16.7 UNDERGROUND

As shown on Figure 16-1, even with the extension of the open pit to its maximum limits, and application of highwall mining where possible, additional resources would only be accessible via underground mining. The conceptual mine plan incorporates a room and pillar mine operation with decline access from the open pit highwall. Underground mining would be exclusively in the C zone and could begin as early as year 6 following the completion of pit 4.

Based on the tenor of the mineralization, i.e. tabular, continuous, and reasonable in thickness, 6 to 10 feet within the conceptual mine area; the cost model includes CAPEX and OPEX for underground mine operations utilizing a continuous miner. For the base case 60% extraction was applied with no pillar recovery or retreat mining.

16.8 MINE CAPEX

Mine CAPEX cost were estimated for equipment, pre-production, and surface facilities. Pre-production costs include delineation drilling, mine planning and design, metallurgical testing, heap and process facility design, environmental baseline studies, Arizona and BLM mine permitting, and NRC source materials licensing. Pre-Production capital is estimated at 8 million $US for project development, engineering, and permitting. Construction of the site access road to BLM/Forest Service standards for a 14 foot running surface gavel access road is estimated at 1.04 million $US. Electrical Power costs are included in the processing facility CAPEX, Section 17.

Estimated CAPEX for open pit mine equipment and facilities is summarized on Table 5.1. Initial Replacement capital was based on 5% of rolling equipment, applicable to years 3 through 12. Open pit mine capital for the life of mine including replacement is estimated at 26 million $US.

The capital cost of the highwall miner is estimated at 15.7 million $US and relates only to the unit and initial materials inventory. When the highwall miner is deployed the rate of open pit production is declining and there is sufficient capacity in the mine haulage and support equipment to service the highwall miner.   This cost is incurred in years 4 and 5. Underground equipment and access development capital cost is estimated at 37.5 million $US. The cost is for a continuous mining system. Haulage from the mine portal to the heap would be accomplished with existing mine haulage and support equipment. This cost is incurred in years 6 and 7.

Total mine CAPEX, including capital replacement, for all mine operations and support is estimated at 80.2 million $US over the life of mine.

TABLE 16.2 – OPEN PIT MINE CAPEX

Mine Capital Costs	Units	Unit cost (x 1,000)	Initial Capital (x1,000)	Added Capital (x 1,000)	Total Capital (x 1,000)
330 LX Linkbelt	1	$136	$136		
16M Motor Grader-	1	$744	$744		
140 Grader	1	$304	$304		
D-8TDozer-	1	$658	$658		
D-9T	1	$775	$775		
A30D Volvo Artic Truck	4	$225	$902		
980 Wheel Loader-	1	$462	$462		
657G Scraper -	4	$1,400	$5,600	$5,600	
623H Scraper	1	$674	$674		
Water truck 3000 gal	1	$100	$100		
Water truck	1	$360	$360		
Sub Total Major Equipment			$10,714		$16,314
Mine Support vehicles					
Fuel/lube truck	1	$155	$155		
Mechanical service truck	1	$112	$112		
Rubber tire backhoe Cat 414e	1	$60	$60		
Pickup trucks, 4WD, ¾-ton	8	$29	$232		
Shop equipment	1	$400	$400		
Subtotal			$959		
Facilities					
Shop/Warehouse	1	$453	$453		
Office	1	$340	$340		
Lab Trailer	1	$50	$50		
XRF	3	$50	$150		
2 water storage tanks, 500,000 gal	2	$210	$420		
Dry room	1	$173	$173		
			$ 1,585		
Total Capital			$ 13,258		
Contingency 15%			$ 1,989		
Total Open Pit			**$ 15,247**		**$20,847**

16.9 Mine OPEX

Mine OPEX is comprised of labor, salaried professional and hourly workers, equipment operating costs and consumables. Operating cost for open pit mining equipment is based on owning and operating the mine equipment on a year round basis for the life of mine, including the reclamation phase of the project. The number and schedule for the equipment is based on cycle times for the major equipment and is adequate for the conceptual mine plan volumes and configuration.

The primary stripping fleet of (8) 637 scrapers is capable of excavating and hauling between 14 and 18 million cy of overburden per year. The production profile, Table 16.3, requires excavation of approximately 11 to 18 million cy/year with an average of just over 14 million cy per year. Annual operating costs for primary stripping are estimated at an annual cost of 10.6 million $US which is a weighted cost of $14.59 per ton of mined material.

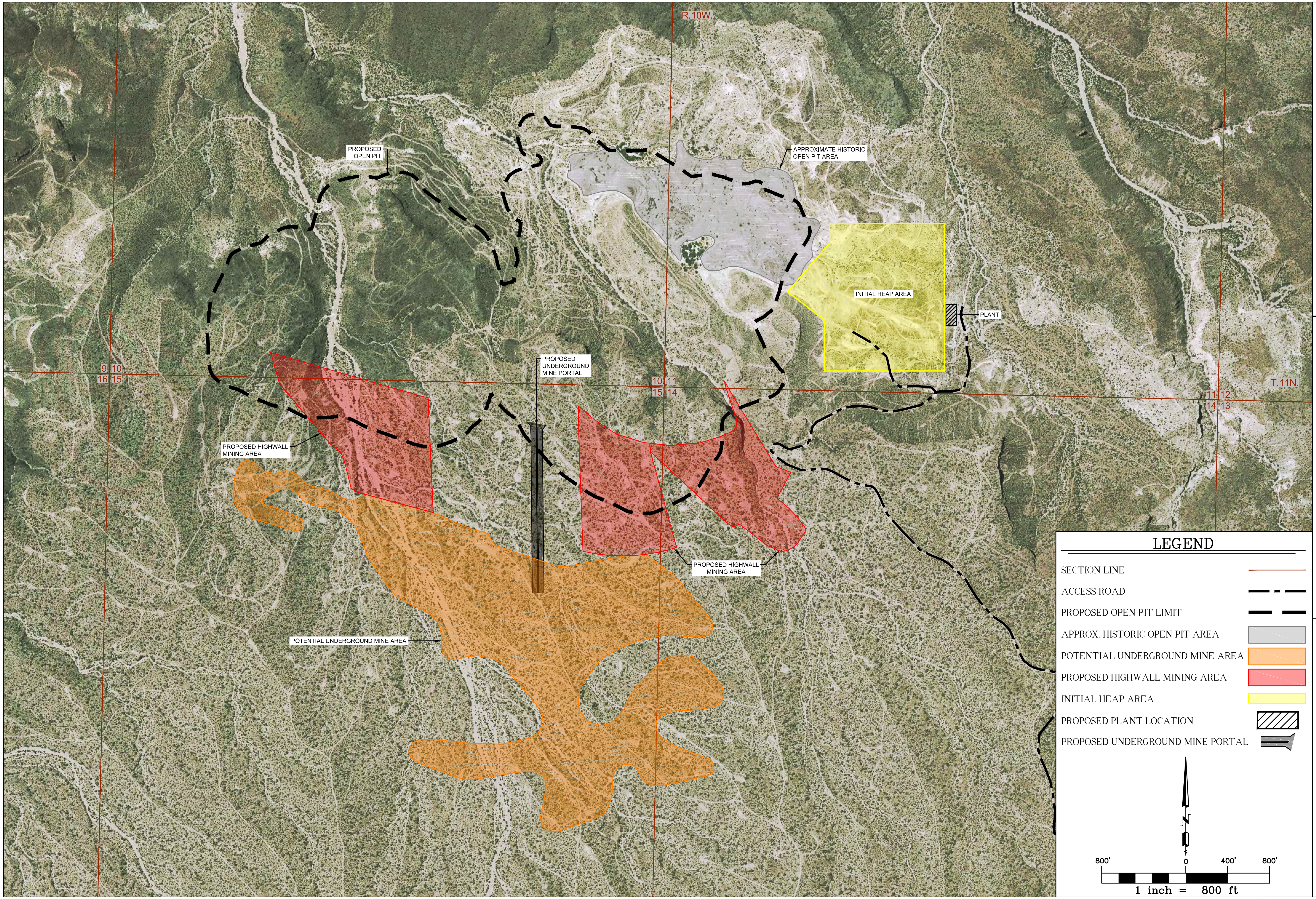
The open pit mining fleet consisting of an excavator, backup loader, and 4 articulated mine haulage trucks. The mining fleet will, mine and haul mineralized material to the stockpile located near the heap leach facility and excavate and place interburden within the pits. Average mine tonnage is less than 800,000 tons per year. At a 1 to 1 waste ratio the mining fleet would be required to produce 1,600,000 tons per year. The crew is capable of producing approximately 2 million tons per year with an average haul distance of 1 mile. Annual operating costs for the mining crew are estimated at 2.2 million $US or approximately $3 per ton of mined material.

Salaried staff includes 17 fulltime employees (refer to Section 16.12) at an estimated annual cost of 1.7 million $US or approximately $2 per ton of mined material.

Specific mine reclamation requirements are not well defined given the level of study of this PEA but were estimated based on the re-handling of 25% of the primary stripping volume for back fill and re-grading. Mine reclamation costs were based on completion of the work by the primary stripping fleet over a two year period. An additional allowance of 20 million $US was included in the PEA for decommissioning and reclamation of the heap leach and processing facility. The estimated costs per ton of mined material for the mine and heap/processing facility reclamation are $1.72 and $1.63 per ton, respectively.

16.10 Mine Recoveries and Dilution

In the absence of detailed mine planning, assessment of mine recoveries and mine dilution is difficult. For the PEA, the run-of-mine grade includes dilution of the mineralized intercepts to minimum mining thicknesses. The inclusion of an additional 10% dilution was evaluated in the sensitivity analysis as part of the financial evaluation. With respect to mine recoveries, a 95% recovery factor was applied to the open pit, a 75% factor to the highwall miner, and a 60% factor to the room and pillar underground mine. No secondary pillar extraction was considered for the underground mine. Future studies could evaluate various options for increasing highwall and/or underground mine recovery which could include retreat mining of pillars, with or without partial backfill of mined voids.



UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA

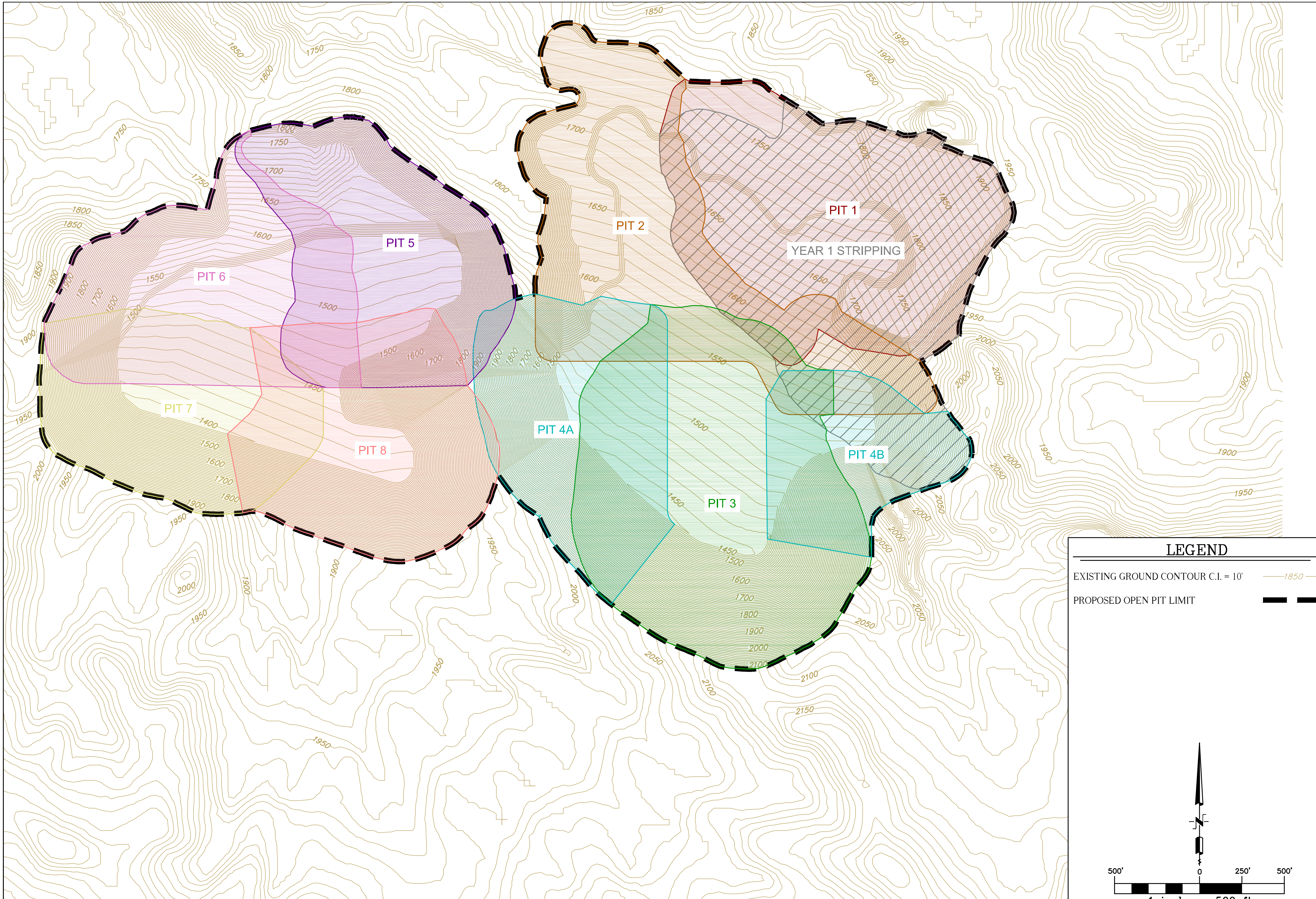
PROJECT LAYOUT

NO.	REVISION DATE:	DATE BY	ISSUED FOR

LAST PLOT DATE: 5/27/14
CAD FILENAME: USERV/UEC/ANDERSON/CAD/DWG/REPORT FIGURES/YEAR 1 STRIPPING.DWG

BRS ENGINEERING

SCALE: 1" = 500'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE 5/27/14
FIGURE NO. 16-1
REV.



BRS ENGINEERING

NO. | REVISION DATE: | DATE | BY | ISSUED FOR

LAST PLOT DATE: 6/2/14

CAD FILENAME: USERV/UEC/ANDERSON/CAD/DWG/REPORT FIGURES/SEQUENCE.DWG

UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA

PIT SEQUENCING

FIGURE NO. 16-2

REV.

SCALE: 1" = 500'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE 2/2/14

16.11 PERSONNEL REQUIREMENTS

Table 16.3 summarizes the project staff and mine operations personnel. Personnel directly associated with the heap leach and processing facility are discussed in Section 17.

TABLE 16.3 – PROJECT STAFF AND MINE PERSONNEL

General Project Manager	1
Mine Manager OP	1
Mine Leadman OP	1
Maintenance Leadman	1
Warehouse/clerk	1
Safety/personnel Manager	1
Environmental Engineer	1
Chief Mine Engineer	1
Chief Mine Geologist	1
RSO	1
Technician	1
Surveyor	1
Technicians	3
Secretary/Clerk	1
Accountant	1
Staff Total	17
Equipment Operators	44
Mechanics	8
Grade Control	6
Total Labor	58
Grand Total Personnel	75

16.12 MINE SEQUENCE AND PRODUCTION PROFILE

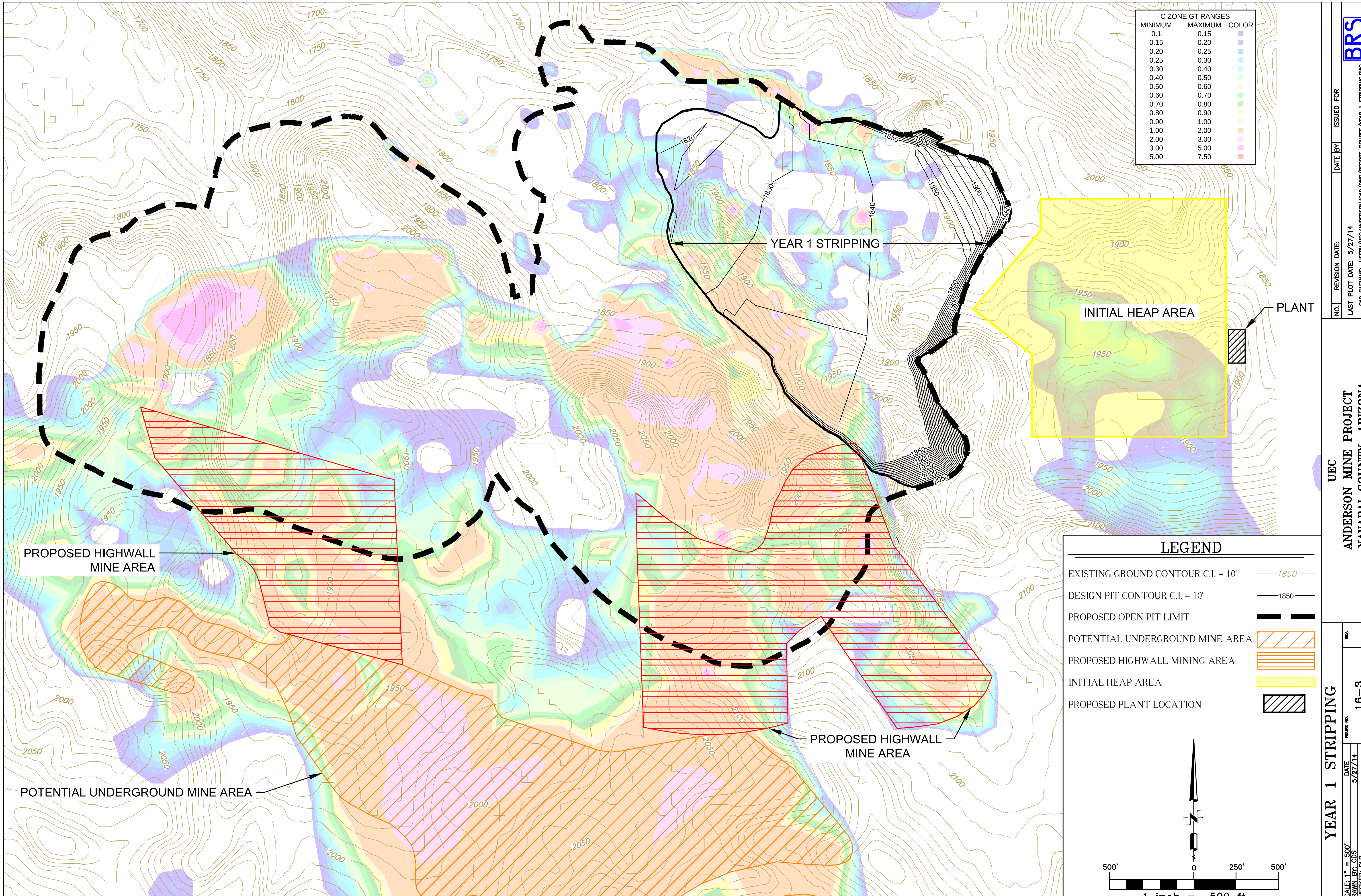
Table 16.4 summarizes the mine sequence and production profile. Figures 16-3 through 16.11 show the mining sequence for the open pit, highwall mining, and underground mining.

Pre-stripping of Pit 1 begins during the initial year of construction prior to production in year 1. The overburden removed during the pre-production period would be utilized for construction of the heap and facility sub-base and mineralization in the A horizon would be mined. The remainder of the stripping of Pit 1 would proceed in year one as would mining of the B and C horizons. Open pit stripping and mining would then follow in sequence.

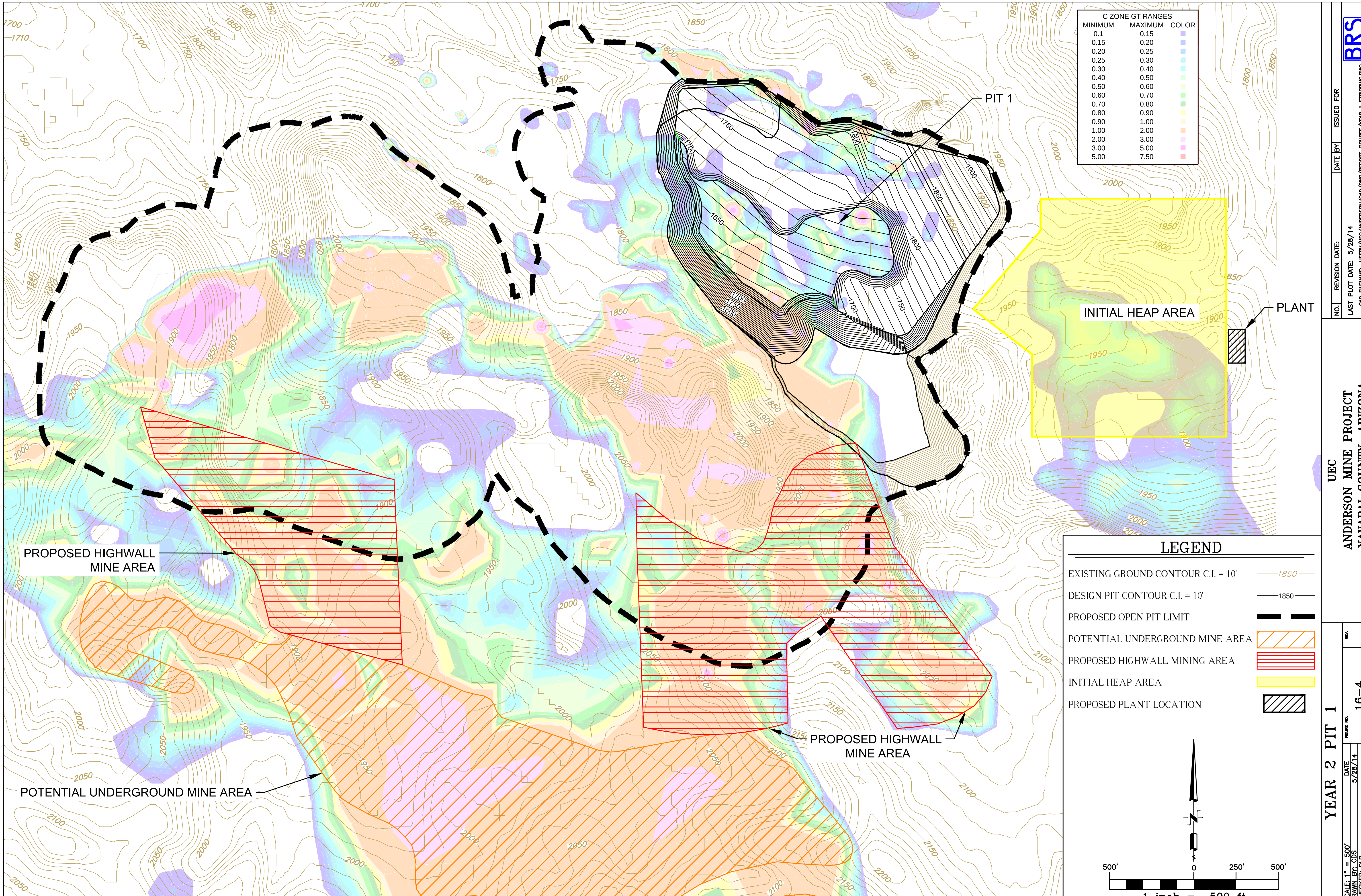
Upon completion of Pit 3 in year 4, highwall mining could begin in the eastern portion of the pit and sweeping to the central portion over a period of four years. The production profile shows a hiatus of highwall mining in year 8 until access for highwall mining is established in Pit 8 in year 9. Highwall mining in the west pit area would then continue for approximately three years. Underground mining would begin following the completion of Pit 4 in year 6 and would proceed through the final year of mining in year 14.

TABLE 16.4 – PRODUCTION PROFILE (UNITS X 1,000)

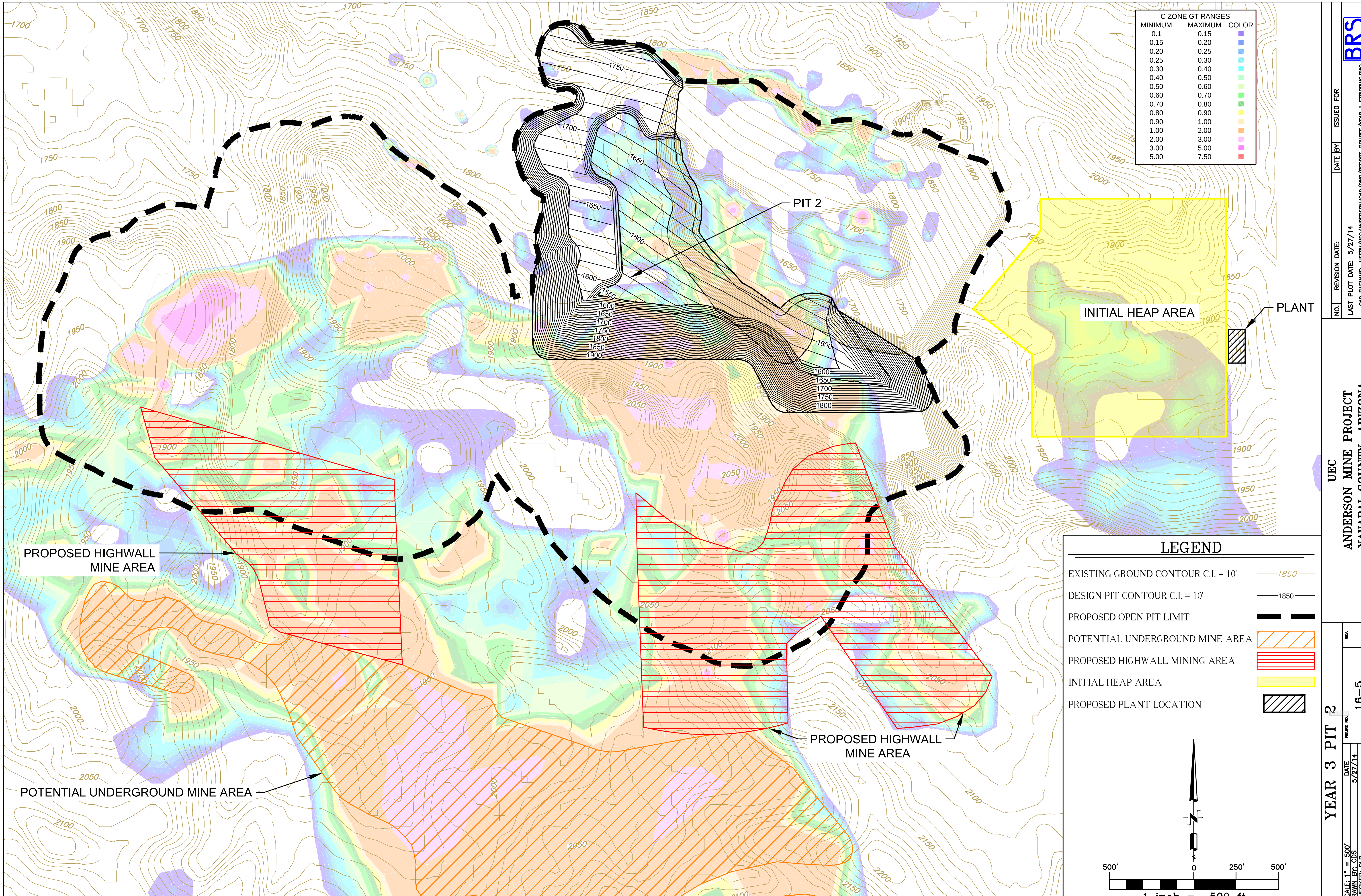
Production Year	**0**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**TOTAL**
Open Pit	**Pit 1**	**pit 1**	**pit 2**	**pit 3**	**pit 3**	**pit 4**	**pit 5**	**pit6**	**pit7**	**pit8**						
Total Pit Cubic Yards	5,512	7,627	10,895	16,000	17,820	14,309	13,729	13,002	13,197	14,068	0	0				126,158
Reclamation											15,000	15,000				30,000
Tons Mined	112	1,029	933	700	606	641	622	691	911	1,039						7,284
Plant feed, % U3O8	0.095	0.065	0.076	0.072	0.073	0.087	0.063	0.077	0.055	0.048						0.067
Pounds Contained U3O8	211	1,340	1,411	1,006	887	1,120	784	1,067	1,004	994						9,825
Highwall Miner					**East**	**East**	**East**	**East**		**West**	**West**	**West**	**West**			
Tons Mined					282	282	188	188		169	169	169	169			1,615
Plant feed, % U3O8					0.086	0.086	0.086	0.086		0.070	0.070	0.070	0.070			0.079
Pounds Contained U3O8					482	482	321	321		238	238	238	238			2,558
Underground																
Tons Mined							200	400	400	400	400	400	400	400	401	3,401
Plant feed, % U3O8							0.088	0.088	0.088	0.088	0.088	0.088	0.088	0.088	0.088	0.088
Pounds Contained U3O8							351	702	702	702	702	702	702	702	704	5,970
Tons Mined	112	1,029	933	700	888	922	1,010	1,279	1,311	1,608	569	569	569	400	401	12,300
Plant feed, % U3O8	0.095	0.065	0.076	0.072	0.077	0.087	0.072	0.082	0.065	0.060	0.083	0.083	0.083	0.088	0.088	
Pounds contained U3O8	211	1,340	1,411	1,006	1,369	1,602	1,456	2,091	1,706	1,934	940	940	940	702	704	18,353
Tons Stockpiled	0	179	212	12	-1	22	32	311	621	1,230	799	368	337	137	0	
grade	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Pounds stockpiled	0	233	320	17	-1	38	46	508	809	1,479	1,320	608	557	240	0	
Tons Processed	112	850	900	900	900	900	1,000	1,000	1,000	1,000	1,000	1,000	600	600	538	12,300
Plant feed, % U3O8	0.095	0.065	0.074	0.073	0.077	0.087	0.072	0.081	0.070	0.063	0.055	0.083	0.083	0.085	0.088	0.075
Pounds contained U3O8	211	1,107	1,324	1,309	1,387	1,564	1,448	1,629	1,405	1,264	1,099	1,652	991	1,018	945	18,353
Pounds recovered	190	996	1,191	1,178	1,248	1,408	1,303	1,466	1,265	1,138	989	1,487	892	917	850	16,328



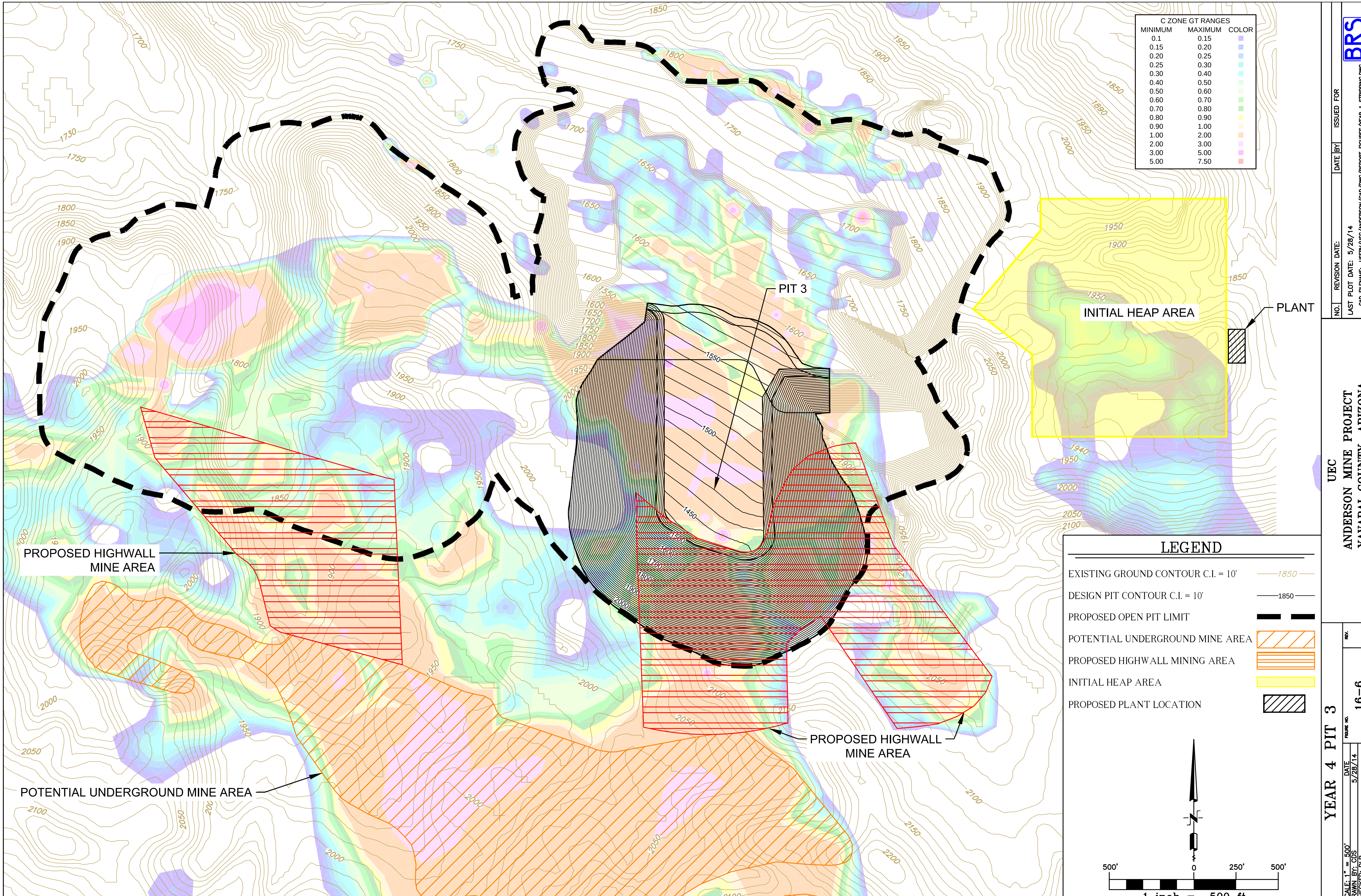
C ZONE GT RANGES
MINIMUM MAXIMUM COLOR
0.1 0.15
0.15 0.20
0.20 0.25
0.25 0.30
0.30 0.40
0.40 0.50
0.50 0.60
0.60 0.70
0.70 0.80
0.80 0.90
0.90 1.00
1.00 2.00
2.00 3.00
3.00 5.00
5.00 7.50
YEAR 1 STRIPPING
INITIAL HEAP AREA
PLANT
PROPOSED HIGHWALL MINE AREA
PROPOSED HIGHWALL MINE AREA
POTENTIAL UNDERGROUND MINE AREA
LEGEND
EXISTING GROUND CONTOUR C.I. = 10'
DESIGN PIT CONTOUR C.I. = 10'
PROPOSED OPEN PIT LIMIT
POTENTIAL UNDERGROUND MINE AREA
PROPOSED HIGHWALL MINING AREA
INITIAL HEAP AREA
PROPOSED PLANT LOCATION
500' 0 250' 500'
1 inch = 500 ft
UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA
YEAR 1 STRIPPING
16-3
SCALE: 1" = 500'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE 5/27/14
LAST PLOT DATE: 5/27/14

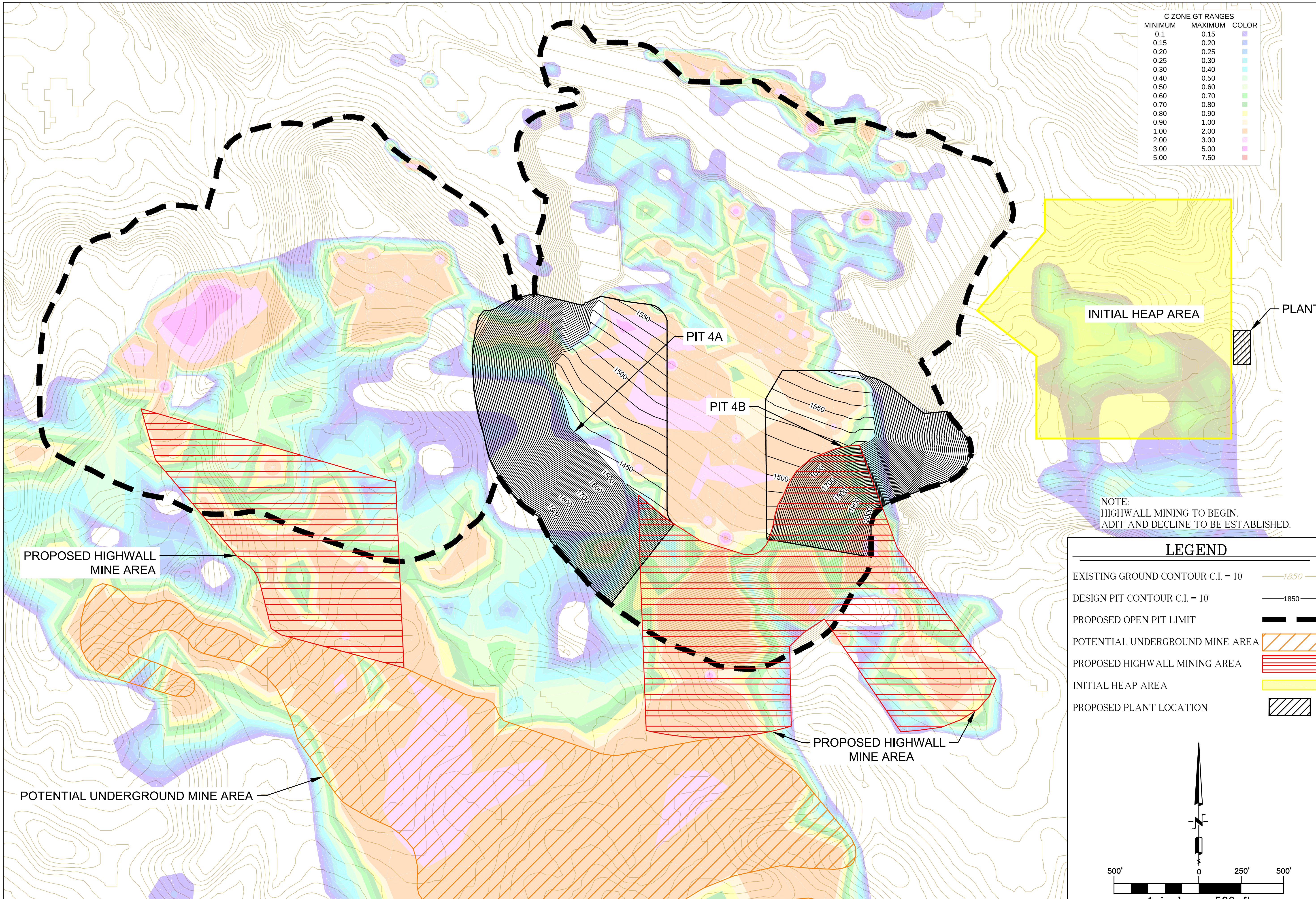


C ZONE GT RANGES
MINIMUM MAXIMUM COLOR
0.1 0.15
0.15 0.20
0.20 0.25
0.25 0.30
0.30 0.40
0.40 0.50
0.50 0.60
0.60 0.70
0.70 0.80
0.80 0.90
0.90 1.00
1.00 2.00
2.00 3.00
3.00 5.00
5.00 7.50
PIT 1
INITIAL HEAP AREA
PLANT
PROPOSED HIGHWALL MINE AREA
POTENTIAL UNDERGROUND MINE AREA
PROPOSED HIGHWALL MINE AREA
LEGEND
EXISTING GROUND CONTOUR C.I. = 10' 1850
DESIGN PIT CONTOUR C.I. = 10' 1850
PROPOSED OPEN PIT LIMIT
POTENTIAL UNDERGROUND MINE AREA
PROPOSED HIGHWALL MINING AREA
INITIAL HEAP AREA
PROPOSED PLANT LOCATION
500' 0 250' 500'
1 inch = 500 ft
BRS ENGINEERING
NO. REVISION DATE: DATE BY ISSUED FOR
LAST PLOT DATE: 5/28/14
CAD FILENAME: USERV/UEC/ANDERSON/CAD/DWG/REPORT FIGURES/YEAR 1 STRIPPING.DWG
UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA
YEAR 2 PIT 1
FIGURE NO. 16-4
SCALE: 1" = 500'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE 5/28/14



C ZONE GT RANGES
MINIMUM MAXIMUM COLOR
0.1 0.15
0.15 0.20
0.20 0.25
0.25 0.30
0.30 0.40
0.40 0.50
0.50 0.60
0.60 0.70
0.70 0.80
0.80 0.90
0.90 1.00
1.00 2.00
2.00 3.00
3.00 5.00
5.00 7.50
PIT 2
INITIAL HEAP AREA
PLANT
PROPOSED HIGHWALL MINE AREA
PROPOSED HIGHWALL MINE AREA
POTENTIAL UNDERGROUND MINE AREA
LEGEND
EXISTING GROUND CONTOUR C.I. = 10'
DESIGN PIT CONTOUR C.I. = 10'
PROPOSED OPEN PIT LIMIT
POTENTIAL UNDERGROUND MINE AREA
PROPOSED HIGHWALL MINING AREA
INITIAL HEAP AREA
PROPOSED PLANT LOCATION
500' 0 250' 500'
1 inch = 500 ft
UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA
YEAR 3 PIT 2
16-5
SCALE: 1" = 500'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE 5/27/14
LAST PLOT DATE: 5/27/14
BRS ENGINEERING


C ZONE GT RANGES
MINIMUM MAXIMUM COLOR
0.1 0.15
0.15 0.20
0.20 0.25
0.25 0.30
0.30 0.40
0.40 0.50
0.50 0.60
0.60 0.70
0.70 0.80
0.80 0.90
0.90 1.00
1.00 2.00
2.00 3.00
3.00 5.00
5.00 7.50
PIT 3
INITIAL HEAP AREA
PLANT
PROPOSED HIGHWALL MINE AREA
PROPOSED HIGHWALL MINE AREA
POTENTIAL UNDERGROUND MINE AREA
LEGEND
EXISTING GROUND CONTOUR C.I. = 10'
DESIGN PIT CONTOUR C.I. = 10'
PROPOSED OPEN PIT LIMIT
POTENTIAL UNDERGROUND MINE AREA
PROPOSED HIGHWALL MINING AREA
INITIAL HEAP AREA
PROPOSED PLANT LOCATION
500' 0 250' 500'
1 inch = 500 ft
BRS ENGINEERING
NO. REVISION DATE: DATE BY ISSUED FOR
LAST PLOT DATE: 5/28/14
CAD FILENAME: USERV/UEC/ANDERSON/CAD/DWG/REPORT FIGURES/YEAR 1 STRIPPING.DWG
UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA
YEAR 4 PIT 3
FIGURE NO. 16-6
SCALE: 1" = 500'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE 5/28/14



NO. | REVISION DATE: | DATE | BY | ISSUED FOR

LAST PLOT DATE: 5/27/14
CAD FILENAME: USERV/UEC/ANDERSON/CAD/DWG/REPORT FIGURES/YEAR 5 PIT 4.DWG

BRS ENGINEERING

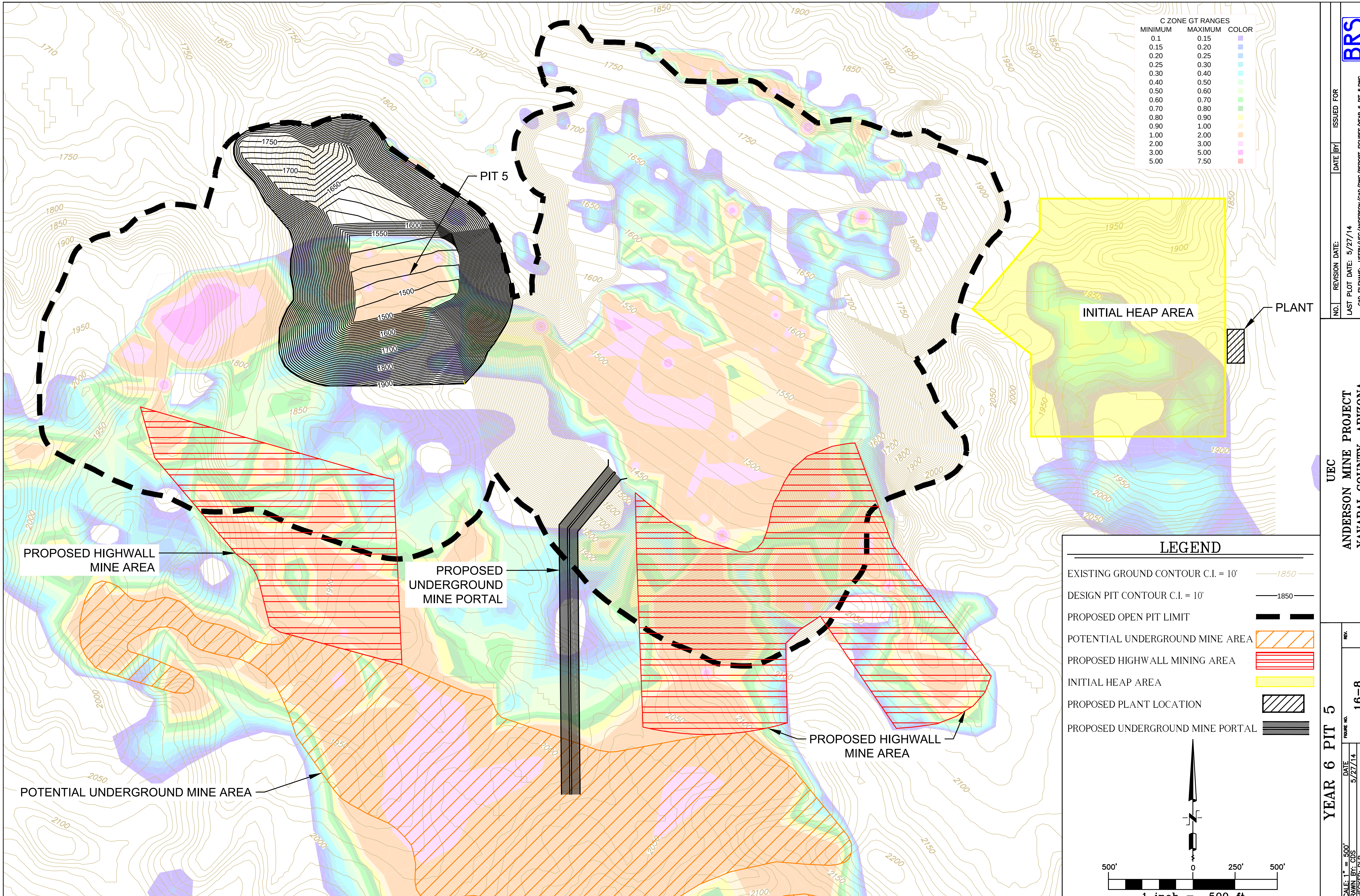
UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA

YEAR 5 PIT 4

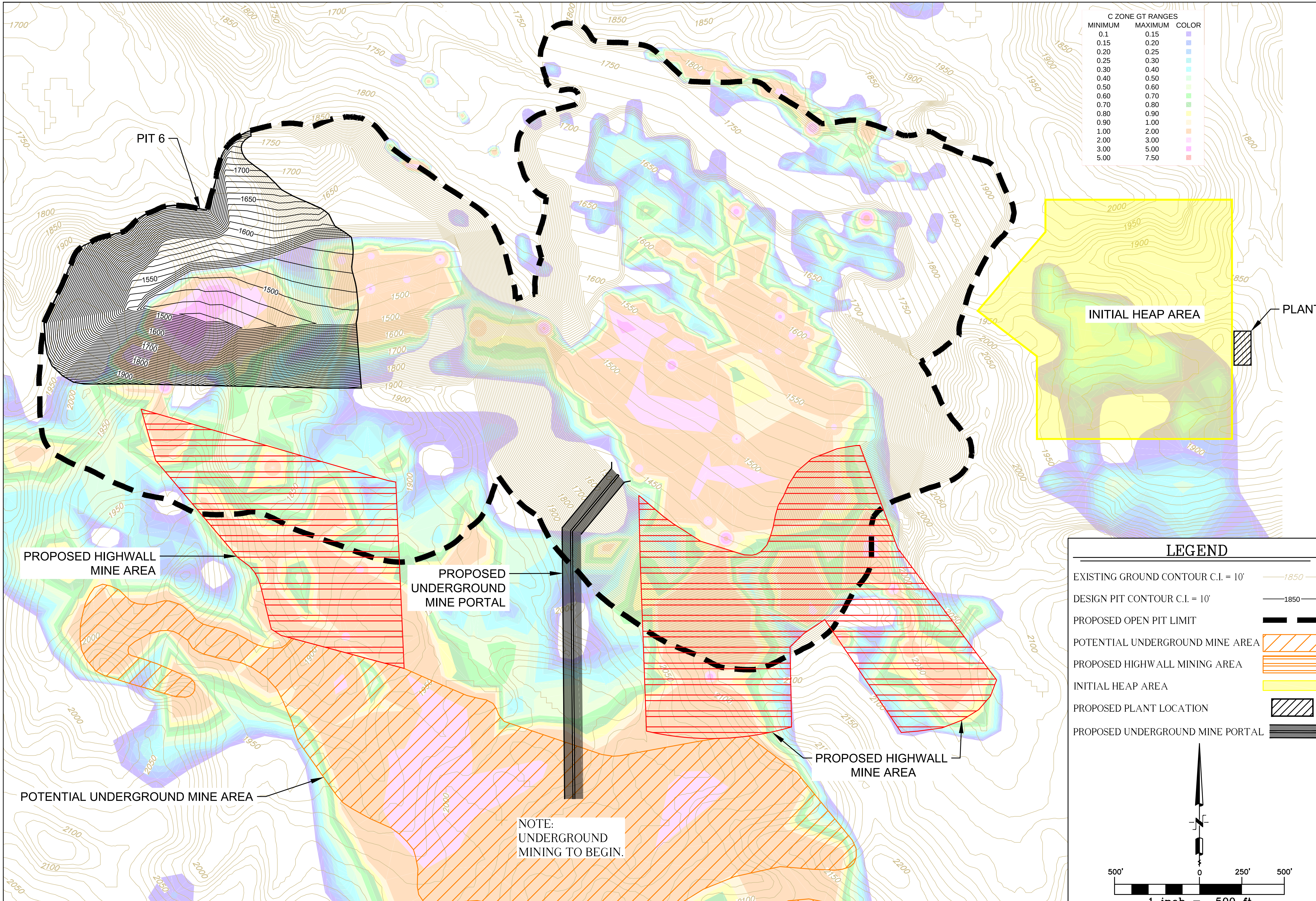
FIGURE NO. 16-7

REV.

SCALE: 1" = 500'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE 5/27/14



C ZONE GT RANGES
MINIMUM MAXIMUM COLOR
0.1 0.15
0.15 0.20
0.20 0.25
0.25 0.30
0.30 0.40
0.40 0.50
0.50 0.60
0.60 0.70
0.70 0.80
0.80 0.90
0.90 1.00
1.00 2.00
2.00 3.00
3.00 5.00
5.00 7.50
PIT 5
INITIAL HEAP AREA
PLANT
PROPOSED HIGHWALL MINE AREA
PROPOSED UNDERGROUND MINE PORTAL
PROPOSED HIGHWALL MINE AREA
POTENTIAL UNDERGROUND MINE AREA
LEGEND
EXISTING GROUND CONTOUR C.I. = 10' 1850
DESIGN PIT CONTOUR C.I. = 10' 1850
PROPOSED OPEN PIT LIMIT
POTENTIAL UNDERGROUND MINE AREA
PROPOSED HIGHWALL MINING AREA
INITIAL HEAP AREA
PROPOSED PLANT LOCATION
PROPOSED UNDERGROUND MINE PORTAL
500' 0 250' 500'
1 inch = 500 ft
BRS ENGINEERING
ISSUED FOR
NO. REVISION DATE: DATE BY
LAST PLOT DATE: 5/27/14
CAD FILENAME: USERV/UEC/ANDERSON/CAD/DWG/REPORT FIGURES/YEAR 6 PIT 5.DWG
UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA
YEAR 6 PIT 5
FIGURE NO. 16-8
REV.
SCALE: 1" = 500'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE 5/27/14



LEGEND

EXISTING GROUND CONTOUR C.I. = 10'

DESIGN PIT CONTOUR C.I. = 10'

PROPOSED OPEN PIT LIMIT

POTENTIAL UNDERGROUND MINE AREA

PROPOSED HIGHWALL MINING AREA

INITIAL HEAP AREA

PROPOSED PLANT LOCATION

PROPOSED UNDERGROUND MINE PORTAL

500' 0 250' 500'

1 inch = 500 ft

NO. | REVISION DATE: | DATE | BY | ISSUED FOR

LAST PLOT DATE: 5/27/14

CAD FILENAME: USERV/UEC/ANDERSON/CAD/DWG/REPORT FIGURES/YEAR 1 STRIPPING.DWG

BRS ENGINEERING

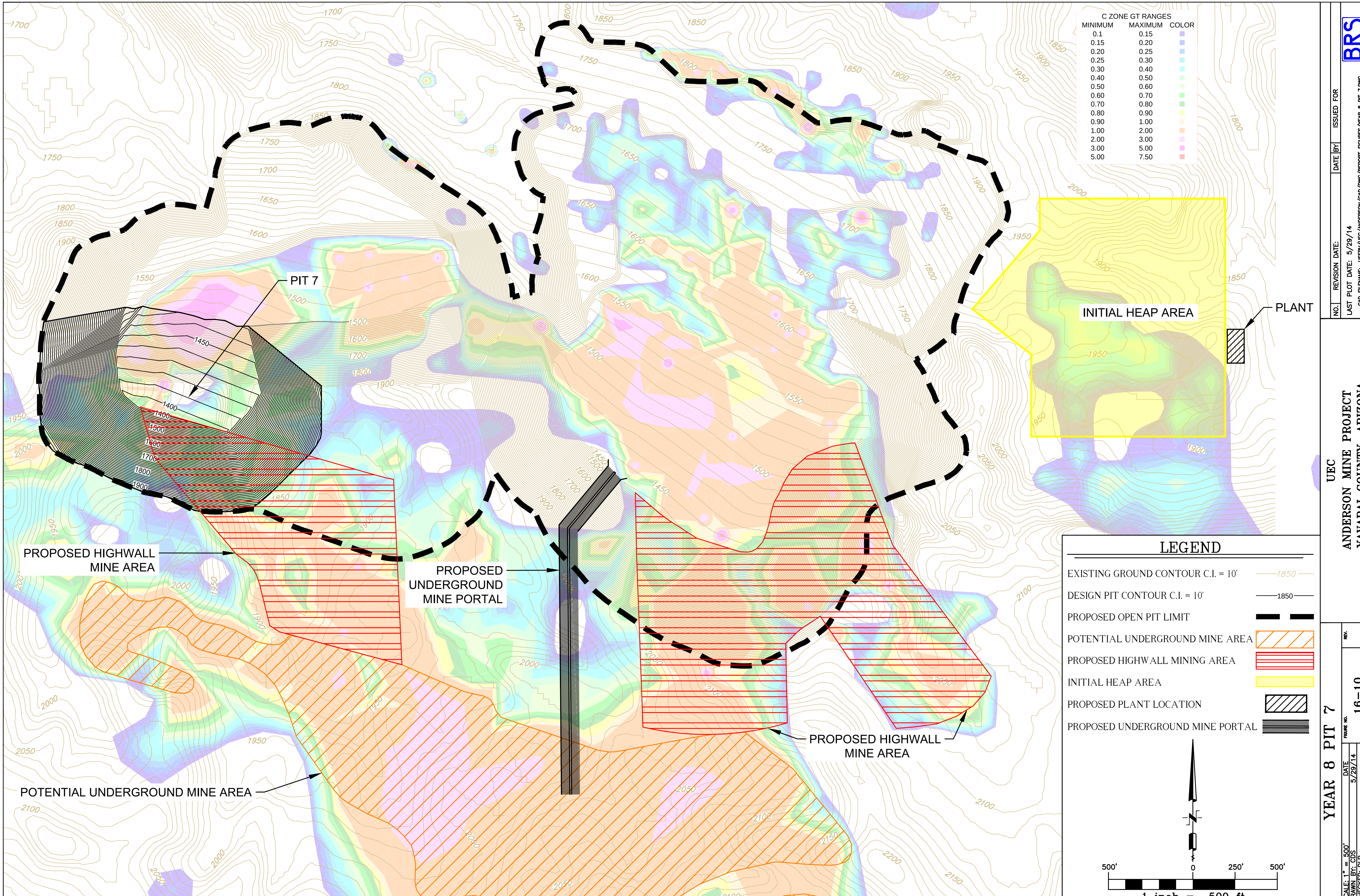
UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA

YEAR 7 PIT 6

FIGURE NO. 16-9

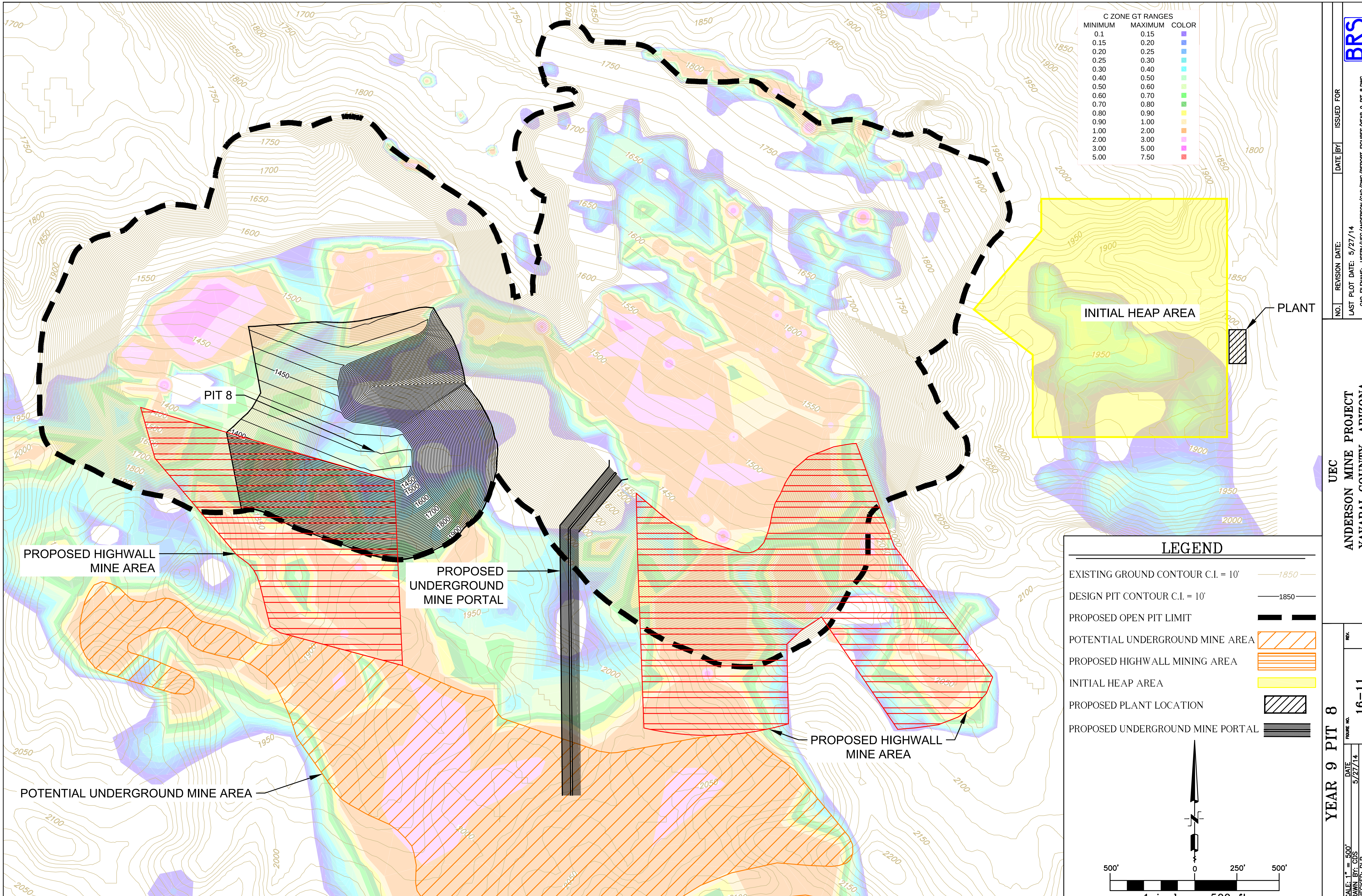
REV.

SCALE: 1" = 500'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE 5/27/14



C ZONE GT RANGES

MINIMUM	MAXIMUM	COLOR
0.1	0.15	
0.15	0.20	
0.20	0.25	
0.25	0.30	
0.30	0.40	
0.40	0.50	
0.50	0.60	
0.60	0.70	
0.70	0.80	
0.80	0.90	
0.90	1.00	
1.00	2.00	
2.00	3.00	
3.00	5.00	
5.00	7.50	


C ZONE GT RANGES
MINIMUM MAXIMUM COLOR
0.1 0.15
0.15 0.20
0.20 0.25
0.25 0.30
0.30 0.40
0.40 0.50
0.50 0.60
0.60 0.70
0.70 0.80
0.80 0.90
0.90 1.00
1.00 2.00
2.00 3.00
3.00 5.00
5.00 7.50
PIT 8
INITIAL HEAP AREA
PLANT
PROPOSED HIGHWALL MINE AREA
PROPOSED UNDERGROUND MINE PORTAL
POTENTIAL UNDERGROUND MINE AREA
PROPOSED HIGHWALL MINE AREA
LEGEND
EXISTING GROUND CONTOUR C.I. = 10'
DESIGN PIT CONTOUR C.I. = 10'
PROPOSED OPEN PIT LIMIT
POTENTIAL UNDERGROUND MINE AREA
PROPOSED HIGHWALL MINING AREA
INITIAL HEAP AREA
PROPOSED PLANT LOCATION
PROPOSED UNDERGROUND MINE PORTAL
500' 0 250' 500'
1 inch = 500 ft
UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA
YEAR 9 PIT 8
16-11
SCALE: 1" = 500'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE 5/27/14
LAST PLOT DATE: 5/27/14
CAD FILENAME: USERV/UEC/ANDERSON/CAD/DWG/REPORT FIGURES/YEAR 9 PIT 8.DWG
BRS ENGINEERING

17 RECOVERY METHODS

17.1 HEAP LEACH SUMMARY

Heap leach recovery for a variety of metals notably copper and precious metals is common practice in the US. Heap leach for uranium recovery has been practiced in the past in the US and around the world and is being considered for several projects in the US.

The planned uranium recovery method at the Anderson Project is conventional heap leaching which includes: the mobilization of uranium into solution from the mined material stacked on the heap pad via acid leaching, delivery of uranium rich solutions to a recovery plant (mill), and concentration of the uranium via Ion Exchange (IX).

Uranium recovery at the Anderson Project as depicted on Figure 17-1 will include the following processes:

- stacking of mined material on the heap leach pad;
- application of leach solution;
- collection of pregnant leach solution (PLS);
- filtering of sand and fines from PLS;
- IX to extract uranium from solution and load it on resin;
- Shipment of the loaded resin to EFR's White Mesa Mill for stripping, precipitation, washing, drying, packaging, storage, and loading as yellowcake.

The uranium recovery or "milling" process equipment will be housed in a single building within the proposed mill boundary. Loaded resin will be produced on-site. Yellowcake processing, including precipitation, washing, drying, packaging, storage, and loading, will be completed off-site. Reagent storage and distribution systems will be located within or next to the process buildings.

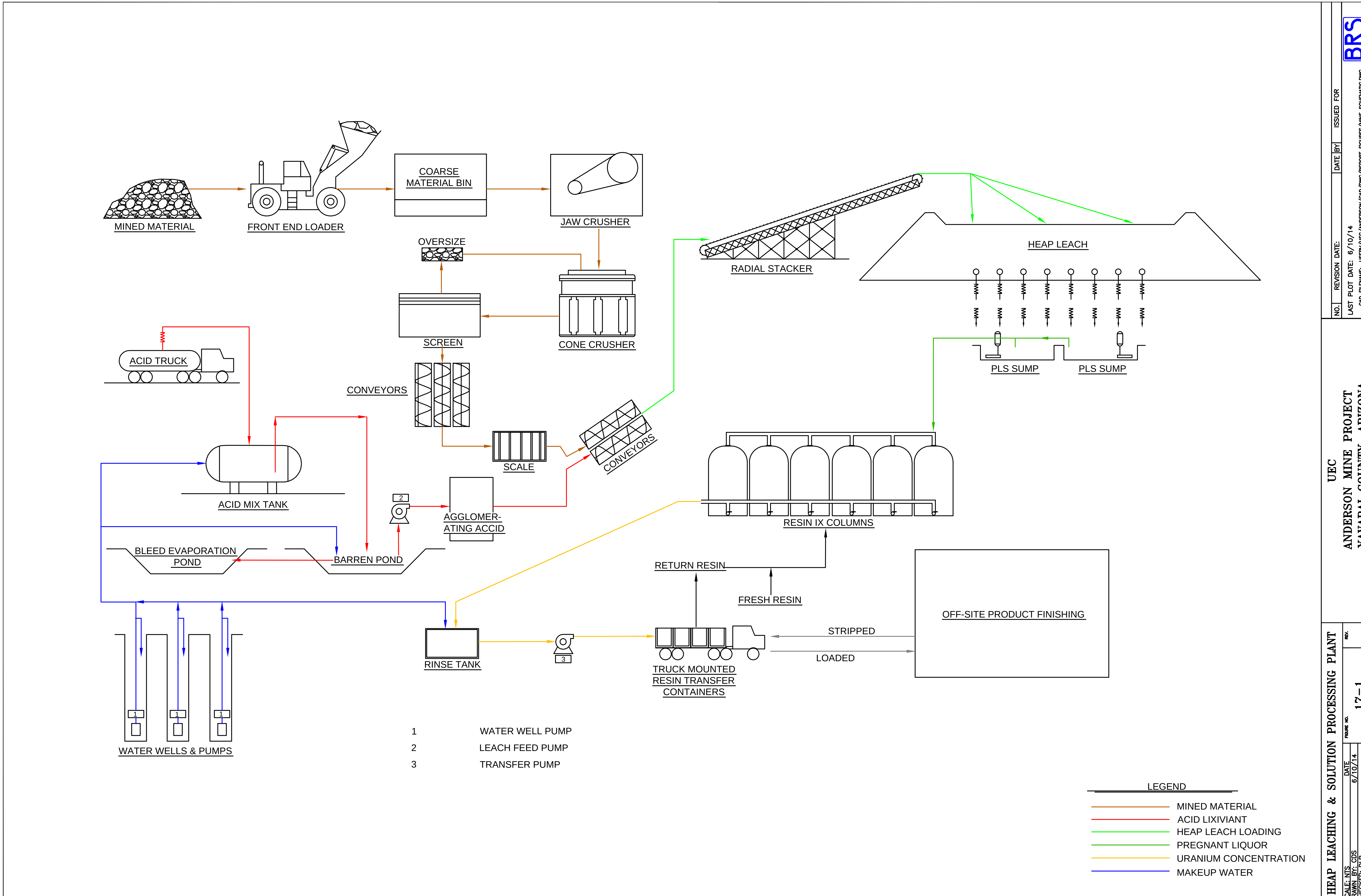
Processing ('milling') begins as run-of-mine product is crushed and then stacked on the double lined heap leach pad using covered belt conveyors and a covered radial arm stacking (RAS) belt . The stacked mined material is leveled with low ground pressure equipment forming a "lift". A protective layer of gravel is place on top of the lift to mitigate fugitive dust and transport of radio-particulates from the heap. A drip irrigation system using conventional plastic piping is then installed on top of the completed lift, and the heap is ready for the application of leach solutions.

The general flow of solutions and uranium within the heap and recovery plant is as follows:

- The process begins with the pumping of the refortified leach solution from the Barren Pond to the top of the heap where it is applied using drip emitters.
- The leach solution consists of water; an oxidizing agent (sodium chlorate, to convert the uranium to the soluble hexavalent state), and a complexing agent (sulfuric acid) to complex and solubilize the uranium.

- The result of the heap leach process is a pregnant leach solution (PLS) containing a mixture of uranyltrisulfate (UTS) and uranyldisulfate (UDS). The process solution percolates through the stacked mined material via gravity drainage and is intercepted by the heap leach pad liner system and then gathered into collection pipes, which drain by gravity into the collection pond and is recirculated until the concentration meets the criteria for PLS.
- The PLS is then pumped from the collection pond into the IX plant where the PLS is filtered to remove suspended particulates, and the uranium is loaded on the resin.
- The resulting uranium-depleted solution, called barren leach solution, flows by gravity from the IX plant to the barren pond. This barren solution is refortified with additional acid and oxidant and additional make-up water. It is then pumped back to the heap in a continuous cycle.
- Resin is shipped from the plant to EFR's White Mesa Mill for final processing to yellowcake.

The production profile, Table 16.4, presumes that construction of the initial heap pad and mineral processing facility will begin early in the initial year of construction such that leaching of the initial mined material form the A horizon in Pit 1 can be processed in the 4th quarter of that year.



MINED MATERIAL
FRONT END LOADER
COARSE MATERIAL BIN
JAW CRUSHER
OVERSIZE
CONE CRUSHER
SCREEN
CONVEYORS
SCALE
CONVEYORS
RADIAL STACKER
HEAP LEACH
PLS SUMP
PLS SUMP
ACID TRUCK
ACID MIX TANK
2
AGGLOMER-ATING ACCID
RESIN IX COLUMNS
BLEED EVAPORATION POND
BARREN POND
RETURN RESIN
FRESH RESIN
OFF-SITE PRODUCT FINISHING
STRIPPED
LOADED
RINSE TANK
3
TRUCK MOUNTED RESIN TRANSFER CONTAINERS
1
1
1
WATER WELLS & PUMPS
1 WATER WELL PUMP
2 LEACH FEED PUMP
3 TRANSFER PUMP
LEGEND
MINED MATERIAL
ACID LIXIVIANT
HEAP LEACH LOADING
PREGNANT LIQUOR
URANIUM CONCENTRATION
MAKEUP WATER
NO.
REVISION DATE:
DATE
BY
ISSUED FOR
LAST PLOT DATE: 6/10/14
CAD FILENAME: USERV/UEC/ANDERSON/CAD/DWG/REPORT FIGURES/MINE SCHEMATIC.DWG
BRS ENGINEERING
UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA
HEAP LEACHING & SOLUTION PROCESSING PLANT
REV.
FIGURE NO.
17-1
DATE
6/10/14
SCALE: NTS
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB

17.2 HEAP LEACH COST ESTIMATION

Heap leach costs, both capital and operating, were estimated from recent and past experience, publically available data from other economic studies, and estimated by escalating historic cost estimates for current material prices and are of a conceptual nature. No site specific designs have been completed. The heap leach cost is based on an acid heap leach recovery system and is supported by recent metallurgical testing as discussed in Section 13 (Refer also to Appendix A). Underlying assumptions used for estimating costs include:

- Material placed on the leach heap will be crushed and screened to 80% passing 1½-inches. This will require a jaw crusher, a cone crusher in closed circuit with a 2-deck vibrating screen, moveable conveyors, and a radial stacker.

- Pad area will be 65 acres at a 20-foot heap depth. Future testing is required for final design height as well as the applicability of agglomeration to maintain permeability.

- Average acid consumption will be 50 pounds of 93% H2SO4 per ton of leached material.

- Mineralization grades of 0.075% U3O8 and 0.050% V2O5.

- 90 percent uranium extraction is estimated. Note that the preliminary laboratory tests suggest that future optimization studies could yield average extractions around 94 percent. Therefore, good operating practices, including effective agglomeration and stacking, could avoid significant reduction in solution contact and make 90 percent uranium extraction a reality. Soluble losses, including those to the bleed evaporation pond, can be minimized by recovering and treating sludge at some point in project life.

- Solution treatment will be in ion exchange columns filled to 40 volume percent with Dowex 21K strong-base anionic resin at a delivered price of $850/ft3. The assumed loading capacity of the resin is 5 pounds of anions per cubic foot. The vanadium concentration in the PLS will be too low to prevent uranium adsorption on the resin, but it will compete with uranium. At the assumed grades and recoveries one cubic foot of loaded resin will contain 3.9 lb U3O8 and 1.1 lb V2O5.

Further testing related to vanadium in recommended. Solvent extraction (“SX”) for uranium using Alamine 336 and kerosene as the organic phase allows partitioning of vanadium into the raffinate which then be extracted with a different solvent to produce a stream from which vanadium can be precipitated. However, using IX and remotely eluting the resin presents a challenge. During the 1950s and 1960s, quite a few plants treated combined U/V in PLS by various methods. For instance pentavalent V can be reduced to tetravalent V which will not adsorb onto a strong-base anionic resin. Also, a number of different elution techniques were used to separate U and V.

For the PEA, Energy Fuels (USA) Inc. (EF) evaluated three options for processing resin delivered from the Anderson Project to the White Mesa mill which the operate. These options and associated CAPEX and OEX are provided in Section 17.8 which follows. The PEA takes no economic credit for production of vanadium as a co-product with the uranium.

17.3 HEAP LEACH RECOVERIES

Heap leach recoveries of 90% were used in the PEA based on the average grades of mineralized material anticipated and recent metallurgical testing as previously discussed. There is a risk that recoveries will vary.

17.4 HEAP LEACH CAPEX

Each heap leach pad will occupy approximately 65 acres and will be loaded with a radial stacker conveyor to a height of approximately 20 feet. The entire pad will be lined with a primary 80 mil HDPE liner installed on a compacted 24 inch base. The lower halves of the cells will have an additional 80 mil HDPE liner installed over geotextile netting for leak detection and will serve as the PLS ponds to minimize evaporative losses. The initial capital for the heap leach pad and all appurtenances, with a 20% contingency is 7.7 million $US. For the PEA it was assumed that a total of four such lined pads would be required over the life of the mine. It is however possible to minimize capital by either off-loading and reusing the heap pad and/or stacking an additional layer of mineralized material on top of the original heap separated by an internal HDPE liner.

17.5 PROCESSING FACILITY CAPEX

CAPEX for the mineral processing facility is summarized in Table 17.1 and includes the plant equipment and appurtenances, the first-fill of resin, indirect costs, working capital and a 20% contingency. Replacement capital is included in the PEA at a rate of 5% per year, for years 3 through 12, based on the initial CAPEX for buildings and equipment.

TABLE 17.1: PROCESSING FACILITY CAPEX ($US X 1,000)

Equipment	Size	Units	Unit Cost	Factor	Cost
Water wells, 200 GPM	300' depth	2	$ 18.8	1	$ 38
Submersible pump, SS	250 gpm@100' TDH	2	$ 21.0	1.5	$ 63
Process water tank, steel, 24'D x 21'H	70,000 gal	1	$ 64.0	1.25	$ 80
Coarse ore bin with 12"x12" bar grizzly	100 tons	1	$ 68.0	1.2	$ 82
Jaw crusher plant, vibrating feeder and conveyors	24" x 36" jaw	1	$ 477.9	1.1	$ 526
Cone crusher with vibrating screen package	36" standard	1	$ 708.8	1.1	$ 780
Transfer conveyor package with 3 folding units	2x60' and 1x70'	1	$ 115.0	1.1	$ 127
Transfer conveyor package with 2 folding units	2x60'	1	$ 72.5	1.1	$ 80
Radial stacker system, portable	30" x 100'	1	$ 91.5	1.1	$ 101
Belt scale		1	$ 15.0	2	$ 30
Crusher control van, complete with MCC		1	$ 190.0	1.1	$ 209
Lime silo, 2,500 CF	10' D x 35' H	1	$ 46.0	1.5	$ 69
Lime feeder	36" x 5'	1	$ 15.8	2.5	$ 40
PLS sump pumps w/recirculating loops	250 gpm@100' TDH	2	$ 14.0	3	$ 84
IX resin column, code welded, dished heads	10' D x 20'	6	$ 125.0	2	$ 1,500
Transfer pump	200 gpm@ 50' TDH	6	$ 18.0	3	$ 324
Area sump pump	150 gpm @ 15' TDH	8	$ 12.0	3	$ 288
Resin sizing screen, SWECO	48-inch	2	$ 21.0	2	$ 84
Resin rinse tank	6'D x 8'	1	$ 12.0	1.5	$ 18
Resin transport containers, SS316, 300 cf	8'D x 8'H	8	$ 25.0	1	$ 200
Barren pond, 2 acre x 3' deep, lined		1	$ 125.0	1	$ 125
Bleed evap cell, 4 acre x 3' deep, double-lined		1	$ 400.0	1	$ 400
Heap feed pump, VS, 200-600 GPM @ 100' TDH		2	$ 40.0	3	$ 240
Sulfuric acid mix tank w/mixer	3,000 gal	1	$ 5.3	2.5	$ 13
Reagent metering pumps	0-30 gph	2	$ 3.5	2	$ 14
Miscellaneous tools, sets		2	$ 2.0	1	$ 4
Safety supplies, kit		1	$ 1.5	1	$ 2
Shower and eye wash station		3	$ 1.9	3	$ 17
Sample prep equipment, lot		1	$ 47.0	2.5	$ 118
Dust collector/filter	4,800 cfm	1	$ 4.9	3	$ 15
Assay laboratory equipment, lot		1	$ 175.0	1.5	$ 263
Prep & assay trailer	8'x12'x50'	1	$ 35.0	1.3	$ 46
Office trailer, furnished	8'x8'x32'	1	$ 16.0	1.2	$ 19
Area lighting		1	$ 15.0	2	$ 30
Power line @ $121,000/mile		17	$ 121.0	1	$ 2,057
Substation		1	$ 145.0	2	$ 290
SUBTOTALS					**$ 8,371**
Engineering, PM, and purchasing @ 9%		1	$ -		$ 753
IX resin first fill @ $850/cu. ft. for Dowex 21K		3768	$ 0.9	1	$ 3,203
Owner's costs @ 5% of installed cost		1	$ -		$ 419
Working capital @ 60 day's operating expense		60	$ 35.2	1	$ 2,111
Freight for equipment		1	$ 95.0	1	$ 95
Contingency @ 20%					$ 2,990
CONSTRUCTED COST					**$17,942**

17.6 HEAP LEACH/PROCESSING PLANT OPEX

The operating costs for the heap leach are estimated on a per ton basis. The estimated cost includes a fixed labor costs and variable operating cost. The total estimated operating cost for the heap leach and mineral recovery facility is $10.63 per ton of mineralization processed based on an average daily tonnage leached of 2,500 tons and average daily production of 3,500 pounds U_3O_8 . Delivery of material to the heap is included in the mine costs, Section 16.

TABLE 17.2: HEAP LEACH/PROCESSING FACILITY OPEX

HEAP/PLANT OPEX	Units/Day	PRICE, $	$/Day
Electrical energy from grid, kWh	19992	0.06	$ 1,200
Sulfuric acid for leaching, lb @ 50 lb/ton consumption	125000	0.075	$ 9,375
Sodium chlorate, lb 2 lb/ton	5000	0.4	$ 2,000
Lime in pneumatic trucks, lb	28300	0.065	$ 1,840
DOWEX 21K resin, cubic feet	3	850	$ 2,550
Lubricants, gallons	25	12.5	$ 313
Diesel fuel for loader, gallons	384	3.5	$ 1,344
Maintenance and repair parts & supplies	1 lot		$ 1,000
Laboratory reagents & supplies	1 lot		$ 1,000
Total consumables			$ 25,553
Total Labor	1 Year	$ 1,552,000	$ 5,953
TOTAL DAILY EXPENSE			$ 26,573
Operating cost/ton of Material			$ 10.63
Operating cost per pound U3O8			$ 7.59

17.7 HEAP/ PLANT PERSONNEL

General project staffing is summarized in Section 16. Staff and hourly personnel exclusively assigned to the heap leach/plant operations are summarized in table 17.3. Labor cost for heap leach and processing facility personnel total $1,552,000 annually and are include in Table 17.2.

TABLE 17.3: HEAP LEACH/PROCESSING FACILITY PERSONNEL

Salaried:	
Operation supt./metallurgist	1
General foreman	1
Shift foreman	2
Assayer	1
Clerk	1
Total salaried personnel	6
Hourly:	
Crusher operator	3
Stacker operator	3
Pad operator	4
Pad helper	4
Loader operator	2
IX column operator	3
Sample preparation	1
Lab technician	1
Mechanic/welder	1
Electrician	1
General plant labor	4
Total hourly personnel	23
TOTAL PAYROLL	29

17.8 RESIN SHIPMENT AND FINAL PROCESSING COSTS

Energy Fuels (EF) was approached by UEC as an option for treating the loaded resin from the Anderson Project at the White Mesa mill in Blanding, Utah. EF was amenable with this approach and via email from Harold Roberts, Chief Operating Officer, dated 7 July, 2014 provided costs for three options, in the absence of specific testing, for treating the Anderson resin and producing yellowcake. These options are summarized as follows:

Three scenarios have been evaluated for processing of the Anderson loaded IX resin. No metallurgical test work was conducted for this evaluation. These conclusions are based on in-house experience with uranium ore solutions and the application of accepted IX processes.

Low Impurity Levels – The impurity levels are low enough to not impact product quality. The proposed process would be a chloride strip resulting in a solution that would be sent directly to ammonia precip along with the strip solution generated from the USX. Uranium Precipitation, Calcining and Packaging would be concurrent with normal production. The estimated processing cost would be approximately $2.60 per lb. U3O8. CAPEX estimated at $954,762.00.

Atypical or High Levels of Specific Impurities - These impurities would require processing the stripped uranium solution through the USX to reduce impurities such as vanadium, arsenic, zirconium, molybdenum, etc. to acceptable product levels. The process would be a sulfuric acid strip followed by concurrent processing with normal USX feed streams through USX, Precip, Calcining and Packaging. The estimated processing cost would be approximately $3.99 per lb. U3O8. CAPEX estimated at $954,762.00.

Impurities Rejectable by Peroxide Precipitation – This process would involve a Chloride Strip followed by Peroxide Precip followed by Calcining and Packaging. The chloride strip solution would be recycled within the IX Resin process. The estimated processing cost would be approximately $2.76 per lb. U3O8. CAPEX estimated at $1,975,735.00.

The resin stripping would be followed by a regeneration step using HCl in all three scenarios. The operating cost for this step is minimal and is not included in the processing cost estimates.

Scenarios #2 and #3 would require some metallurgical test work to develop and define the process.

For the base case PEA the author used the third case, Impurities Rejectable by Peroxide Precipitation. The other case were evaluated as part of the sensitivity analysis.

The costs for resin handling and final processing provided by EF are FOB the mill in Blanding, Utah. The one-way haulage distance from the Anderson Project to Blanding is approximately 450 miles. CAPEX for the plant provides for two resin hauling trucks. OPEX for the trucks is estimated at $100 per hour to include labor, fuel, consumables and repairs. Thus for a round trip the OPEX would be $1,800.00. As each truck is estimated to deliver 2,340 pounds of uranium the cost per pound is estimated at $1.30.

To these costs EF would charge either $1.25 or $1.75 per pound depending on whether or not the mill was processing mined material. For the PEA the author elected to use a median rate of $1.25 per pound based on the assumption that both the Anderson Mine and White Mesa mill would be operating under favorable market conditions.

In summary, the base case for the haulage and final processing of loaded resin at the White Mesa mill is: CAPEX $1,975,735 and OPEX $5.31 per pound.

18 PROJECT INFRASTRUCTURE

18.1 ACCESS

The Project is located in western Yavapai County approximately 75 miles northwest of Phoenix, Arizona. The Project is accessible via four-wheel drive on existing county roads. See Figure 18-1: Site Access and Utilities. The PEA includes provision for upgrading the access road to BLM/Forest Service standards for a 14 foot running surface, gravel surface roadway. The cost estimate for the PEA is not however based on an approved design.

18.2 POWER AND UTILITIES

Utility services including natural gas, electricity, and communications are located in Wickenburg, Arizona 20 miles from the eastern boundary of the Project. The PEA includes the installation of an overhead powerline to serve the project. Costs estimated in the PEA are based on recent cost data from similar projects an approved design or alignment. Natural gas service is not necessary for the project and communications would rely on cellular or satellite service.

18.3 PROCESS WATER

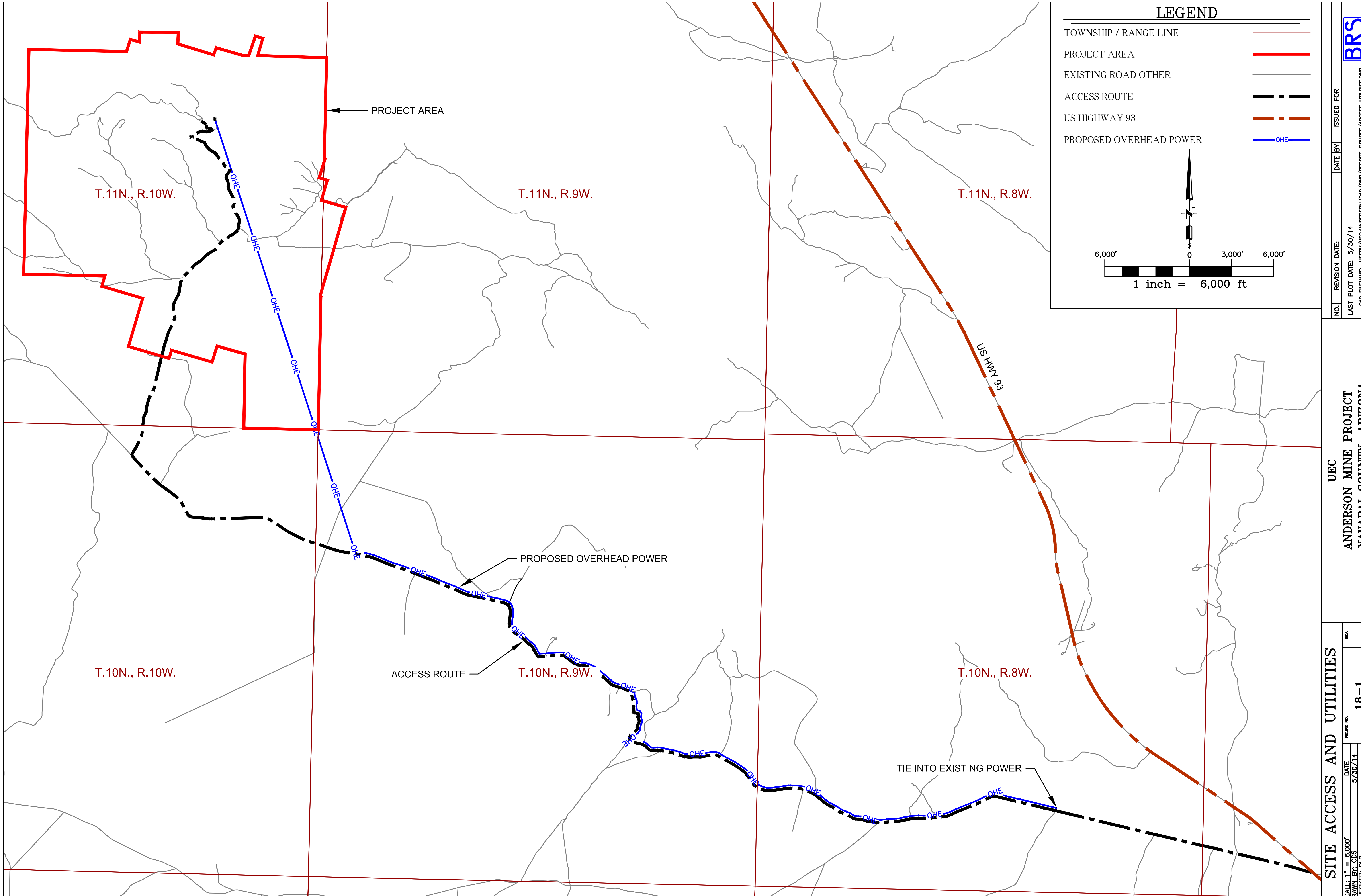
Water supply could be obtained by existing onsite wells. Water rights for both surface and ground water are administered by the Arizona State Engineer's Office and are subject to prior water rights. There are existing water wells on site, however, the PEA does include cost provisions for additional wells if needed.

18.4 MINE SUPPORT FACILITIES

Mine support facilities will consist of an office for all staff, mine shop, and warehouse, and a dry facility. The mineral processing facility will be a separate building and will house the appurtenant mineral processing equipment. The climate is moderate and freezing is not an issue so that some equipment would not need to be housed.

18.5 PUBLIC SAFETY AND FACILITY MAINTENANCE

Access to the site will be controlled where appropriate. The mine facility will be regulated by MSHA. Any persons wishing to enter the facility will be required to complete safety training as required by regulations and be equipped with appropriate Personal Protective Equipment depending on which areas they wish to enter. Access to the mineral processing facility and heap leach will be restricted in accordance with NRC regulations and license conditions.



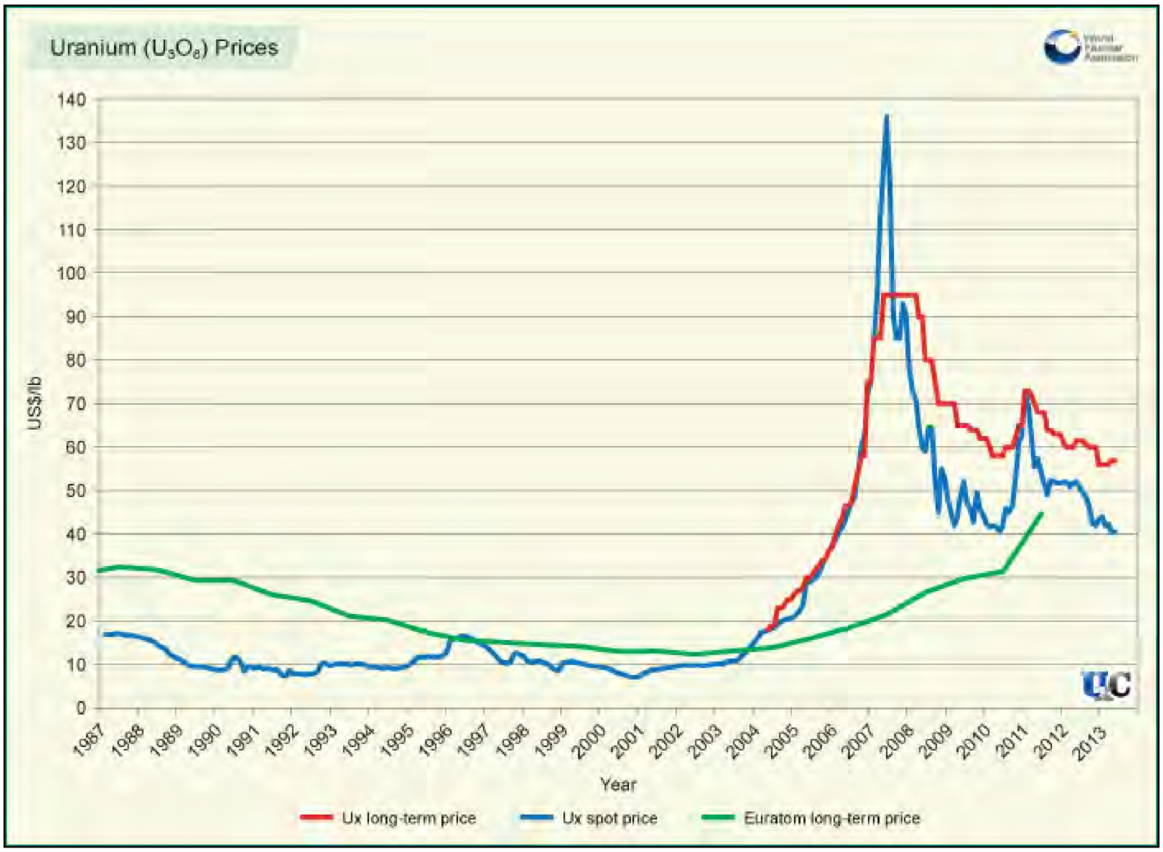
LEGEND
TOWNSHIP / RANGE LINE
PROJECT AREA
EXISTING ROAD OTHER
ACCESS ROUTE
US HIGHWAY 93
PROPOSED OVERHEAD POWER
OHE
6,000'
0
3,000'
6,000'
1 inch = 6,000 ft
PROJECT AREA
T.11N., R.10W.
T.11N., R.9W.
T.11N., R.8W.
T.10N., R.10W.
T.10N., R.9W.
T.10N., R.8W.
US HWY 93
PROPOSED OVERHEAD POWER
ACCESS ROUTE
TIE INTO EXISTING POWER
BRS ENGINEERING
ISSUED FOR
NO.
REVISION DATE:
DATE
BY
LAST PLOT DATE: 5/30/14
UEC
ANDERSON MINE PROJECT
YAVAPAI COUNTY, ARIZONA
SITE ACCESS AND UTILITIES
SCALE: 1" = 6,000'
DRAWN BY: CDS
CHECKED: DLB
APPROVED: DLB
DATE
5/30/14
FIGURE NO.
18-1
REV.

19 MARKET STUDIES AND CONTRACTS

Uranium does not trade on the open market and many of the private sales contracts are not publically disclosed. Monthly long term industry average uranium prices based on the month-end prices are published by Ux Consulting, LLC, and Trade Tech.

As shown on Figure 19.1, the current spot price is less than the long term contract price. However, during periods when the spot price rises, such as the peaks in 2007 and 2011, the spot price equals or exceeds the long term price. Spot prices apply only to marginal trading and usually represent less than 20% of supply (World Nuclear Association, 2013). Thus, the author recommends use of long term uranium pricing in the PEA.

FIGURE 19.1: URANIUM PRICE HISTORY



(http://world-nuclear.org/info/Nuclear-Fuel-Cycle/Uranium-Resources/Uranium-Markets/)

Tables 19.1 and 19.2, show the monthly long-term and spot uranium prices, respectively. (Trade Tech, 2013).

TABLE 19.1: LONG TERM URANIUM PRICE*

Long Term Uranium Price*

	2009	2010	2011	2012	2013
Jan	$ 69.00	$ 60.00	$ 70.00	$ 61.00	$ 57.00
Feb	$ 69.00	$ 60.00	$ 70.00	$ 60.00	$ 57.00
Mar	$ 69.00	$ 60.00	$ 68.00	$ 60.00	$ 57.00
Apr	$ 69.00	$ 60.00	$ 68.00	$ 61.00	$ 57.00
May	$ 65.00	$ 60.00	$ 68.00	$ 61.00	$ 57.00
Jun	$ 65.00	$ 60.00	$ 68.00	$ 61.00	$ 57.00
Jul	$ 65.00	$ 60.00	$ 68.00	$ 61.00	$ 54.00
Aug	$ 65.00	$ 60.00	$ 65.00	$ 60.00	$ 53.00
Sep	$ 65.00	$ 62.00	$ 63.00	$ 61.00	$ 51.00
Oct	$ 65.00	$ 62.00	$ 63.00	$ 59.00	$ 50.00
Nov	$ 60.00	$ 65.00	$ 62.00	$ 59.00	$ 50.00
Dec	$ 60.00	$ 67.00	$ 61.00	$ 57.00	$ 50.00
Average	$ 65.50	$ 61.33	$ 66.17	$ 60.08	$ 54.23
Average long-term price 2009 through 2013 -					$ 61.46

TABLE 19.2: SPOT URANIUM PRICE*

Spot Uranium Price*

	2009	2010	2011	2012	2013
Jan	$ 47.00	$ 42.25	$ 72.25	$ 52.25	$ 43.75
Feb	$ 44.00	$ 40.50	$ 69.50	$ 52.00	$ 42.00
Mar	$ 42.00	$ 41.75	$ 58.50	$ 51.10	$ 42.25
Apr	$ 45.00	$ 41.75	$ 55.00	$ 51.50	$ 40.50
May	$ 49.00	$ 40.75	$ 56.50	$ 51.25	$ 40.40
Jun	$ 51.00	$ 41.75	$ 51.50	$ 50.75	$ 39.55
Jul	$ 47.00	$ 45.25	$ 52.00	$ 49.50	$ 35.00
Aug	$ 46.00	$ 45.50	$ 49.25	$ 48.00	$ 34.00
Sep	$ 43.00	$ 46.75	$ 52.00	$ 46.50	$ 35.00
Oct	$ 46.50	$ 52.00	$ 51.75	$ 41.00	$ 34.25
Nov	$ 45.25	$ 60.25	$ 51.50	$ 42.50	$ 36.08
Dec	$ 44.50	$ 62.00	$ 52.00	$ 43.25	$ 37.50
Average	$ 45.85	$ 46.71	$ 55.98	$ 48.30	$ 38.07
Average spot price 2009 through 2013 -					$ 47.37

* As quoted by Trade Tech, 2013

http://www.uranium.info/

Thus, in a 5-year look-back from 2009 to the present, average uranium prices have been $47.37 per pound for spot delivery to $61.46 per pound for long-term delivery. On June 25, 2014, the average spot price was approximately $28.00 and the average long-term price was $45.00

(Cameco, 2014). Near- to mid-term uncertainty has created recent weakness in uranium markets. The shutdown of reactors in Japan, building inventories, and a general lack of demand has been largely to blame for this near-term price weakness. However, longer-term market fundamentals in the uranium sector remain strong. Nations around the World, led by China, are building new nuclear reactors. Yet, current weakness in uranium prices is leading to new uranium projects being deferred or canceled. Indeed, the World Nuclear Association reports that there are now 70 nuclear reactors under construction around the World right now. In addition, Japan has signaled that it will restart many of their reactors in the coming years, with several potentially restarting in 2014. As a result, though predicting spot- and long-term prices is speculative, many analysts expect rising spot- and long-term prices in the coming years (Ux Consulting, 2013). While the effect of supply and demand and other factors on long term uranium price is difficult to predict, Table 19.3 shows the price forecasts from several analysts.

TABLE 19.3: URANIUM LONG-TERM PRICE FORECASTS

URANIUM ANALYST FORECASTS

Bank	2013	2014	2015	2016	2017
Dundee Capital	65.00	65.00	65.00		
Haywood Securities	60.00	70.00	75.00	75.00	70.00
JPM Australia	43.00	58.00	70.00	90.00	
Scotiabank	46.00	52.00			
Macquarie	45.00	52.50	63.00	70.00	70.00
RBC Capital Markets	45.00	65.00	75.00	75.00	80.00
UBS	50.00	55.00	65.00	65.00	65.00
Morgan Stanley	46.75	60.00	63.00	64.00	69.50
Raymond James	40.00	52.00	70.00	70.00	70.00
BMO	49.00	52.00	60.00	70.00	70.00
Canaccord Genuity	44.00	50.00	55.00	60.00	70.00
Cantor Fitzgerald	55.00	70.00			
TD Securities	41.30	48.00	55.00	70.00	70.00
Mean	**48.47**	**57.65**	**65.09**	**70.90**	**70.50**

(Outsider Club, 2013)

The author recommends utilizing a uranium price of $65/lb. as a base case in the PEA representing the average price forecast for 2015. The reader is cautioned that any forecast of commodity price is a Forward-Looking. There can be no assurance that such Forward-Looking price forecasts will prove to be accurate and actual results and future events could differ materially from those anticipated in such statements.

UEC does not have any contracts in place for the sale of product produced from the Anderson Mine Project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

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 Bureau of Land Management. The disturbed area is a result of minor production via dozer cuts from surface mining done in the 1950s. No specific social or community related requirements, negotiations, and/or agreements are known to exist with local communities and/or agencies other than those discussed herein.

No outstanding environmental liabilities to UEC are known to the author.

20.1 REQUIRED PERMITS

The permitting and licensing requirements for this project, as discussed herein, are substantial as they are for any similar project in the US. To the author's knowledge there are no identified environmental conditions that would materially affect the development of the project.

All exploration and mining activities must comply with the National Environmental Policy Act (NEPA). Required environmental permits and licenses would include but may not be limited to:

- Mine Land Reclamation Plan; Arizona State Mine Inspector
- Exploration Permit; Arizona State Land Department
- Plan of Operations; Bureau of Land Management (Kingman Field Office)
- Source Material License; U. S. Nuclear Regulatory Commission
- Water Wells and Appropriations; Arizona Department of Water Resources
- Dams and Impoundments; Arizona Department of Water Resources
- Air Quality Control Permit; Arizona Department of Environmental Quality
- Water and Stormwater Discharge Permits; Arizona Department of Environmental Quality
- Hazardous Waste; Arizona Department of Environmental Quality and EPA
- Solid Waste; Arizona Department of Environmental Quality
- Mine Safety and Health; Arizona State Mine Inspector and MSHA
- County Zoning and Construction Permits

20.2 ENVIRONMENTAL BASELINE AND RELATED STUDIES

Environmental and related studies in support of the required permits, Section 20.1, would include but may not be limited to:

- Land Use; Regulations typically require that current land use be assessed and the potential impacts of the proposed operations to such uses be evaluated. Also, the final reclamation plan must be sustainable and compatible with land use.
- Cultural Resource Surveys; Cultural resource surveys and paleontological surveys are required by BLM for mining activities including an assessment of impacts and mitigation measures, if required. The Arizona State Historical Preservation Officer (SHPO) will also need to approve actions related to cultural resources. Tribal consultation will also be required under 36 CFR Part 800, Section 106.
- Meteorology and Air Monitoring; Background air quality conditions need to be measured for at least one year prior to operations and potential air quality impacts address. Tier 4 diesel equipment will likely be required as the emission standards are being phased in over a period which began in 2008 and continues to 2015.
- Geology; The environmental geologic setting of the project area will need to be defined with respect to potential pathways and geologic hazards including faulting, landslides, and flooding.
- Hydrology; Surface and ground water regimes will need to be defined with respect to quality and quantity and potential environmental pathways. At least one year of monitoring data is required to establish background.
- Soils and Vegetation; Soils and vegetation surveys are required to assess background conditions and are applied in the development of a sustainable reclamation plan. Vegetation surveys will also need to address any potential Threatened and Endangered (T&E) plant species and critical habitat such as wetlands.
- Wildlife; Wildlife surveys are required with respect to general wildlife populations but most also address any T&E species which may be present and any critical habit.
- Radiology; Background radiologic surveys are recommended whether or not specifically required prior to operations to define both the NORM, Naturally Occurring Radioactive Materials, and TENORM, Technically Enhanced Naturally Occurring Radioactive Materials, to separate impacts of current or planned operations from past operations and elevated natural background. An assessment of radiological impacts and exposures will be required by EPA for Radon emissions from the operations. If on-site mineral recovery is contemplated extensive assessment of radiological conditions and potential impacts and exposures will be required.

20.3 RECLAMATION AND DECOMMISSIONING COST PROVISIONS

Arizona mine regulations do not require backfill and regrading to approximate original contours and do allow remnant highwalls so long as stability and protection of human health and the environment are adequately addressed. Heap leach recovery will require the isolation of all mill waste material including contaminated buildings and equipment to be disposed of in a lined disposal cell which is isolated from dispersion through all environmental pathways. The heap pads meet this criterion when covered with a radon cap and erosional protection layer. If properly sited the heap pads can be reclaimed in place along with any contaminated materials for the plant decommissioning.

The cost model allows for the re-handling of approximately 25% of the mine waste for both mine and heap backfill and cover; grading and reclamation of an estimated 400 acres of disturbance; and includes a lump sum of $20 million $US in consideration of the additional requirement for plant decommissioning and reclamation. Reclamation of the heap leach and mineral processing facility will be in accordance with USNRC source materials license conditions for the project.

20.4 SOCIAL OR COMMUNITY RELATED REQUIREMENTS AND PLANS

The authors are not aware of any specific current social or community related requirements or of any negotiations or agreements with local communities related to the project. As the project proceeds with the permitting process NEPA requires public notifications and hearings and consultations.

21 CAPITAL AND OPERATING COSTS

21.1 CAPEX SUMMARY

Project cost estimates are based on a conventional open pit mine operation with on-site heap leach extraction. It is anticipated that the project will not produce a final product but will operate as a satellite facility for shipping loaded resin to another facility. For the PEA it was assumed that resin would be shipped to the planned EFR's White Mesa.

All costs are estimated in constant 2014 US Dollars. Operating (OPEX) and Capital (CAPEX) costs reflect a full and complete operating cost going forward including all pre-production costs, permitting costs, mine costs, and complete reclamation and closure costs for of the mine. CAPEX does not include sunk costs or acquisition costs.

Mining and mineral recovery methods are described in Sections 16 and 17, respectively. The mine production profile is discussed in Section 16. Table 21.1 provides a summary of CAPEX.

TABLE 21.1: CAPITAL COST SUMMARY($ X 1,000)

Capital Expenditures:	Year -3	Year -2	Year -1	Year 0	Total**
Permitting	$ 500	$ 1,000	$ 2,000	$ 500	$ 4,000
Development	$ 1,000	$ 1,500	$ 1,500		$ 4,000
Access Road			$ 1,040		$ 1,040
Buildings, Shop, Support Equip				$ 2,544	$ 2,544
Open Pit Mine				$ 10,714	$ 16,314
Open Pit Mine contingency 15%				$ 1,989	$ 1,989
HW Miner Equipment w/ 25%					$ 15,625
UG Room & Pillar w/ 25%					$ 37,500
Processing Plant (Including White Mesa Costs)				$ 10,411	$ 10,347
Heap Pad/Equipment				$ 7,656	$ 28,056
IX Resin First Fill				$ 3,203	$ 3,203
Indirects (EPM, Freight, Owner Costs)				$ 1,419	$ 1,267
Working Capital				$ 2,111*	
Processing Plant Contingency 25%				$ 3,238	$ 2,990
Replacement Mine @5%					$ 5,150
Replacement Plant @5%					$ 5,173
TOTAL CAPITAL EXPENDITURES	$ 1,500	$ 2,500	$ 4,540	$ 43,784	$ 139,200***

*Working Capital is included in initial CAPEX then credited at end of project so total is zero.
**Total CAPEX includes pre-production, initial capital, capital added during operations, and capital replacement. Significant added capital includes, open pit mine equipment, highwall mining equipment, underground mining equipment, and additional heap leach pads.
***Rounded

21.2 OPEX SUMMARY

Project operating cost estimates are based on a conventional open pit mine with heap leach processing. Operating cost estimates were based upon vendor quotations, published mine costing data, and contractor quotations. Such estimates were generally provided for budgetary purposes and considered valid at the time the quotations were provided. In all cases, appropriate suppliers, manufacturers, tax authorities, smelters, and transportation companies should be consulted before substantial investments or commitments are made.

Operating costs were estimated for the following major items and are summarized on Table 21.2:

- Mine Operating Expenses
- Heap Leach and Processing Plant Expenses
- Reclamation and Closure
- Reclamation Bond
- Taxes and Royalties
- Transport of Resin to White Mesa
- Allocated Costs for Final Processing at White Mesa. However, the loaded resin could be shipped for final processing to other central processing facilities.

TABLE 21.2: OPERATING COST SUMMARY

Direct Costs:	$/ton	$/lb Contained	$/lb Recovered
Mining			
Pit Stripping	$ 14.04	$ 10.47	$ 11.64
Pit Mining	$ 2.88	$ 2.15	$ 2.39
HW Mining $/ton	$ 11.50	$ 7.26	$ 8.07
UG Mining $/ton	$ 34.12	$ 19.44	$ 21.59
Weighted Average Mining Costs	$ 20.81	$ 13.94	$ 15.49
Mineral Processing $/ton	$ 10.63	$ 7.12	$ 7.92
Project Staff	$ 2.08	$ 1.39	$ 1.55
Reclamation Mine	$ 1.72	$ 1.15	$ 1.28
Reclamation Mill/Tailings	$ 1.63	$ 1.09	$ 1.21
Resin Shipping and Processing $/lb	$ 7.13	$ 4.78	$ 5.31
Reclamation Bond	$ 0.49	$ 0.33	$ 0.36
Severance tax (2.5% on 50% of net)	$ 0.70	$ 0.47	$ 0.52
TOTAL Direct Costs	$ 45.18	$ 30.68	$ 33.65

22 ECONOMIC ANALYSIS

22.1 COST MODEL AND NPV ANALYSIS

The current The PEA shows a positive return on investment with an IRR ranging from 40% to 48% with uranium prices in the range of $60 to $65 per pound. Including Arizona and US Federal income tax the IRR ranges from 32% to 39% with uranium prices in the range of $60 to $65 per pound. Table 22.1 shows the NPV at various discount rates both before and after taxes.

TABLE 22.1 – NPV AND IRR

Before Income Tax

		@ $60/lb	@ $65/lb
NPV at 8% discount rate		$131.7 Million $US	$170.7 Million $US
NPV at 10% discount rate		$109.0 Million $US	$142.2 Million $US
NPV at 12% discount rate		$90.7 Million $US	$119.0 Million $US
IRR		53%	63%

After Arizona and Federal Income Tax

		@ $60/lb	@ $65/lb
NPV at 8% discount rate		$93.6 Million $US	$122.8 Million $US
NPV at 10% discount rate		$76.4 Million $US	$101.1 Million $US
NPV at 12% discount rate		$62.4 Million $US	$83.6 Million $US
IRR		42%	50%

22.2 SENSITIVITY

Sensitivity of the projected IRR and NPV with respect to key parameters other than price are summarized in Table 22.2. The sensitivity analysis shows that the project is not highly sensitive to minor changes in OPEX and/or CAPEX. A 10% change in either OPEX or CAPEX results in a variance in IRR in the range of 4 to 5%. NPV is more sensitive to changes in OPEX than CAPEX.

With respect to heap leach or mine recovery, the project is roughly twice as sensitive to variances in mine recovery as it is to variance in OPEX or CAPEX. A 10% change in either heap leach or mine recovery results in a variance in IRR of 7 to 11% with the project more sensitive to heap leach recovery than mine recovery.

TABLE 22.2 – SENSITIVITY ANALYSIS AT $65/LB

Parameter	Change in Base Case	Change in IRR Pre-Tax	Change in IRR Post-Tax	Change in NPV @10% Pre-Tax	Change in NPV @10% Post-Tax
Heap Recovery	10 %	13 %	11 %	$43.9 million	$32.9 million
Mine Recovery	10 %	10 %	8 %	$34.2 million	$25.6 million
CAPEX	10 %	6%	5%	$ 7.8 million	$ 7.1 million
OPEX	10 %	6%	5%	$21.6 million	$16.2 million

As three options were presented by EF for the treatment of resin at the White Mesa mill the sensitivity of the project to these variable cost was also considered.

TABLE 22.3 – RESIN PROCESSING COSTS AT WHITE MESA MILL

Before Income Tax

@ $65/lb		Base Case Peroxide Precipitation	Low Impurities	High Impurities
NPV at 8% discount rate		$170.7 Million $US	$171.1 Million $ US	$160.1 Million $US
NPV at 10% discount rate		$142.2 Million $US	$142.5 Million $ US	$133.2 Million $US
NPV at 12% discount rate		$119.0 Million $US	$119.4 Million $ US	$111.4 Million $US
IRR		63%	63%	61%

After Arizona and Federal Income Tax

@ $65/lb		Base Case Peroxide Precipitation	Low Impurities	High Impurities
NPV at 8% discount rate		$122.8 Million $US	$123.1 Million $ US	$114.9 Million $US
NPV at 10% discount rate		$101.1 Million $US	$101.5 Million $ US	$94.5 Million $US
NPV at 12% discount rate		$83.6 Million $US	$84.0 Million $ US	$78.0 Million $US
IRR		50%	50%	48%

22.3 PAYBACK PERIOD

Capital investment was assumed to begin four years prior to start up to include such items as exploratory drilling, environmental baseline studies, engineering and design related studies, and permitting and licensing. Once in operations the project has a positive cumulative cash flow after year 2 in constant dollars. Refer to Table 22.4, Cash Flow.

Table 22.4 - CASH FLOW (Units X 1,000)

Production Year	-3	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	Production Totals
Open Pit																				
Total Pit Waste Cubic Yards				5,512	7,627	10,895	16,000	17,820	14,309	13,729	13,197	13,197	14,068							126,352
waste:ore strip ratio cy/lb																				
Reclamation														15,000	15,000					30,000
Tons Mined					1,029	933	700	606	641	622	691	911	1,039							7,172
Plant feed, % U3O8					0.065	0.076	0.072	0.073	0.087	0.063	0.077	0.055	0.048							0.067
Pounds Contained U3O8					1,340	1,411	1,006	887	1,120	784	1,067	1,004	994							9,613
Highwall Miner																				
Tons Mined								282	282	188	188		169	169	169	169				1,615
Plant feed, % U3O8								0.086	0.086	0.086	0.086		0.070	0.070	0.070	0.070				0.079
Pounds Contained U3O8								482	482	321	321		238	238	238	238				2,558
Underground																				
Tons Mined										200	400	400	400	400	400	400	400	401		3,401
Plant feed, % U3O8										0.088	0.088	0.088	0.088	0.088	0.088	0.088	0.088	0.088		0.088
Pounds Contained U3O8										351	702	702	702	702	702	702	702	704		5,970
Tons Mined				$ 112	1,029	933	700	888	922	1,010	1,279	1,311	1,608	569	569	569	400	401		12,300
Plant feed, % U3O8				0.095	0.065	0.076	0.072	0.077	0.087	0.072	0.082	0.065	0.060	0.083	0.083	0.083	0.088	0.088		0.075
Pounds contained U3O8				$ 211	1,340	1,411	1,006	1,369	1,602	1,456	2,091	1,706	1,934	940	940	940	702	704		18,353
Tons Stockpiled					179	212	12	(1)	22	32	311	621	1,230	799	368	337	137			
`				0.095	0.065	0.076	0.072	0.077	0.087	0.072	0.082	0.065	0.060	0.083	0.083	0.083	0.088	0.088		
Pounds stockpiled					233	320	17	(1)	38	46	508	809	1,479	1,320	608	557	240			
Tons Processed				$ 112	850	900	900	900	900	1,000	1,000	1,000	1,000	1,000	1,000	600	600	538		12,300
Plant feed, % U3O8				0.095	0.065	0.074	0.073	0.077	0.087	0.072	0.081	0.070	0.063	0.055	0.083	0.083	0.085			0.075
Pounds contained U3O8				$ 211	1,107	1,324	1,309	1,387	1,564	1,448	1,629	1,405	1,264	1,099	1,652	991	1,018	945		18,353
Pounds recovered U3O8				$ 190	996	1,191	1,178	1,248	1,408	1,303	1,466	1,265	1,138	989	1,487	892	917	850		16,518
Recovery %				0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90		0.900
U3O8 price/pound				65	$ 65	$ 65	$ 65	$ 65	$ 65	$ 65	$ 65	$ 65	$ 65	$ 65	$ 65	$ 65	$ 65	$ 65		
GROSS REVENUES				$ 12,368	$ 64,739	$ 77,427	$ 76,578	$ 81,135	$ 91,496	$ 84,698	$ 95,279	$ 82,211	$ 73,959	$ 64,301	$ 96,647	$ 57,988	$ 59,577	$ 55,269		$ 1,073,670
Direct Costs:																				
Pit Stripping Annual Direct				$ 5,299	$ 10,599	$ 10,599	$ 10,599	$ 10,599	$ 10,599	$ 10,599	$ 10,599	$ 10,599	$ 10,599							$ 100,690
Pit Mining Annual Direct				$ 558	$ 2,231	$ 2,231	$ 2,231	$ 2,231	$ 2,231	$ 2,231	$ 2,231	$ 2,231	$ 2,231							$ 20,640
HW Mining $/ton								$ 3,240	$ 3,240	$ 2,160	$ 2,160		$ 1,944	$ 1,944	$ 1,944	$ 1,944				$ 18,573
UG Mining $/ton										$ 6,823	$ 13,646	$ 13,646	$ 13,646	$ 13,646	$ 13,646	$ 13,646	$ 13,646	$ 13,689		$ 116,034
Mineral Processing $/ton				$ 1,189	$ 9,035	$ 9,566	$ 9,566	$ 9,566	$ 9,566	$ 10,629	$ 10,629	$ 10,629	$ 10,629	$ 10,629	$ 10,629	$ 6,378	$ 6,378	$ 5,721		$ 130,741
Staff Annual				$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 1,651	$ 826	$ 25,598
Reclamation Mine														$ 10,599	$ 10,599					$ 21,198
Reclamation Mill/Tailings																		$ 10,000	$ 10,000	$ 20,000
Resin Shipping and Processing $/lb				$ 1,010	$ 5,289	$ 6,325	$ 6,256	$ 6,628	$ 7,474	$ 6,919	$ 7,784	$ 6,716	$ 6,042	$ 5,253	$ 7,895	$ 4,737	$ 4,867	$ 4,515		$ 87,711
Reclamation Bond 2% of $20,000,000 per year				$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400		$ 6,000
Severance tax (2.5% on 50% of net)				$ 67	$ 536	$ 688	$ 677	$ 694	$ 823	$ 653	$ 700	$ 564	$ 436	$ 476	$ 880	$ 450	$ 494	$ 448		$ 8,587
TOTAL Direct Costs				$ 10,175	$ 29,741	$ 31,461	$ 31,381	$ 35,010	$ 35,986	$ 42,066	$ 49,801	$ 46,437	$ 47,579	$ 44,598	$ 47,645	$ 29,206	$ 27,437	$ 36,425	$ 10,826	$ 555,773
Cash Flow Pre-tax				$ 2,193	$ 34,998	$ 45,966	$ 45,197	$ 46,125	$ 55,510	$ 42,631	$ 45,478	$ 35,774	$ 26,380	$ 19,702	$ 49,002	$ 28,782	$ 32,140	$ 18,844	$ (10,826)	$ 517,898
Capital Expenditures:																				
Permitting		$ 1,500	$ 2,000	$ 500																$ 4,000
Development		$ 2,000	$ 2,000																	$ 4,000
Access Road			$ 1,040																	$ 1,040
Buildings, Shop, Support Equip				$ 2,544																$ 2,544
Open Pit Mine				$ 10,714		$ 5,600														$ 16,314
Open Pit Mine contingency 15%				$ 1,989																$ 1,989
HW Miner Equipment w/ 25%								$ 7,813	$ 7,813											$ 15,625
UG Room & Pillar w/ 25%										$ 18,750	$ 18,750									$ 37,500
Processing Plant				$ 10,347																$ 10,347
Heap Pad/Equipment				$ 7,656			$ 5,100			$ 5,100			$ 5,100			$ 5,100				$ 28,056
IX Resin First Fill				$ 3,203																$ 3,203
Indirects (EPM,Freight, Owner Costs)				$ 1,267																$ 1,267
Working Capital				$ 2,111															$ (2,111)	
Processng Plant Contingency 25%				$ 2,990																$ 2,990
Replacement Mine @5%							$ 536	$ 536	$ 816	$ 816	$ 816	$ 816	$ 816							$ 5,150
Replacement Plant @5%							$ 517	$ 517	$ 517	$ 517	$ 517	$ 517	$ 517	$ 517	$ 517	$ 517				$ 5,173
TOTAL CAPITAL EXPENDITURES		$ 3,500	$ 5,040	$ 43,320		$ 5,600	$ 6,153	$ 8,866	$ 9,146	$ 25,183	$ 20,083	$ 1,333	$ 6,433	$ 517	$ 517	$ 5,617			$ (2,111)	$ 139,198
NET CASH FLOW PRE-TAX		$ (3,500)	$ (5,040)	$ (41,127)	$ 34,998	$ 40,366	$ 39,044	$ 37,260	$ 46,365	$ 17,448	$ 25,395	$ 34,441	$ 19,947	$ 19,185	$ 48,484	$ 23,165	$ 32,140	$ 18,844	$ (8,715)	$ 378,699
CUMULATIVE NET CASH FLOW:		$ (3,500)	$ (8,540)	$ (49,667)	$ (14,669)	$ 25,696	$ 64,740	$ 102,000	$ 148,365	$ 165,813	$ 191,208	$ 225,649	$ 245,596	$ 264,781	$ 313,265	$ 336,430	$ 368,570	$ 387,415	$ 378,699	

23 ADJACENT PROPERTIES

There are no adjacent properties according to NI 43-101 definitions.

24 OTHER RELEVANT DATA AND INFORMATION

This section is not applicable.

25 INTERPRETATION AND CONCLUSIONS

Based on the recent assembly and verification of data by UEC on the Anderson Uranium Project, the following conclusions can be made:

Mineral Resources:

- The level of understanding of the geology at Anderson Project is relatively good.
- The practices used during the various drilling campaigns were conducted in a professional manner and adhered to accepted industry standards.
- There are no evident factors that would lead one to question the integrity of the database.
- There are no unusual risks associated with the resource estimates.
- A significant uranium deposit was outlined. Mineralization is hosted in lacustrine facies fixed by the presence of carbonaceous material.
- Drilling to date has outlined an Indicated open pit resource (at a 0.01% eU_3O_8 cut-off) of 25.4 Mtonnes (28.0 M tons) at 0.028% eU_3O_8 which contains 15.5 million pounds of uranium and an Inferred resource (at a 0.01% eU_3O_8 cut-off) of 4.6 Mtonnes (5.1 M tons) at 0.024% eU_3O_8 which contains 2.5 million pounds of uranium.
- The underground Indicated resource (at a 0.035% eU_3O_8 cut-off) is 1.4 Mtonnes (1,6 M tons) at 0.049% eU_3O_8 which contains 1.5 million pounds of uranium and an Inferred resource (at a 0.035% eU_3O_8 cut-off) of 8.4 Mtonnes (9.2 M tons) at 0.052% eU_3O_8 which contains 9.5 million pounds of uranium.

Preliminary Economic Assessment:

- Conceptual mine plans were developed for a conventional mine operation which includes open pit, highwall, and underground mining. Portions of the current mineral resource, both indicated and inferred mineral resources, were included within the conceptual mine designs for the PEA. The indicated and inferred mineral resources used in the PEA are fully included in the total Indicated and Inferred mineral resources reported in Section 14 of this report. They are that portion of the mineral resources which meet minimum cutoff criterion and are incorporated within conceptual mine designs and represent approximately 80% of the mineral resources as stated in Section 14 of the report/
- Conceptual plans were developed for the processing of the mined material via conventional heap leach methods using an acid lixiviant.
- Recent metallurgical testing of mineralized material from on-site stockpiles was completed which indicates greater than 90% recovery with an average acid consumption of 50 pounds per ton of material processed.
- The base case for the Preliminary Economic Assessment (PEA) considers conventional mining in conjunction with on-site heap leach recovery, producing an intermediate uranium concentrate in the form of loaded resin which would be shipped to EFR's White

Mesa mill in Utah for final processing. However, once the uranium is concentrated and loaded on resin it could be shipped to other central processing facilities.

- Vanadium is present in the mineralized material. The PEA is based on the recovery of uranium only. Future studies will determine the feasibility of recovering vanadium as a by-product.
- CAPEX for the project is estimated at 8 million $US for pre-production costs, 43.9 million $US for initial capital, and 87.3 million $US for additional capital during operations for a total life-of-mine capital of 139.2 million $US.
- OPEX is estimated at approximately $45/ton or $34/lb U_3O_8 recovered including all operating and reclamation costs exclusive of income tax.
- The current The PEA shows a positive return on investment. Table 22.1 shows the IRR and NPV at various discount rates both before and after taxes.
- This is a restricted disclosure as allowed under section 2.3(3) of NI 43-101 which includes a Preliminary Economic Sssessment (PEA). It is also preliminary in nature such that it includes portions of both indicated and inferred mineral resources, as reported in Section 14 of the report. The PEA is based on open pit mining and heap leach extraction of uranium values, utilizing methodologies, equipment, and a generalized design criterion which has been employed at the site and/or similar sites in the past but has not been specifically developed for the Project. Mineral resources are not mineral reserves and do not have demonstrated economic viability in accordance with CIM standards. Inferred mineral resources are too speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the outcomes estimated in the PEA will be realized.

26 RECOMMENDATIONS

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

- Additional drilling to expand confirmation results from historic drilling in both the open pit and underground portions of the deposit. The use of both PFN and chemical assay should be used for the confirmation of grade. A budget of US$780,000 has been proposed to complete this test work (Table 26.1).
- Additional metallurgical testing on both open pit and underground areas. A budget of US$100,000 has been proposed to complete this work (Table 26.1).
- After drilling is completed, an updated resource estimate should be prepared. A budget of US$75,000 has been proposed to complete this work (Table 26.1).
- Environmental studies to provide a baseline for future exploration and possible development work on the project. A budget of US$675,000 has been proposed to complete this work (Table 26.2).

The recommended drilling and assaying will attempt confirm historic results and upgrade the classification of resources in some areas. The Prompt Fission Neutron (PFN) logging 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 26.1: EXPLORATION BUDGET

Item	Cost (USD)
Permitting and reclamation	$25,000
5 diamond drill holes (100 m average, 500 m total)	$150,000
20 RC holes (200 m average 4,000 m total)	$400,000
PFN probing 25 holes	$125,000
Assay of core and RC chips (2,000 samples by ICP-MS)	$88,000
Metallurgical heap leach testing	$100,000
Resource model update and report	$75,000
Road maintenance	$25,000
Exploration TOTAL	**$988,000**
Rounded Use	**$1,000,000**

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

TABLE 26.2: 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 26.3: 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 26.4: ENVIRONMENTAL BASELINE AND RELATED STUDIES

Item	Cost (USD)
BLM Plan of Operations and Environment Impact Statement (Mine)	$1000,000
State and Local Mine and Related Permitting	$500,000
U. S. NRC Licensing and Environmental Impact Statement (Mill)	$1500,000
Environmental Baseline TOTAL	**$3,000,000**

The recommendations outlined in Tables 26.3 and 26.4 related to final design and development and permitting and licensing would need to be implemented on concurrent work schedule.

27 REFERENCES

Arseneau, G. (2011), "Anderson Uranium Project, Yavapai County, Arizona, USA." Report prepared for Global Uranium Corporation by SRK Consulting (Canada) Inc., May.

Chapman, Wood & Griswold Inc. (1977), "A Review of Estimated Mineable Uranium Reserves, Anderson Mine Project, Yavapai County, Arizona," September.

COLEDT (1978), Minerals Exploration Company Exploration Drill Holes Coordinate Database Printout.

Gilbride, P.E., Hardy, M.P., and Hambley, D. (2010), "Mineral Resources of the Anderson Uranium Property Yavapai County, Arizona, USA" Report prepared for Global Uranium Corporation by Agapito Associates, Inc., July 01.

Harris and Thompson (2007), "Updated title opinion on Anderson mine project, Yavapai County, Arizona," Richard W. Harris letter to Ralph W. Kettel, June 26, 14pp.

Hazen Research, Inc. (1977), "Alkaline Leaching of Anderson Mine Samples for Minerals Exploration Company," August.

Hazen Research, Inc. (1978a), "Mode of Occurrence of Uranium in Core Samples From Date Creek Property, Arizona," February.

Hazen Research, Inc. (1978b), "Uranium Recovery From Anderson Mine Ore- Laboratory Studies," March.

Hazen Research, Inc. (1978c), "Mode of Occurrence of Uranium in Core Sample DC-77C from Date Creek Property, Arizona," October.

Hazen Research, Inc. (1979), "Analytical Results of Date Creek Drill Cores," February.

Hertzke, D. C. (1997), "Summary Report on the Uranium Potential of the Anderson Uranium Property, Yavapai County, Arizona," July 28.

Lucht, Robert F. (1978), "Development Report on Equilibrium at the Anderson Mine Project," April 28.

Minerals Exploration Company of Union 76 Minerals (1978a), "Feasibility Study, Anderson Project," Volume 1, Executive Summary, July.

Minerals Exploration Company of Union 76 Minerals (1978b), "Feasibility Study, Anderson Project," Volume 2, Geology and Mining, July.

Minerals Exploration Company of Union 76 Minerals (1978c), "Feasibility Study, Anderson Project," Volume 3, Ore Processing, July.

Roberts, John C. (1978) "Anderson Mine Bulk Density Study," Minerals Exploration Company of Union 76 Minerals, January.

Scott, James H. (1962), "The GAMLOG Computer Program," Report RME-143, US Atomic Energy Commission, Grand Junction, CO, 43pp.

Sherborne, J.E., Jr. (1979), "Major Uranium Discovery in Volcaniclastic Sediments, Basin and Range Province, Yavapai County, Arizona." A.A.P.G. *Bulletin*, Vol. 63, No. 4, pp. 621-646.

SME Mining Engineering Handbook, Third Edition, Chapter 13.2, Room and Pillar Mining of Coal.

Urangesellschaft USA, Inc. (1979a), "Date Creek Project Preliminary Evaluation for Prospective Underground Mine," September.

Urangesellschaft USA, Inc. (1979b), "1978 Annual Report, Date Creek Project," February.

Urangesellschaft USA, Inc. (1979c), "Mineable Ore Reserves by Polygon. Underground Mining Areas. Cutoff: 6' min. mining width 0.05% eU_3O_8," August 8.

Urangesellschaft USA, Inc. (1980), "1979 Date Creek Project, Annual Report," January.

Water Development Corporation (1978), "Ground Water Hydrology of Anderson Mine Area, Arizona," D.K. Greene, L.C. Halpenny, and S.D. Clark authors, February.

Water Development Corporation, (1979), "Hydrogeology of Anderson Mine Area, Arizona," D.K. Greene, S.D. Clark, and L.C. Halpenny authors, August.

WEB SITES REFERENCED

Cameco 2014; http://www.cameco.com/investors/uranium_prices_and_spot_price/
Outsider Club, 2013; http://www.outsiderclub.com/uranium-investing-in-2014/606
Trade Tech, 2013; http://www.uranium.info/
World Nuclear Association, 2013; http://world-nuclear.org/info/Nuclear-Fuel-Cycle/Uranium-Resources/Uranium-Markets/

28 CERTIFICATE AND SIGNATURES OF QUALIFIED PERSONS

SIGNATURE PAGE AND CERTIFICATE of AUTHOR

I, Bruce Davis, FAusIMM, do hereby certify that:

1. I am an independent consultant of BD Resource Consulting, Inc., located at 4253 Cheyenne Drive, Larkspur, CO, U.S.A., 80118, incorporated January 18, 2008.

2. I graduated with a Doctor of Philosophy degree from the University of Wyoming in 1978.

3. I am a fellow of the Australasian Institute of Mining and Metallurgy, Registration Number 2111185.

4. I have practiced my profession continuously for 33 years and have been involved in geostatistical studies, mineral resource and reserve estimations and feasibility studies on numerous underground and open pit base metal and gold deposits in Canada, the United States, Central and South America, Europe, Asia, Africa and Australia.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for Sections 4, 5, 6, 7, 8, 9, 10,11,and 12, and co-authored and am responsible for Sections 1, 2, 3, 13, 23, 24, 25, 26, and 27 of the Technical Report titled "Technical Report and PEA on the Anderson Uranium Project Yavapai County, Arizona" dated July 31, 2014, with an effective date of July 6, 2014 (the "Technical Report"). I personally visited the site May 8, 2012.

7. I have had no prior involvement with the property that is the subject of the Technical Report.

8. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to make the Technical Report not misleading.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report

Dated this 31st day of July , 2014.

Bruce M. Davis

____

Bruce M. Davis, FAusIMM

ROBERT SIM, P.Geo.

SIM Geological Inc.

6810 Cedarbrook Place, Delta, British Columbia, Canada V4E 3C5
Telephone: 604-596-6339 Fax: 604-596-6367
Email: rsim@dccnet.com

SIGNATURE PAGE AND CERTIFICATE OF QUALIFIED PERSON

I, Robert Sim, P.Geo., do hereby certify that:

1. I am an independent consultant SIM Geological Inc.

2. I graduated from Lakehead University with an Honours Bachelor of Science (Geology) in 1984.

3. I am a member, in good standing, of the Association of Professional Engineers and Geoscientists of British Columbia, License Number 24076.

4. I have practiced my profession continuously for 28 years and have been involved in mineral exploration, mine site geology and operations, mineral resource and reserve estimations and feasibility studies on numerous underground and open pit base metal and gold deposits in Canada, the United States, Central and South America, Europe, Asia, Africa and Australia.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of section 14 of the technical report titled "Technical Report and PEA on the Anderson Uranium Project, Yavapai County, Arizona, USA" dated July 31, 2014, with an effective date of April 15, 2012.

7. I have not visited the property.

8. I have not had any prior involvement with the property that is the subject of the Technical Report.

9. As of as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 31st day of July, 2014.

"original signed"

____

Robert Sim, P.Geo.

SIGANTURE PAGE AND CERTIFICATE OF QUALIFIED PERSON

DOUGLAS L. BEAHM

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 co-author of the Technical Report titled "Technical Report and PEA on the Anderson Uranium Project Yavapai County, Arizona" dated July 31, 2014, with an effective date of July 6, 2014 (the "Technical Report").
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; and Registered Member of the SME.
4. I have worked as an engineer and a geologist for over 40 years. My work experience includes: uranium exploration, mine production, and mine/mill decommissioning and reclamation and have completed more than twenty technical reports on uranium projects since 2005.
5. I was last present at the site on December 17 and 18, 2013.
6. I am responsible for Sections 15, 16, 18, 19, 20, 21, and 22, and co-authored and am responsible for Sections 1, 2, 3, 13, 23, 24, 25, 26, and 27.
7. I am independent of the issuer as described in section 1.5 of NI 43-101.
8. I do not have prior working experience on the property.
9. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of my education, professional registration, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with same.
11. As of the date of this report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority.

_July 31, 2014

Signed and Sealed

Douglas L. Beahm

SIGNATURE PAGE AND CERTIFICATE OF QUALIFIED PERSON

TERENCE P. McNULTY
I, Terence P McNulty, D. Sc., P.E., do hereby certify that:

1. I am the co-owner and President of T. P. McNulty and Associates, Inc. located at 4550 North Territory Place, Tucson, AZ 58750-1855
2. I am co-author of the Technical Report titled "Technical Report and PEA on the Anderson Uranium Project Yavapai County, Arizona" dated July 31, 2012, with an effective date of July 6, 2014 (the "Technical Report").
3. I obtained with a Bachelor of Science degree in Chemical Engineering from Stanford University in 1961, a Master of Science degree in Metallurgical Engineering from Montana School of Mines in 1963, and a Doctor of Science degree from Colorado School of Mines in 1966. I am a Registered Professional Engineer in the State of Colorado (License # 24789) and a Registered Member (# 2,152,450RM) of the Society of Mining, Metallurgy, and Exploration, Inc.
4. I have worked as a metallurgical engineer for a total of 51 years, including years worked between degrees. My recent experience for the purpose of the Study is as follows:
   a. I have worked as a consultant on 27 uranium projects during the last 8 years and have contributed to NI 43-101 compliant studies for many of those;
   b. I was Manager of Corporate R&D and Technical Services for a large diversified mining firm, The Anaconda Company, which was a major uranium producer.
5. I last visited the site on December 18, 2013.
6. I am responsible for Section 17 and co-authored and am responsible for Section13 of this report.
7. I am independent of the issuer as described in section 1.5 of NI 43-101.
8. I do not have prior work experience on the property.
9. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of my education, professional registration, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with same.
11. As of the date of this report, to the best of my knowledge, information and belief, the part of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority.

July 31, 2014
Signed and Sealed

Terence P. McNulty